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OF SOUTH AFRICA.

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THE JOURNAL
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VOL. XI. JULY, 1910. No. 1.

Proceedings
AT
Ordinary General Meeting, July 16, 1910.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, July 16th. At the outset Mr. A. McArthur Johnston presided, and after confirmation of the minutes of the previous meeting, Dr. J. Moir, the new President, took the chair.

There were also present:—

44 Members: Messrs. C. B. Saner, F. F. Alexander, E. H. Croghan, K. L. Graham, Tom Johnson, A. Richardson, A. Whitby, H. A. White, J. Littlejohn, R. G. Bevington, A. F. Crosse, W. A. Caldecott, E. H. Johnson, A. C. Adams, H. A. Adams, S. Beaton, W. M. Butler, D. Coll, W. M Coulter, J. Gray, W. H. Jane, J. H. Johnson, G. A Lawson, H. Lea, J. Lea, P. Morrisby, S. Newton, F. B. Ogle, E. Pam, F. D. Phillips, J. W. Philpott, H. S. Potter, O. P. Powell, J. F. Pyles, T. F. Smith, J. A. Smith, A. R. Stacpoole, H. Taylor, J. A. Taylor, A. Thomas, A. Treloar, J. Watson, and A. Wilkinson.

15 Associates and Students: Messrs. A. R. Adams, H. Brazier, C. J. Crocker, C. E. Deakin, C. E. Hawkins, W. J. R Hunter, A. King, C. J, McCaffrey, R. Murdoch, G. Musson, D. Nicholas. H. Olivier, H. B. Powter, J. Thorlund, and C. K. White.

20 Visitors, and Fred. Rowland, Secretary.

NEW MEMBERS.

Messrs. E. H. Croghan and A. S. Gillies were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows:—

ADAM, H. R., B.Sc.(Aberdeen), c/o Mr. A. Heymann, P. O. Box 3427, Johannesburg. Analytical Chemist.

CHALKLEY, GUY AUBREY, Knights Deep, Ltd., P. O. Box 143, Germiston. Mine Manager.

DIXON, JOHN MCKAY, Simmer and Jack Proprietary Mines, Ltd , P. O. Box 192, Germiston. Mill Foreman. *(Transfer from Associate Roll.)*

FREEDMAN, H., 13, Third Street, Boksburg North. Surveyor.

HAINE, W. E., Simmer and Jack Proprietary Mines, Ltd , P. O. Box 192, Germiston. Assistant Mill Foreman.

JACKSON, WILLIAM, Leyland and Birmingham Rubber Co., Ltd., P. O. Box 4856, Johannesburg. Manager.

MEYER, EDWARD CARL JOACHIM, East Rand Proprietary Mines, P. O. Box 119, East Rand. Mining Engineer.

STEPHENS, CHARLES T., Carlton Hotel, Johannesburg.

STUART, ANDREW NEILSON, Simmer Deep, Ltd., P. O. Box 178, Germiston. Slime Foreman.

THOMAS, CHARLES FREDERICK, M.I.M.M., P. O. Box 3520, Johannesburg. Mining Engineer.

The Secretary: Since the last meeting of the Society the following have been admitted by the Council:—

A.: Associates.—

BEATTY, JAMES HENRY, Sub-Nigel, Ltd., Nigel. Shift Boss.

BRANFORD, ERIC WOMACK, Lonely Mine, Inyati P. O., Rhodesia. Cyanider and Assayer.

CANNING, ADAM BIRCH, Anglo-French Exploration Co., Ltd.. P. O. Box 2927, Johannesburg. Draughtsman.

CARBIS, ALEXANDER ROBERT, Sheba G. M. Co., Ltd., Eureka. Cyanider.

CARBIS, FREDERICK CHARLES, New Kleinfontein Co., Ltd., P. O. Box 129, Benoni. Shift Boss.

DEGENHARDT, WILLIAM RUSSELL, Messrs. Bewick Moreing & Co., P. O. Box 120, Kalgoorlie, W. Australia. Engineer.

DUNCUM, ERIC CUNDY, Geldenhuis Deep, Ltd. (West), P. O. Box 1, Cleveland. Cyanider.

GATES, ALFRED ATKINSON, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Amalgamator.

KOBUSE, JUNJIRO, Kosaka Mine, Akita-Ken, Japan Engineer of Mining and Metallurgy.

MAASDORP, HENRY ERNEST, City Deep, Ltd., P. O. Box 1411, Johannesburg. Mine Surveyor.

SEDGWICK, ARNOLD WILLIAM NITICHEN, Simmer and Jack Proprietary Mines, Ltd.. P. O. Box 192, Germiston. Cyanider.

WEBSTER, JOHN CECIL, Geldenhuis Deep, Ltd. (West), P. O. Box 1, Cleveland. Learner.

Williams, Gwilym Watkin, Geldenhuis Deep, Ltd., P. O. Box 3404, Johannesburg. Cyanide Shift Leader.

As Students.—

Cooper, Frank Hugh, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Mill Learner.

Damant, Cecil Atherstone, S.A. School of Mines and Technology, P. O. Box 1176, Johannesburg. Student

Doble, Frank Edward, S.A. School of Mines and Technology, P. O. Box 1176, Johannesburg. Student.

Grace, James Sidney, Geldenhuis Deep, Ltd. (West), P. O. Box 1, Cleveland. Mill Learner.

Heydenrych, Percy Walter, Geldenhuis Deep, Ltd. (West), P. O. Box 1, Cleveland. Cyanide Learner.

Metelerkamp, Lawrence, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Mill Learner.

Nellmapius, Ernest Harold, S.A. School of Mines and Technology, P. O. Box 1176, Johannesburg. Student.

Roberts, Austin, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanide Learner.

Sharp, William, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Mill Learner.

White, Cecil Kennedy, Mines Department Chemical Laboratory, Plein Street. Johannesburg. Chemistry Learner.

General Business.

The President : You will have noticed that the form of the agenda has been rearranged. That arises through the Council having adopted a scheme for simplifying the business of the Society by giving precedence at successive meetings to chemical, mining and metallurgical papers respectively. We are doing it as an experiment, of course, to see how it will work, and in the hope that it will make our meetings more efficient. I have to point out that a scheme of this sort calls for your co-operation. Previously papers have been available for discussion during four meetings, and the result of this new scheme is that papers will be given great prominence at one meeting and afterwards very little prominence, so that it is advisable for you to have your discussions ready for the particular month a paper is prominent, as for instance, mining papers for next month and metallurgical papers for September Of course I do not want to cause any misunderstanding : I mean the other papers will not be passed without discussion, but it will be at the end of the meeting when there may not be so much time to spare.

Mr. Andrew F. Crosse (*Past President*): I should now like to make a few remarks on the arrangements made, regarding the reading of the papers at our monthly meetings. There are only twelve meetings annually, and we have a larger number of papers to discuss than we had a few years ago, consequently our meetings are somewhat too prolonged. I should like to ask the Council if they could devise some scheme whereby the meetings could be shortened, and we might get through the discussions rather more quickly.

Might we not follow the example of the Institution of Mining and Metallurgy in London, where the papers are printed and sent out to members, so that the said papers may be carefullly considered beforehand? The author of the paper merely makes a few remarks—the members have already read over the paper and the discussion can begin at once—we waste too much time listening to the papers being read—it is impossible to digest the facts by simply listening to the authors reading the papers. If we could receive and read the paper over, three or four days before the meeting is held, a great deal of time would be saved on the night of discussion. Again, would it not be better also for new members to be elected by the Council? This would also economise time.

We should seek, in every way, to save time in the same way as we economise the materials we handle.

The President : I think we are in sympathy with any scheme for making the Society more efficient.

Mr. R. G. Bevington (*Past-President*): As regards the election of members being left to the Council, that is absolutely against our Constitution. The Constitution provides that members must be elected by ballot and by the members present at a general meeting. Therefore any suggested deviation from this procedure necessitates calling a special meeting to alter the Constitution. With regard to the printing of papers before they are read before the Society, I think they certainly might be printed and circulated, but I also think that the *reading* of the paper by the author impresses the subject upon those who are listening very much more than if they read the papers themselves in cold type. There is no reason why papers, if they are accepted by the Council, should not be printed and circulated amongst the members, but I think the papers should be *read*, and I do not think that the time taken to read them is wasted. It gives a paper very much more interest and emphasis if it be read and explained by its author.

Mr. A. Whitby (*Member of Council*): It appears to me that the whole thing rests upon this: Authors and critics should condense their matter as much as they possibly can. As a fact they do not do so, and have never done so during my connection with the Society.

The President: I think Mr. Whitby's remarks contain the essence of the thing. When authors read their papers there is no need to read them *in extenso* as printed, but only the most striking matter. We shall bring this before the Council and do our best to pay attention to it.

Mr. J. H. Johnson (*Member*): Would it be possible for contributions to discussions to be printed and distributed amongst the members in the same way as the original papers are?

The President: You mean articles sent in in advance of the meetings?

Mr. J. H. Johnson: Yes.

The President: I understand the Secretary does his best to oblige interested members in this way already.

INAUGURAL ADDRESS.

By James Moir, D.Sc., M.A., F.C.S. (President).

I have had some difficulty in choosing a subject for an address, chiefly for the unusual reason of an excess of material: and this combined with my own somewhat discursive temperament compels me to avoid the customary sustained essay on one particular subject, and to talk on things in general. This is perhaps just as well, because for one thing I look on essay-writing as an unpleasant reminiscence of school days, and really feel that unless one happens to possess a sort of archangelical gift of speech, the sustained essay is apt to be tiresome—at all events, it almost invariably contains about 50% of platitude.

I have been a member of the Society for eight years, and have worked on its Council most of the time. Although I have been *opsitting* for the Presidentship (as our Dutch friends would say) for four years, I regret that I have had next to no practice in the Chair, owing to the extreme conscientiousness and devotion to duty of my predecessors while I was Vice-President. I am also, I believe, the youngest of those who have reached the Chair, and you will probably excuse it as youthful egotism if I point out another fact, namely, that you have had an abnormal number of Scots Presidents — something near 50% since the war. This may possibly have something to do with the *praefervidum ingenium Scotorum*, but more probably is an instance of the meek inheriting the earth!

I propose to commence my Address with a few remarks, as little platitudinous as possible, on the nature of the scientific spirit. I take it that you will accept me as being representative more of the scientific than of the technical side of your multifarious activities.

In the first place, the scientific man sooner or later finds out that he must always keep his mind in suspense in every branch of knowledge where advance is possible: he must always be expecting exceptions and abnormalities, the "ugly facts" spoken of by Huxley which can "slay a cherished hypothesis." Science has gone far in the last 50 or 60 years: in fact I am in the habit of thinking that the "dark ages" lasted pretty nearly to 1850; that is, that all the pre-scientific ages were "dark ages," for really, up to 1850 no one could honestly claim that the civilisation of the Greeks had been surpassed—whereas now one can merely say that it is nowhere in comparison. Anyway, I think that the advance of science in these sixty years is not abnormal, and that there is no fear of a relapse except through war: science must continue to grow, and to grow in geometrical progression faster and faster, until either the whole population is scientific, or the whole of the scientists are confined in asylums!

For the present and immediate future science has two dangers to face: (1) over-specialisation, *i.e.*, too wide divergence of the sciences, and (2) dogmatism and blind worship of hypothesis. I shall deal with the latter first. Dogmatism is the opposite of the scientific spirit, and the negation of the suspended judgment. It is already far too common in Germany, for example, but in other countries seems to be the special preserve of the "practical man," who allows his preconceived notions to distort his view even of facts; and when he gets involved in argument, to adopt a somewhat papal attitude towards his opponents. This is true, I am sorry to say, of many medical men as well as of technicals; in scientific matters many of them are just excellent lawyers spoilt. One would expect them to know that intellects differ and that now-a-days it is given to no one to dictate the direction in which knowledge shall advance.

I see I am now exemplifying in my own person another characteristic of the scientific man, who is frequently getting to be a most unpleasant being from his tendency to possess an ideal and to criticise everything from its standpoint[*] whereas, as everybody knows, many of the ordinary affairs of life (like politics), and certain phases of personal ambition, do not bear this critical scrutiny at all well. On the other hand, the scientist, both from his wider knowledge and from his receptivity of mind, is more sympathetic and generally takes a wider view of most questions than the purely literary or technical man—he can nearly always see the other fellow's point of view, though perhaps through critical spectacles.

In any case, the higher scientific pioneer—the man who may imagine some conception new to the human mind, must have the power to abandon at a moment's notice, at the behest of fact, every trammel of early education, convention, or accepted belief: but the fact and the evidence must be clear. While I am on this subject I can not resist quoting a remark of Dr. Wiley (U.S. Agric. Bureau) though it should make us blush—"The men who have been great lights in the scientific world have been men of unimpeachable morality . . . . who could not possibly do a wrong thing knowingly." I fear we do not all come up to this standard: in fact there are signs in this country of the growth of a scientific evil which seems to have originated, I fear, in Dr Wiley's country—that of authors using their assistants' work without sufficient acknowledgment of their indebtedness. No scientific man who values the honour of his profession would commit this serious breach of professional ethics. As usual, Sir W. S. Gilbert has something apposite: it is in the Lord Chancellor's song in "Iolanthe."

---

[*] "The habit of critically [illegible] the [illegible] of things by a theoretic standpoint." —*[illegible]*

"My learned profession I'll never disgrace
By taking a fee with a grin on my face
When I've never been there to attend to the case."

I have already mentioned the danger of over-specialisation in science; no human intellect can possibly cope with all its branches--not even the superficially omniscient brain of Herbert Spencer could deal with the whole of the last generation's Science: and now the only hope is to correlate and help to re-unite a few of its more closely connected branches. In the meantime the collection and classification of results must go on, even when apparently quite useless from the practical man's point of view, and solely in the hope that a genius will arise to build them into the edifice of knowledge. The members of this Society for example have many very valuable data lost and hidden from view in notebooks, and I call on them in the name of science to read short papers to put these data on record. It may be only a few months before they become practically useful. For example, for stope filling we want a complete set of "*total* cyanide" values in last-drainings over the Rand, and at present scarcely anything except so-called "available cyanide" is known. Let them not fear criticism: criticism at worst is only an example of what one of our British Association visitors kindly called "the atmosphere of intellectual vigour noticeable in this Society." It may also introduce them to the self-discipline of Science—which makes people think for themselves and distrust authorities and text-books. This young and vigorous country is quite safe in being allowed to think for itself, though there are other countries that we wot of where, as Gilbert says:

"The prospect of a lot,
Of dull M.P.'s in close proximity,
All thinking for themselves is what,
No chap can face with equanimity."

I hope my friend Prof. Young will pardon me if I cite his science of Geology as giving a case in point of my charge of over-specialisation. The accusation I have to make is, briefly stated, that all sorts of geological authorities accept the belief that water can be made red hot and yet preserve its solvent properties. Now every chemist and physicist knows that above the critical point water can only exist as steam, entirely devoid of solvent powers[*] except for other vapours; and this however high the pressure. This critical temperature is quite low, about the boiling point of mercury in fact, and as rock magmas are certainly not vapours, there is no possibility of anything except an uncombined emulsion of rock and steam. I referred to this a couple of years ago,[†] but the seed fell on stony ground: I shall now try and make it germinate by saying that until Geology becomes an experimental science, it will continue to be almost as incoherent and purely disputatious as Theology is.

I now come to the kernel of my address—a *resumé* of the newest scientific ideas, mainly in connection with chemistry and related branches of other sciences.

The first two points deal with ionisation, and serve to illustrate my contention that even the most beautiful and simple of hypotheses must go under when in conflict with ascertained fact, and that nevertheless the original hypothesis may continue to be held with what I have termed papal tenacity in the land of its development, Germany. Ostwald originally showed that the dissociation of weak electrolytes strictly follows the law of mass-action, thus in the case of acetic acid the equation $H^{\bullet} + \overline{A}' = H\overline{A}$ leads to the algebraical equations $Kx^2 = C - x$ where x is the dissociated part and C the total concentration of acetic acid, or $KC = \frac{1-\gamma}{\gamma^2}$ if γ is the co-efficient of ionisation, *i.e.*, $x = C\gamma$. It was soon found not to be true of good conductors, and Storch discovered an *empirical* law which expresses the behaviour of the latter very well: this is $K^2C = \frac{(1-\gamma)^2}{\gamma^3}$. The new discoveries, mostly from America, give the unexpected result that Storch's formula is pretty nearly true, not only for substances like salt and saltpetre, but also for what the old theory would have called poly-ionic salts, *e.g.*, barium nitrate potassium oxalate, lanthanum nitrate and potassium citrate, and lanthanum sulphate and potassium ferricyanide—which according to Ostwald, should give 3, 4 and 5 ions respectively. This result is quite revolutionary and shows that ionisation is either not a matter of mass-action at all or that the mass only acts normally when it is insignificant in quantity. It remains to find out the theoretical significance of Storch's empirical law, and when this is done it will constitute a discovery of the most far-reaching character, which will affect even many technical processes. Any explanation must be of such a character that it leads to a chemical equation with two *equivalent* substances on the left and three equivalent substances on the right. I have an idea of my own, but it involves hypotheses as to the nature of liquid water, which would require a good deal of proof to be acceptable: the equation is $NaCl + H_6O_3 = Na^{\cdot} + H_2ClO' + H_4O_2$: it requires the assumptions that undissociated salt turns an equivalent quantity of water into H_6O_3, and that the positive ion turns water into H_4O_2 in equivalent amount and that the negative ion is hydrated. I noticed a similar anomaly when I first came to this country and enquired into the solubility of gold by cyanide: its behaviour is entirely in disagreement with any theory deduced by mass-action from Elsner's equation

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[*] Refer to *Journal of the Chemical Society*, 1909, Abstracts ii., 985; taken from Smits and Wuite, Amsterdam Acad. of Sci., 1909.

[†] See this *Journal*, vol. viii., 1908, p. 304.

$$4Cy' + 2Au + O + H_2O = 2AuCy'_2 + 2OH'.$$

I have to confess that in 1904 I believed in these mass-action shibboleths, and I have now to retract certain statements I then published.*

Elsner's equation is merely an integration of a number of more fundamental primary reactions which have never been properly investigated. I recognise that it is too much to expect from the Rand's technical men to take up a problem of this sort, though it is really connected with a possibility of improved or quickened extraction; but I should have thought our University College might have had a try at it.

Generally speaking, we want a new conception of hydrated ions and of the nature of water, and it is possible that a mathematical theory of the depolymerisation of water will be found to reconcile all the discrepancies.

In inorganic chemistry everything, as is to be expected, hangs on the development of the Periodic Law and its curious irregularities. The marvels of radium continue: its latest triumph is the "creation" of carbon out of materials carefully purified from that element. Ramsay has found that all the five common tetravalent elements Si, Ti, Zr, Pb, Th generate CO_2 when exposed to the radium emanation, whereas mercury does not, and that thorium nitrate generates CO_2 from itself without radium. Those of you who paid any attention to my freak atomic theory of 1908 will remember that I predicted this, by making all the tetrad elements derivatives of carbon, though in addition I gave a carbon nucleus to a number of non-tetrad elements.

One of the most puzzling problems in chemistry is the fact that different values for the exact atomic weights are got by different methods, each apparently of unimpeachable exactitude: thus the ratio of silver to oxygen (which cannot be determined directly) is uncertain by about 0·07 (107·86 to 107·93) although the individual results of each method appear to be trustworthy to 0·01. Again carbon is found by combustion of hydrocarbons to be 12·000, whereas by difference (Scott's method) it is 12·02 (Ag = 107·9). I have suggested elsewhere* that the assumption of the existence of an element of very small fractional atomic weight could explain these discrepancies by being transferable from one element to another; *e.g.*, divalent sulphur might be about 32·0, and hexavalent sulphur about 32·1, the increase in atomic weight being taken from the oxidising agent used. Recently a very important complete analysis of nitrosyl chloride has been done, chlorine, oxygen and nitrogen being absorbed successively by silver, copper and calcium: the result was N:O:Cl::14·006:16:35·468. If this were repeated with determination of the amounts of AgCl, CuO and Ca_3N_2 formed, it would go far to clear up the problem.

Another inorganic subject of more popular interest is the formation of diamonds, but I shall content myself with calling attention to the fact that the new progress is in the direction of proving that diamonds cannot be formed at high temperatures whatever the pressure may be, but in nature must have crystallised from a solvent under enormous pressure but at a comparatively low temperature.† It looks as if the new transparent variety of silicon carbide might eventually replace diamonds in popular esteem: it certainly appears to have the properties of the diamond in an exaggerated form.

Another new conception is about the atmosphere, namely, that its composition is only uniform when a particular level is considered. The higher atmosphere is nearly devoid of CO_2, the lighter gases predominate as we ascend, and it is believed that after 20 miles or so there is only neon and hydrogen left, possibly with coronium as the highest of all, there being no definite outer limit to the atmosphere.

I have two other fairly interesting theoretical points to demonstrate, both taken from the very innermost recesses of chemical theory. As Armstrong says "chemistry is the science of inward and spiritual grace": viz., two new points about the nature of the carbon and nitrogen atoms respectively. Carbon has long been known to have its four affinities equally distributed in space, and consequently the tetrahedron is used as a model to explain this equal distribution. By the way, many of the text-books give the misleading idea that the tetrahedral configuration is always regular, whereas in reality it is always distorted according to the various weights and mutual attractions of the attached groups, extreme examples of which are given by CO, fulminic acid and triphenyl-methyl, which must have practically a flat configuration. The particular point I have to show you however, is that one can get two different theories of affinity from the tetrahedral figure according as one assumes the forces to be resident at the points or in the faces. The points would be the positions of maximum force on any electrical theory of chemical affinity, whereas the maximum force would be in the centres of the faces if the force resembled gravitation or depended on a solid tetrahedral mass. For example, on the first theory (the customary one) the picture of ethane is as in Fig. I. and acetylene as in Fig II.

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* S.A. Association Advancement of Science Report, 1904, p. 64 (middle).

* Trans. Royal Soc. S.A., 1909, p. 115, and S.A. *Journal of Science*, 1910, p. 189.

† See S.A. *Journal of Science*, 1910, p. 170, *Chemical Society of London*, Trans. 1908, p. 1333, and Proceedings, 1909, p. [illegible] also new edition of Sir W. Crookes' book.

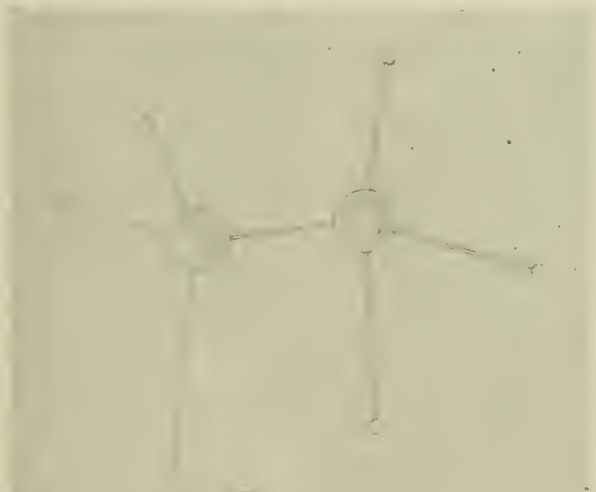
Fig 1

Fig 2

FIG. I. Configuration of ethane (electrical affinities) or of acetylene (gravitational affinities).

FIG. II. Acetylene (electrical affinities) or ethane (gravitational affinities).

whereas on the other theory they are *vice versa*. Now two very interesting points arise which are mutually contradictory; they are (1) that ethane is the stable compound and might be presumed to have configuration 2. (2) that if the gravitational theory is followed out, it involves the existence of more isomerides than there really are: thus it predicts three butanes instead of two, viz., as in Figs. III., IV. and V.

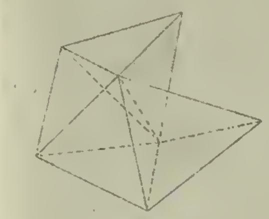
Fig 3

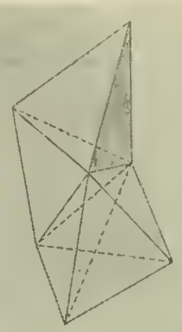
Fig 4

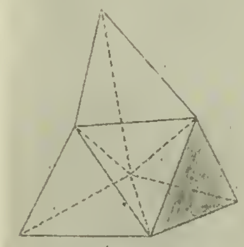
Fig 5

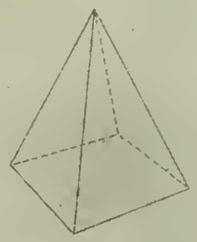
Fig 6

FIGS. III. & IV. Two different solid formulæ for normal butane.
FIG. V. Isobutane.

The electrical theory is therefore the more nearly correct, but it is evidently still far from the truth since it represents acetylene as being more closely bound together then ethane is.

As regards the nitrogen atom, the customary representation is that of a square pyramid, as in Fig. VI., in which the salt forming group, like chlorine, is supposed to be attached to the apex. I have noticed, however, that this formula possesses the same defect as the old plane formula for carbon, viz., that it could give two different pictures for a di-substituted ammonium, just as the flat carbon formula gives two pictures for dichloromethane. It is therefore certainly wrong, and I prefer my own tetrahedral model for nitrogen, as having at least no experimental evidence against it.*

In practical organic chemistry the chief recent advances are the artificial production of rubber, of adrenaline, "ergotine," and hordenine, and the discovery of easy catalytic methods of making ether, aniline and the ketones. Another striking, if somewhat out-of-the-way discovery is the constitution of the famous murex, the "purple of the ancients." The research required the sacrifice of 12,000 of the colour-bearing animals, and the colour turned out to be 6 – 6′ dibrom-indigo, a fact extraordinary both in the occurrence of bromine in a metabolic product, and also in its connection of the animal and vegetable kingdoms, reminding one of the close genetic connection between chlorophyll and hæmoglobin.

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* See this *Journal*, Vol. IX., p. 337.

Agricultural chemistry, which for 20 years or so has been merely a dry record of analyses, has now wakened up from its long sleep, chiefly owing to its contact with bacteriology. It has long been known that soils contain several distinct families of bacteria (counting up to over a million per gram in favourable circumstances), which may be classified according to their behaviour towards combined nitrogen: one class converts elementary nitrogen into protein, the second hydrolyses protein into ammonia, the third oxidises ammonia to nitrites, and the last liberates elementary nitrogen from humus. Some of these are unfavourable to plant growth, and the new experiments aim at their suppression by means of partial sterilisation of the soil, either by means of mild antiseptics like CS_2, $CHCl_3$ or $C_6H_5CH_3$, or by means of the regulated application of heat. The latter point seems to show that the hoary practice of veld burning in this country is not to be condemned outright, as has been done by most authorities, including authors in our *Journal*. The experiments I refer to show that eventually the number of favourable bacteria and the amount of combined nitrogen in the soil increase concurrently, while the other mineral foods become less insoluble. The original soil must therefore contain something capable of limiting the natural increase of the favourable bacteria, this something being put out of action by heat or volatile antiseptics. Russell and Hutchinson, of Rothamsted, have shown that in all probability this limiting factor consists of living protozoa, amœbæ and so forth, which are comparatively highly organised beings and therefore succumb to treatment which bacterial spores can withstand. Unfortunately the antiseptic treatment goes too far and kills off the nitrifying organisms as well as the harmful ones, so that the whole of the beneficial effect seems to come from an increase of the ammonia forming organisms. Even as it is, however, the increase of fertility caused by partial sterilisation is quite remarkable, and the future outlook very promising.

In analytical chemistry the most interesting points are (1) two new precipitants for sodium, viz.: $CuHAlF_6$ and $K_3Bi(NO_2)_6 + CsNO_3$, and (2) a very sensitive colour reaction for nickel with $(COSK)_2$: also interesting general precipitating reactions with dimethylsulphanilic acid and methylorange.

Physiological chemistry has no local interest, but I notice that it looks as if gelatine would be synthesised in the next few years, and possibly even albumen later on.

In the electro-chemical theory I have to confess myself a Conservative. I cannot digest the conception that our atoms are wholly aggregations of electrons: I cling to matter, even if it be only as the "sphere of positive electrification," a fiction which the electronists have (apparently unwillingly) to postulate to hold their electrons together. I think the evidence shows the great bulk of the atomic weight to be non-electrical, *i.e.* that the electrons are simply "an extra," a sort of vivifying principle. They are certainly connected with chemical affinity, a force which is too intense to be explained in terms of gravitation alone. As Nernst says, "It has become evident that electrical forces unquestionably play a great part in many 'purely chemical' reactions." In all probability our "chemical affinity" is a combination of gravitational and electrical forces, and valency is due to a special local aggregation of electrons. The conception of atoms losing some of their electrons without very serious alteration of properties must be considered. Many obscure colour-phenomena, *e.g.* those of ruby, sapphire, and fluor spar, as well as the effect of light on silver salts, seem to belong to this category. There are all stages of stability amongst compounds, and there is every reason to extend the idea to elements. Gold may never be synthesised, but it may quite likely be broken down, given sufficient command over temperature and pressure. I am also not quite satisfied that the α particle of radium is a helium atom with two electric charges. I think it more likely that it has atomic weight 2 and only one charge and that when stopped the two half-heliums lose their charges and combine to form inert helium. The evidence for the monatomic molecule of helium only proves that the molecule has a single sphere of influence.

Another science which appears to be threatened with an overthrow of its fundamental principles is Geology, or rather Geophysics: and again the disturber of the peace is radium. Briefly stated, the revolutionary conception is that all solar and terrestrial heat is due to radium, that the earth's centre is nearly cold, and that the famous nebular hypothesis is not only unnecessary but erroneous. Chamberlin was the pioneer in this field; he re-awakened attention to the fact that the nebular hypothesis is contradicted by all we know about the kinetic theory of gases, which shows that a gas cannot hang together so as to be able to rotate as a whole another case of two branches of science having grown too far apart — and suggested that our system gradually grew from a collection of cold meteorites in space. Wiechert, of Gottingen, has elaborated this, and has collected evidence from four different sources to show that the earth's core, up to three-quarters of its radius, is nothing but metallic iron, solid, comparatively cold and highly compressed of course. The first argument is the relation of the earth's elasticity to the flattening at the poles: the figures are satisfied by an iron globe about 6,100 miles in diameter with a rocky mantle about 900 miles thick. The second is the fact that the earth's central density is about 8, and that experiments show that surface rocks could not be compressed to such a density by any force: the density of uncompressed iron is about 7½, and the conception of an iron core gives an easy explanation of the earth's magnetism. The third argument comes from mathematical calculation of the effects of (1) the fortnightly north and south displacement of the tides, and (2) the time of revolution of the poles round the axis of figure: these calculations agree fairly closely in giving the earth a rigidity, as a whole, rather higher than that of steel. The fourth and most convincing argument is that derived from the velocity of transmission of earthquake shocks; the velocity is constant in the core, and the figures require a comparatively abrupt transition from iron to rock at about 900 miles down: they also show that the central yield to compression is not very great despite the enormous pressure. Wiechert calculates the central pressure to be three million atmospheres, and the pressure at the beginning of the iron to be 500,000 atmospheres. The "heating layer" of the earth is supposed to be only about 50 miles thick, and even in it, the radium is mostly near the surface: consequently the temperature-gradient must diminish as we go deeper, a fact which has important practical consequences. Places like South Africa which have a low gradient near the surface are supposed to be highly denuded, and yet to have an abnormally thick rocky layer below to compensate for the lower density of rock as compared with iron. The theory predicts a maximum temperature in the earth of from 1,500° C to 2,000° C: still this is a temperature quite sufficient to render rock plastic or viscous—which is, of course, required to explain the familiar phenomena of volcanoes. This pseudo-liquid or plastic layer probably lies at between 30 and 50 miles depth, and is quite probably not continuous, but a honeycomb of pillars and cavities. Further details of this interesting theory are to be gained in Prof. Schwarz's paper in the *S. A. Journal of Science* for April, 1910.

In view of the current beliefs about the possibility of very deep mining here, I have worked out the figures for the rock temperature to be expected at different depths, using the suppositions of Wiechert and Schwarz. Assuming 70° F. at 1,000 ft., I calculate 78° F. for 3,000 ft., 85½° F. for 5,000 ft., 92½° F. for 7,000 ft., and 99° F. for 9,000 ft. depth. These results are rather more favourable to the Rand than earlier calculations were, but it is nevertheless necessary to point out that even if the actual temperatures met with turn out to be as above, the temperature of these deep workings must be reduced by ventilation to figures very considerably lower, else work will be very inefficient. Depths over five or six thousand feet will require the installation of ventilation systems quite as powerful as those of European collieries, not for the sake of noxious air, but merely to reduce the temperature. Unless the ventilation is good, the workings will attain a wet-bulb temperature about the same as the rock, and it is now perfectly well-known that efficiency falls off at a wet-bulb temperature of only 82° F. or so, which must therefore be the figure to be aimed at. This question of deep mining is one which, I hope, will be discussed during the coming year, now that the Society has resolved to give mining subjects special encouragement. By the way, I have in my possession a number of observations of rock temperatures recently taken, which are in fair agreement with the foregoing calculations; also the corresponding temperatures in the workings near at hand. These latter are *higher* than the rock temperatures, not lower as one would expect. Now the only other heating factor is breath (and explosives), since the variety of pyrite found in deep levels does not oxidise readily; hence the ventilation must be pretty bad in these cases. If the ventilation were efficient, the air would be colder than the rock: and if the rock is much hotter than 82° F., a considerably stronger air current will become necessary.

In conclusion, I wish to say a few words on education, despite the danger of straying into platitude. Having been brought up on Classics I fully recognise their value in teaching us definiteness of expression and fine verbal and philosophical distinctions: but I am sure they are not the only possible vehicle of culture; as a recent writer says "mental refinement does not exclusively result from literary studies, but is a manifestation of the critical or scientific habit, whereas the so-called *liberal education* is mainly a mere specialisation and does not necessarily carry with it any wideness of view." I am not devoted to the "wisdom of the ancients," in fact I feel that when they did stumble on truth in these pre-scientific days—say before 1850, as I said already—it was in branches of knowledge where, even in our day, there has been no advance, *i.e.*, in subjects not amenable to experimental or statistical treatment, such as ethics. I was amused last year on straying into the Bloemfontein Cathedral to hear a parson instituting invidious comparisons between "ever changing science," and what the parson presumed to call the "unchanging word of God." I regret to have to remind our clerical friends that science is just knowledge and nothing else, and that any institution which does not change with knowledge must naturally get left behind, as the churches undoubtedly have been already.

Anyway what is wanted of modern education is that it shall produce a man who is both a gentleman and an accurate problem-solver; as an American writer says, "it is necessary to understand that moderately accurate knowledge is like a moderately fresh egg": and I may say at once that accurate knowledge is not to be gained from text-books and a 50% pass examination, but solely from the experimental method. Unfortunately, as Armstrong says, our headmasters are "people who have never knowingly made an experiment, and do not even know how to set about making one." The result is seen in the so-called "educated population," three-quarters of whom still believe in water-divining, or in finding live frogs in stones or similar superstitions, all on evidence too that wouldn't convict a nigger. Even the Universities want reforming badly: they should leave text-book examinations to preparatory schools, and they should give proper scientific degrees. Why, for example, should the Cape University call a man a Bachelor of Arts after he has passed an Honours examination in Science? What they should organise is a degree in Technics, or Applied Science, only to be given after training in experiment and research. I assume, of course, that *elementary* science will have been taught at school. The fact is, that British University training is, almost everywhere except in Scotland, modelled on the Oxford and Cambridge type: the former of these Universities apparently considers the acme of human culture to consist in the knack of rendering into more or less indifferent Greek such things as "We don't want to fight, but by Jingo if we do;" whilst the strong point of the other University appears to be the cultivation of a certain carefully artificialised variety of English speech: and in both Universities, sad to say, Chemistry, almost the most fundamental branch of human knowledge, is relegated to a back seat under the name of "Stinks." I will myself say nothing of the educative and disciplinary value of chemistry: let Nernst speak: "the doctrine of the constitution of organic compounds stands at the head of all theories that the human mind has ever conceived."

South Africa will do well for itself if it continues to better its scientific *personnel* and equipment: there are plenty of other scientific fields than agriculture and veterinary science. For example, the number of assayers and samplers here should be much increased, and the importance of their function recognised — goodness knows, their salaries are not a serious item in working costs. One reason for this is: it is the duty of the industry to clear up the disquieting rumours of gold-stealing by accurately ascertaining the gold called-for, not guessing at it within 5% or so. This can only be done by trebling the number of samples taken. Again, in a young country like this, it is natural enough that research should be left to take care of itself, but if this continues too long, the country will feel the loss. As Sir G. H. Darwin said at Capetown "men of science do well to impress on the captains of industry that they should not look askance on those investigations which may seem for the moment beyond any possibility of practical utility." The trouble is not, however, with the captains of industry, but with the practical man. Research is not loved of the practical man—I sometimes fancy he fears it. Anyway the Rand Industry requires officers with more than a technical training, I mean inventors as well as craftsmen: and in the past, as your Transactions show, the Rand has unearthed a number of inventors quite remarkable for its population. I do not know however that they all receive encouragement proportioned to their value

I shall finish with a few words on other industries. So far very few have succeeded except at the coast, a circumstance which is, in my opinion, largely a matter of excessively high railway rates. An example of the handicap is shown by the Modderfontein Dynamite Factory, which so far as I know, uses no Transvaal material whatever except the oxygen of the atmosphere All our paper is imported, though it could easily be made from South African fibres if caustic soda were not so ridiculously dear. The latter and hydrochloric acid would find all sorts of uses in smaller industries, and could be made economically from our numerous salt-pans, which can also frequently give decent yields of sodium sulphate. Bleaching powder could also find many uses, including its very rapid action as a 'cyanicide.' We still import an immense quantity of cornflour and other maize foodstuffs. It is in fact extremely quaint to think that our boarding house blancmange is frequently Transvaal mealie which has travelled 12,000 miles to find its apotheosis! We also ought to make our own "tin plate" and lead pipes, and as Prof. Stanley has shown, our iron industry can do all right if it starts on a sufficiently large scale. I am glad to see that the Electrical Company is going to tackle the manufacture of $CaCN_2$ and of nitrates at once. This is what we need to make South Africa self-supporting. Generally speaking, this country will be better off when it settles down to quiet industry instead of looking for "hyperbolical dividends"* and "Stock Exchange counters."

Another thing that commerce would be simplified by is the introduction of a metric or decimal coinage. This would be very easy, as has often been pointed out before. It only requires the abolition of the half-crown and penny, and a trifling alteration of the value of the "tickey," to five for a shilling = £0·01. If eventually necessary, a half-tickey could be instituted as the smallest coin = £0·005, for stamps and tram-fares.

Well, gentlemen, I have finished my long talk. In the course of it I have had a tilt at all my "pet aversions," and though it may seem conceited to have so many "pet aversions," I think I can truthfully say that they are all cases either of humbug or of inefficiency, two things for which I have no patience. Hitherto, in this Society, I have not made unprovoked attacks on persons or institutions, but have only retaliated to criticism in a somewhat *nemo me impune lacessit* style; and most of us *are* pretty convinced of the righteousness of our own particular brand of personal politics. Still I think that the decent scientific man does not ever wittingly make an enemy, and I hope too that his friends will recognise that this critical tendency of his is constitutional and practically incurable, and that if he has not yet acquired wisdom, it has a chance of coming with age.

Mr. Chas. B Saner (*Vice-President*): I have pleasure in proposing a hearty vote of thanks to Dr. Moir for his deep address I will not attempt to criticise it, as it is not in my line. I am glad to see, however, that he has identified himself with this country by taking up "opsitting." We have heard a good many political speeches lately from the Premier, who has over and over again told us that racialism must be killed, and that we must go in for co-operation and conciliation; yet here our President brings up this thorny problem by telling us with glee that "our people," the Scotch, have been running this Society! Why, that is rank racialism!!

Mr. A. Whitby (*Member of Council*): I should like to second the vote of thanks to our new President for his address. Dr. Moir has touched on so many points of interest in chemical technology that it is rather unfortunate, in a way, for it to be the custom of our Society not to discuss the Presidential Address, but I have no doubt that if we are given an opportunity to discuss some of the more controversial points raised in the Inaugural Address at future meetings, we might find a great deal to say.

The President: In thanking Mr. Saner and Mr. Whitby for their kind remarks I should like to suggest that if they wish to controvert any point in my remarks they may get over the difficulty of the precedent by putting their views in the form of a paper.

THE ASSAY OF TIN ORES.

(*Read at March Meeting, 1910.*)

By Jas. Gray, F.I.C. (Member).

DISCUSSION.

Mr. A. Whitby (*Member of Council*): In reviewing this paper one is struck by the author's industry in the bibliographic line, but disappointed beyond measure in the results obtained both by his researches into the literature, and the conclusions drawn by him from the few experiments quoted. It is not too much to say that the paper represents the most exhaustive research in the domain of published matter on the tin assay that this Society is ever likely to see, and the author consequently deserves due credit for wading so deeply into the mud of history. Unfortunately, however, from this morass he has not apparently been able to hold up one single sprig of hope to sustain others. The scanty reference to his own work, the results he quotes, and his description of the method adopted by him as giving the best results, not only lead me to confirm his own conclusion, "there is little or nothing new in these notes," but also convey a most bewildering impression of what is really one of the most difficult assays the technical chemist is called upon to undertake.

A 70% cassiterite presents no terrors. Even the abused cyanide fire-assay carried out under reasonable conditions gives results within $\frac{1}{4}\%$ of the truth on such an ore, but what we chemists are looking for is a method which will estimate with reasonable accuracy the tin in an ore ranging, say, from $\frac{1}{2}$ to 1%. Reading between the lines of the author's paper, the assumption is that in his experiments he took ·25 grm. of high grade ore and mixed it with various substances more or less typical of possible impurities. What I am really anxious to know is, how far his experiments would have carried him if, say, he had worked on a 2% ore with similar quantities of impurities?

Let us presume that the author's iodine had a titre equal to ·005 grm. tin. per c.c. This is reasonable and would point to the use of a 72% cassiterite in his experiments. Taking 1 grm. of a 1% ore which is the maximum allowed in the process outlined by the author, it would require 2 c.c. of the iodine.* He does not think it worthy of attention that with such a ratio of gangue, even silica, there is a considerable absorption of the starch blue without any tin at all being present, so that it is possible that traces may be reported as tangible amounts.

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* [illegible]

Coming to more serious matters, his assumption that 37% arsenic has no great deterrent effect on his results depends entirely on the degree of acidity used. The deliciously vague manner used in describing his method of operation leaves it open to the student to use a maximum, say, 10% free HCl, when I have proved that tin is co-precipitated with the arsenic to a serious extent, as an alloy. Again, with regard to tungsten he lightly dismisses the intense blue oxide of tungsten produced as no mask to the starch blue. This is entirely incomprehensible to me. Recent experiments of my own have shown an utter impossibility of correct reading under such conditions especially when using iron as reducer, since the carbon tends to mask the colour.

Why he should dismiss the good old method of reduction in hydrogen because of the uncertainty of the extent of reduction, when his own method, as I have frequently proved, is open to the same objection, I must again question.

Pressure of work in other directions has unfortunately precluded an examination into the effect on the tin assay of the other impurities mentioned, but I have said enough, I think, to show that the author's paper, excellent as it is from the historical point of view, sheds no new light on the subject of tin assay, but rather tends to render confusion more confounded.

The President: It appears as if we would almost require a sub committee of the Society to deal with this subject, seeing that the authorities differ so much.

Mr. F. H. Thorburn (*Associate*): Would the author kindly let us know how he makes up his standard solutions of ferric chloride and iodine: the quantity taken of each to a litre of water?

Mr. James Gray (*Member*): 21·32 grm. iodine dissolved in a solution in water of 15 grm. KI and made up to 1 litre. 1 cc. = 10 mgm. tin.

$FeCl_3$ about 180 grm. $FeCl_3$ dissolved in 200 cc hydrochloric acid, evaporated to dryness, taken up with 300 cc HCl and diluted to 2 litres with water. Standardised against pure tin, 1 cc. = 10 mgm. tin (approx.).

EXPERIMENTS ON THE PRECIPITATION OF GOLD FROM CYANIDE SOLUTION BY CARBON IN LIME.

(Read at April Meeting, 1910.)

By Edward H. Croghan (Member of Council).

DISCUSSION.

Mr. C. Toombs (*Member*): In January last we had a very excellent paper on "the precipitating effects of substances containing various forms of carbon and cellulose on cyanide solutions containing gold and silver," by Messrs. Clarke and Sharwood, two of our oversea members. In a table given by them showing the relative activity of several precipitating agents, we find that 5 gm. ordinary bituminous coal precipitated 7% of the total gold from 300 cc. solution containing 2·63 dwt. gold per ton. Also a similar experiment with coke gave a precipitation of 19·6% of the total gold. They also gave facts and figures showing the prejudicial effect of various forms of carbonaceous matter on extraction, when introduced inadvertently into a cyanide plant.

In view of this it is somewhat surprising to find Mr. Croghan coming forward with his theory that the carbon contained in lime is innocuous or almost so.

Since Dr. Caldecott first brought this matter forward a good deal of work has been done on the subject, and one is safe in saying that no hard and fast' precipitating value can be fixed for the carbonaceous matter contained in lime—it all precipitates gold, some to a small extent, and some to a very surprising extent indeed.

Amongst the work done mention may be made of that of Mr. E. H. Johnson, one of our Past Presidents, who used various forms of carbon (carbon picked from lime amongst them) as precipitating agents, pumping gold bearing solution through a filter-press containing the precipitating agent, till saturation was obtained, and then precipitating the gold by electrolysis on a suitable cathode in cyanide solution, the oxygen evolved at the gold-bearing carbon anode quickening the re-solution of the gold from it. Encouraging results were obtained, but owing to the carbon becoming non porous, through gradual packing in the frames, the experiment was discontinued.

Coming directly to the experimental work in Mr. Croghan's paper, his use of HCl in separating the carbon from the lime has negatived the value of his results—occluded gases may be set free, both from increase of temperature and chemical action—sulphides which I have

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Note.—In response to Mr. F. H. Thorburn's query regarding the strength of solution I find that Mr. Gray uses a standard of 1 cc. iodine = 0·01 tin. Of course this only makes my argument the more cogent.

almost invariably found associated with this form of carbon, and which may precipitate gold from solution, are broken up also.

Some experiments have been made to test the theory that his preliminary treatment in obtaining the carbon is prejudicial to its gold precipitating power.

Several pieces of coke-like cinder were picked from a large mass of lime. They were brushed with a stiff brush to remove all loose adhering lime and ash, then crushed through a 60 × 60 sieve.

Specific Gravity 2·35
Carbon = 67·2%.

Some picked lumps of well burnt white lime were also crushed through a 60 × 60 sieve, and a mixture of 50 gm. carbon and 950 gm. lime made. 100 gm. of this mixture was then treated carefully as described by Mr. Croghan, the residue being well washed and dried. Carbon in residue = 70·4%. This increase in carbon contents from 67·2% to 70·4% shows that the HCl has dissolved out some compounds—in this case probably FeS, as SH_2 was given off vigorously when a small portion of the original material was treated with acid.

A slime solution was obtained for use in all the following precipitation experiments. It contained

KCN = 0·013%
CaO = 0·0017%
Gold = 1·44 dwt. per ton

1 litre of solution was used for each experiment.

The scheme followed was—various weights of the carbon lime mixture were each added to 1,000 c.c. of above solution, and a weight of HCl-separated carbon corresponding to the amount of carbon contained in each respective weight of the mixture was also added to 1,000 c.c. solution. The bottles were shaken intermittently for about five hours, and left over-night. They were filtered and the solutions assayed. All the carbon limes were collected on one filter paper, and the HCl-separated carbons on another, washed free from KCN, and their gold contents determined by pot assay.

Original solution = 1·44 dwt. per ton.

Precipitating Agent.	Solution on Mixture.	Loss of gold.	Solution on HCl Carbon.	Loss of gold.
gm.	dwt.	dwt.	dwt.	dwt.
0·5	1·37	0·07	1·40	0·04
0·25	1·39	0·05	1·41	0·03
0·1	1·41	0·03	1·42	0·02
0·05	1·42	0·02	1·43	0·01
0·025	1·43	0·01	1·43	0·01
0·01	1·44	Nil	1·44	Nil

Gold contained in lime + 0·935 gm. C = 0·19 mgm.
,, ,, in same weight HCl carbon = 0·08 ,,

Weights of the mixture containing 1 gm., 0·5 gm., and 0·1 gm carbon, and the corresponding weights of the HCl-separated carbon were added to 1,000 c.c. solution, shaken intermittently, filtered and washed well on the following day. The gold contents on each filter-paper being determined by pot assay.

Precipitating Agent.	Carbon lime mixture.	HCl carbon.
1 gm.	0·23 mgm. gold	0·08 mgm. gold
0·5 ,,	0·12 ,, ,,	0·04 ,, ,,
0·1 ,,	0·03 ,, ,,	0·01 ,, ,,

The original precipitating power of the carbon was determined and 1 gm. of the original carbon precipitated 0·25 mgm gold.

The precipitating power of the carbon as separated by Mr. Croghan is only one-third of its original power, although containing 4% more pure carbon than the original.

A very large proportion of the lime used in reduction works is added in the slime plants and the carbon remains with the settled slime owing to its high specific gravity.

Recently, during the course of some elutriation experiments on slime residues, I always found some specks of carbon, with flocculent organic matter remaining with the sand portion elutriated—I passed the sand through a 90 × 90 sieve—and caught an assortment of carbonaceous matter, which I found to vary from 0·05 to 0·2% of the total dry slime, and which gave assay values varying between 10 dwt. and 20 dwt. per ton. This carbonaceous matter consisted of small particles of coke associated with some fibrous matter, and was by no means all caught on the 90 × 90 sieve — the above figures speak for themselves.

As a disinterested onlooker, Mr. Croghan's view is breezily optimistic, but his optimism is all on the seller's side. It is surely no great expense to hand-pick the lime as free as possible from carbon—one firm of sellers already does this, and their lime contains from ·001% to ·01% free carbon, comparing most favourably with unpicked limes, which often contain as much as 0·2% free carbon.

To Mr. Salkinson's suggestion as to a discussion on the deleterious effect of organic matter coming from underground, I would say "where ignorance is bliss, etc."

Mr. Jas. Gray, F.I.C. (*Member*): There is no doubt that the subject brought forward by Mr. Croghan is of great importance both to the metallurgist and lime burner, especially in view of the divergent opinions held by the former. It is, therefore, greatly to be regretted that the results of the author's experiments are so unsatisfactory that only superficial deductions can be made from them.

An examination of Table I., p. 392, which contains the details of the experiments together with the results shows this to be the case. Tables II. and III. being merely exercises in simple arithmetic, need not be considered, and can be regarded as padding. The author found that 11·1 mgm. of carbon had the same precipitating effect as 2·1 mgm.—both precipitating 0·01 mgm. gold. One would certainly have expected that some relationship depending on the quantity of carbon employed would have been found. The only difference, however, is to be noticed in the column headed "remarks on slides," from which the following equation can be deduced :—

1 large speck = 4 specks (size not specified).

To me it appears that the only deduction to be made from the results of the first five experiments is that 0·45, 0·34, 0·25, 0·17 and 0·085% carbon in lime are equally good or bad, which, as Euclid has it, is absurd.

The sixth experiment using 0·42 mgm. carbon less than in the fifth, results in the gold precipitated being unweighable. A consideration of what 0·01 mgm. means on the assay balance, using a half milligram rider will convey an idea of the difference between 0·01 mgm. and an unweighable quantity. Might I suggest that the author in "straining at the gnat has swallowed the camel."

As the gold precipitated by 0·42 mgm. of carbon was also unweighable, a lime containing 0·068% carbon is on a par with one containing 0·017% carbon, or to quote the result of the microscopical examination of the slides—1 speck = 1 speck. Q.E.D.

It is absurd to endeavour to make deductions on experiments such as these, and it would appear that the author has recognised this, as he says, "the preceding remarks are in a sense difficult to deduce, but possibly others have data, and I trust will go fully into the matter." I cannot see why 0·5, 0·4 and 0·3% carbon in lime should not be accepted when they, according to the author, only precipitate as much gold as 0·2% and 0·1%, of which the first is open to question and the second might be passed. Further, as 0·08 as well as 0·02% precipitates an unweighable quantity of gold, it cannot be said they differ. The least that can be asked when an important matter like this is brought forward, and conclusions drawn from the results of experiments, is that the experiments should be like Cæsar's wife and the deductions drawn from them logical.

I regret that pressure of work has prevented me from contributing data on the subject, and I second the author's hope that those who discuss the subject will do so by means of new data.

NOTES ON PRECIPITATION.

By F. D. Phillips (Member).

Precipitation being such an important factor in the cyanide process, and one which so seldom comes up for discussion before the members of our Society, I thought it might be of interest to quote the results of some special samples taken of the solution entering and leaving the separate compartments in some of the extractors at the Knights Deep. The idea arose from a desire to find out what work each compartment did, and to see if it was possible to effect some saving in the zinc consumption.

Up to this period we had been dressing each compartment of 18 cub. ft. capacity with approximately 120 lb. of zinc, or 23·3 lb. per cub. ft. Great care was taken in dressing, so as to avoid the subsequent formation of geysers or channels, and particular attention was paid to the corners and sides of the compartments, the aim being as far as possible to get each molecule of the solution into actual contact with the zinc. To get as accurate a sample as possible a small test tube was employed, and quantities of 80 c.c. of inflow and outflow solution were taken every fifteen minutes over a period of eight hours. Care was taken to avoid buboles of hydrogen and any particles of dust floating on the surface.

The following is a table of results of sampling a strong extractor two days after dressing, the rate of flow being 1·54 tons of solution per cub. ft. of zinc per 24 hours. Strength of solution: 0·029% KCN, NaOH 0·005%.

Strong Extractor Sampled Two Days after Dressing.

Compartment No.	Entering dwt. per ton.	Leaving dwt. per ton.	Extracted Dwt. per ton	Per cent Extraction	Per cent Extraction of Original Value
1	2·56	0·48	2·08	81·2	81·2
2	0·48	0·14	0·34	70·8	13·3
3	0·14	0·06	0·08	57·1	3·1
4	0·06	0·03	0·03	50·0	1·2
5	0·03	0·01	0·02	66·6	0·8
6	0·01	0·01	—	—	—
7	0·01	0·01	—	—	—

In this box it will be seen that the extraction in the top compartment was 81·2%, whilst the remaining four varied from 70% down to 50%. It is somewhat difficult to understand why the larger extraction did not continue throughout the box. Had this desirable precipitation been obtained, the total work would have been done in four compartments at the most, whereas six were necessary. Some information as to the reason why precipitation falls off at the lower compartments would be valuable.

Although these samples were taken only two days after dressing it appeared the seventh or last compartment might be dispensed with, so it was decided to try with six only. After a run of fourteen days, during which time the storage or foot of the box had daily averaged 0·010 dwt., another series of samples were taken, the rate of flow being 1·46 tons of solution per cub. ft. of zinc per 24 hours, strength of solution: 0·035% KCN, NaOH 0·0051%.

Strong Extractor (Six Compartments) Sampled Fourteen Days after Dressing.

Compartment No.	Entering dwt. per ton.	Leaving dwt. per ton.	Extracted dwt. per ton.	Per cent. Extraction.	Per cent. Extraction of Original Value.
1	2·87	1·19	1·68	58·5	58·5
2	1·19	0·75	0·44	37·0	15·3
3	0·75	0·24	0·51	11·3	10·8
4	0·24	0·06	0·18	75·0	6·3
5	0·06	0·04	0·02	33·7	0·7
6	0·04	0·02	0·02	50·0	0·7

This time, although the value entering was 0·310 dwt. higher than in the former, the extraction was 22·7 less in the first compartment, and this may have been due to channels in the zinc, or in other words the solution not coming into contact with the shavings. The fourth compartment was responsible for a good extraction—namely, 75%. Had the box been left another week or so without redressing, the values of the fifth and sixth compartments would probably have considerably increased. Incidentally I may say that the whole of the strong boxes—five in number—ran a further three days before dressing, and although the one here quoted shows the tail at 0·020 dwts., the average of the five for the seventeen days was 0·010 dwt. From these results we therefore felt that we were quite justified in working with six compartments instead of seven, and this is now being done with very good results. It was next decided to see what the effect would be with a lighter packing of zinc, so at the next clean up about 330 lb. or 18·3 lb. per cub. ft. were returned to each compartment instead of 420 lb. per compartment, or 23·3 lb. per cub. ft. as formerly. This meant that instead of filling the last compartment with new zinc at each dressing of the boxes, as had hitherto been the practice, more than 60 of the amount put in at the previous clean up had to be returned to each individual compartment at this particular dressing, thus to a certain extent eliminating the safety valve that had previously been somewhat relied on.

A weak box of six compartments, lightly packed, was sampled. The solution entered at 0·28 dwt. The rate of flow was 1·88 tons per cub. ft. of zinc per 24 hours. It contained 0·014% KCN, and 0·006% NaOH.

Weak Extractor Sampled Eight Days after Dressing.

Compartment No.	Entering dwt. per ton.	Leaving dwt. per ton.	Extracted dwt. per ton.	Per cent. Extraction.	Per cent. Extraction of Original Value.
1	0·28	0·08	0·20	71·4	71·4
2	0·08	0·06	0·02	25·0	7·1
3	0·06	0·04	0·02	33·0	7·1
4	0·04	0·02	0·02	50·0	7·1
5	0·02	0·02		—	—
6	0·02	0·03		—	—

Again the first compartment did nearly all the work with an extraction of over 71%. The following three compartments precipitated the same amounts, namely 0·020 dwt., whilst the last two were useless. The value leaving the sixth compartment being higher than those of the fourth and fifth might have been due to faulty sampling or salting. This system of lighter packing was also tried on the strong and slime extractors, but proof soon came from the assay reports that the system would not work except with a very high tail value, so the conclusion arrived at was that the lighter packing was inefficient, and the larger weight was again resorted to.

A slime extractor was now taken, and the results showed that the first compartment was in greater evidence than either the strong or weak, resulting in a precipitation of nearly 90%, leaving only 0·040 dwt. for the last three compartments, each of which extracted 0·010 dwt. These boxes are slightly larger than the others, having a capacity of 22 cub. ft. to each compartment, and the amount of zinc 19·5 lb. per cub. ft. The rate of flow was 2·05 tons of solution per cub. ft. of zinc per 24 hours, strength of solution: 0·008% KCN, and 0·012% NaOH.

Slime Extractor Sampled Seven Days after Dressing.

Compartment No.	Entering dwt. per ton.	Leaving dwt. per ton.	Extracted dwt. per ton.	Per cent. Extraction.	Per cent. Extraction of Original Value.
1	0·36	0·04	0·32	88·9	88·9
2	0·04	0·03	0·01	25·0	2·8
3	0·03	0·02	0·01	33·3	2·8
4	0·02	0·01	0·01	50·0	2·8

The temperatures of the slime solution on this plant are considerably increased by means of waste heat from the power plant, and it is evident that this played an important part in the precipitation, inasmuch as there was very little white precipitate present to foul the zinc; but it was felt that the dressing of these boxes could not be improved upon, so they remain as before.

An important point not to be overlooked in the matter of precipitation is the cost. It can, of course, be readily understood that it would be unwise to try and obtain a very low tail value at a cost exceeding the value recovered, so some calculations were made to find out what the expense would be to precipitate on a value leaving the boxes at 0·020 dwt. Taking an average over the twelve months ending March last, the figures including power, zinc, labour, white and native, solution pumps, lead acetate and smelting, the cost came out at 0·75d. per ton of solution precipitated, or to be on the safe side, say 1d. It will thus be seen that in precipitating at a tail value of 0·020 dwt. we were well in hand and could even afford to fill up an extra compartment with zinc in each box—not a small item when it is considered that we have forty boxes in the extractor house here.

The writer does not pretend to have made a deep investigation into the subject of precipitation, but simply quotes the results of a series of special samples which were taken for the purpose of ascertaining exactly what occurred in each compartment, with the ultimate object, if possible, of effecting a saving in zinc.

The President: I have great pleasure in moving a very hearty vote of thanks to the author for his interesting paper, and for the scientific way in which the results are expressed. I hope some of our authorities will take the matter up and endeavour to give us some new ideas on this poor precipitation in the last compartments of the boxes.

Mr. A. F. Crosse (*Past-President*): I should like to second the vote of thanks for a very interesting paper. It is a good many years since we went into the question of precipitation, and I am very glad it has cropped up again.

Dr. W. A. Caldecott (*Past-President*) (contributed): The author has raised a very interesting point in the matter of the first compartment of a zinc extractor box precipitating a higher percentage of the gold in solution entering it than do the lower compartments of zinc. I have been unable to arrive at a satisfactory explanation of this occurrence unless it may possibly be due to the solution in the first compartment exciting a more vigorous chemical action owing to its containing more free cyanide and less combined zinc cyanide than does the solution in subsequent compartments. Another point of interest is the earlier formation of white precipitate in the first compartment than in subsequent compartments. This may be largely due, as suggested by Mr. F. Maxwell, to the presence of dissolved oxygen in the solution entering the box which facilitates the formation of zinc hydrate in the first compartment.

THE DESTRUCTION OF CYANIDE.

(Read at June Meeting, 1910.)

By James Moir D.Sc. (Vice-President) and Jas. Gray, F.I.C. (Member).

DISCUSSION.

Mr. Tom Johnson (*Member of Council*): I should like to ask Dr. Moir and Mr. Gray what amount of dangerous gases would be given off from current tailing run into the mine, such tailing containing the usual amount of total cyanide? How fast would the gases be given off, and what would be a dangerous percentage in the air? What I am trying to get at is, what part ventilation can play in removing the danger from the gases given off. What is the specific gravity of the gases? I do not know whether I am right or not, but I think too much has been made of the dangers of sand filling: I do not think the danger can be any greater than the danger from falling hanging. Two years ago, before this Society, I expressed the opinion that any dangers attending flushing tailings into the mines would be overcome, and thanks to our chemists I am sure they will.

Mr. H. A. White (*Member of Council*): There are some further points in this paper on which I desire to offer some remarks. I notice the authors have a strong predilection for acid solutions in their miscellaneous data (*c*) and (*i*): and it is curious that both these reactions are possible in alkaline solution in so far as destruction of cyanide is concerned. I find that a 0·10% solution of KCN is very slowly changed in the cold to KCNS by excess of thiosulphate, though the reaction is not a commercially possible one as a 1% conversion takes about a week. On boiling the mixture for about two hours the KCN is completely "converted" in accordance with the following reaction:—

$$KCN + Na_2S_2O_3 = KCNS + Na_2SO_3. \quad (G).^*$$

I have suggested the trial of alkaline hypochlorites for destruction of cyanide in the residues but I do not support the use of acid with formation of CNCl, which is as deadly as HCN, unless the authors have a method of evading this action. The product, in alkaline solution of bleaching powder, is immediately KCNO, and ultimately K_2CO_3, NH_3 or N_2, etc. The following is the reaction:—

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* See this *Journal*, Vol. viii., p. [illegible].

$$Cl_2 + KCN + H_2O = KCNO + 2HCl. \quad (H)$$

I used a saturated solution of fairly old bleaching powder, but this only contained 2·31 gm. of chlorine per litre, titrated with As_2O_3. It was strongly alkaline to phenolphthaleine. On mixing 50 c.c. of this solution with 100 c.c. 0·10% KCN the destruction of cyanide was absolutely complete at once, as tested with Prussian blue reaction and phenolphthaline with copper sulphate. The Prussian blue reaction is given with CNCl if the $FeSO_4$ is added first, but not in this case where KCNO only is formed. The "phine" test is not accurate with large excess of strong hypochlorite, unless a small excess of $Na_2S_2O_3$ is first added. So rapid is this reaction that the equation can be tested by using excess of KCN and titrating back with $AgNO_3$ and KI. Excess of NaHO and a little NH_4HO added before titration will prevent the formation of an obscuring precipitate (possibly $Zn(HO)_2$ where working solutions are used). In this way with synthetic solutions I found 1 gm. KCN equal to 1·121 gm. Cl (theory 1·089), and with working solutions 1·143 gm. Cl. This is sufficiently exact, if the indirectness of methods be allowed for, to show that the side reactions are even less significant that when permanganate is used. There is practically no action on KCNS (even when acid): with $K_4Fe(CN)_6$ when alkaline the reaction is very slow (not complete in 15 hours) but if acid is first added, the formation of $K_3Fe(CN)_6$ takes place at once. If it is ultimately decided that sulphocyanides may safely be left untouched, this will form a cheaper way of destroying the cyanide in current residues, especially in cases where those compounds are responsible for a large consumption of permanganate. In any case with the same assumptions, as on p. 448, the cost per ton of sand would be reduced to 0·35d. (Bleaching powder, 35% available chlorine, is quoted at £4 5s. per ton, London, and I have an offer at £15 5s., Johannesburg. I have assumed only 32% available chlorine, in accordance with my analysis of a sample from a drum not, however, newly opened.) This powder is only slowly soluble in water and it is possible in practice that a proper dissolving plant, such as is used by dyers, may be required.

When reading my first contribution to this discussion I mentioned the fact that coagulants not only increased the rapidity of the action of permanganate upon cyanides but also enable more oxygen to be abstracted so that only two-thirds of that shown by equation A is required. In practice I think that enough electrolyte may be found, especially after working some time, to enable the reaction, especially in presence of fine sand, to proceed as follows:—

$$2KMnO_4 + 3KCN = 2MnO_2 + K_2O + 3KCNO. \quad (I)$$

(200 c.c. 0·10% KCN would require 102·4 c.c. $N/10$ $KMnO_4$

with $MgSO_4$, at once, found 107·0 c.c. $N/10$ $KMnO_4$

with $MgSO_4$, pink for five minutes, found 116·0 c.c. $N/10$ $KMnO_4$

with working solutions and $MgSO_4$, found 136·0 c.c. $N/10$ $KMnO_4$

with working solutions and NaCl, found 154·0 c.c. $N/10$ $KMnO_4$

with working solutions and no coagulant, found 176·0 c.c. $N/10$ $KMnO_4$

Equation A (p. 443) requires 153·2 c.c. for KCN only.)

It is clear that, in this case also, the secondary reactions proceed simultaneously. Red filtrates from above are decolorised at once by H_2SO_4. It will be seen that the monad coagulant NaCl, in accordance with experience in other fields of action, is not so effective as the dyad $MgSO_4$, which is not, however, more efficient than salts of calcium and would probably be excelled by triad iron salts.

I do not fear the accumulation referred to in the addendum to this paper, and I think, in any case, the authors have exaggerated the effect, when it is considered that probably more solution will be left in contact with the sand in the mine than is found in the residues, and that in practice a certain balance will be maintained by means of the water it will be necessary to add in replacement.

I have experimented with glucose as a cheaper form of aldehyde than formalin, but though in strong solution the action is fairly rapid and almost instantaneous in presence of acid, yet in such weak solution as would probably be necessary in practice the destruction of cyanide is so slow as to be, so far as my experiments have gone at present, quite out of the question as a solution of our problem. I had calculated the work could be done at even less than the cost of using alkaline bleaching powder and as the other advantages would be great I hope to find time to pursue the matter further.

The President: I should like to reassure Mr. Tom Johnson on the subject of poisonous gases. Cyanide enclosed in alkaline sands cannot be very dangerous for in Australia they have made a regulation that if ·01% is left in the sands it is not considered dangerous. We have no experiments in this country at present. It seems an astonishing quantity to leave in, and I am in correspondence with the people there to make certain about it, and will let the Society know. The question as to the amount of prussic acid which is poisonous in the air is difficult to determine: I think it is certainly less than one part in a million, and therefore a certain amount of ventilation is absolutely necessary. Mr. Johnson is wrong in supposing the specific gravity of the gas makes any difference: even if it were the heaviest gas in the world it would diffuse.

Mr. Tom Johnson (*Member of Council*): But diffusion is slow and black damp will go to the floor.

The President: Only if it is generated on the floor.

Mr. Tom Johnson: If you go to a colliery you will find the gas on the roof before you have it on the floor.

The President: That is so, but in collieries the gas is given off at a tremendous rate. One other point in regard to what Mr. White said about bleaching powder. Our idea was to make a mixture of bleaching powder and acid mine water, and after the action make it alkaline with lime. We tried it on a small scale, but the conversion of the possibly poisonous cyanogen chloride gas into cyanate seemed to be rather slow.

SAND-FILLING ON THE WITWATERSRAND.

(*Read at June Meeting, 1910.*)

By Edgar Pam, A.R.S.M. (Member).

DISCUSSION.

Mr. Chas. B. Saner (*Vice-President*): I see in the June *Journal* that Mr. Sartorius says "I pose as a sand-filling expert." I do nothing of the sort. Sand-filling is an important scheme for the mines, and we should all try to help to make it a success. I do not believe in destructive criticism only.

Mr. Tom Johnson (*Member of Council*): Mr. Pam's paper comes at an opportune moment as there are, no doubt, many who are waiting to know how sand-filling is going to turn out: and, knowing that it will become general practice, wish to hear and learn something of the subject before they personally have to undertake the subject.

This system is used in a number of countries and has been successful in America, Germany, Austria, Belgium, France, Spain, Sweden and England; its use has generally been confined to collieries, and collieries have what I consider to be an advantage, that is, the weight comes on quickly, also the pack can be carried closer to the face, leaving the roof exposed for a shorter time. We here have the advantage that the roof will stand good for a longer time. In new work it will be necessary to keep the barriers some distance away from the face, especially in machine stopes with the heavy blasting.

This brings us to the author's idea that it might do better to work straight across the strike in lifts of 50 ft. or 60 ft. I do not think this would do, as there is nothing to stop drive pillars being taken out now and packs being put in, which would act as barriers. Stoping on the strike in 60 ft. lifts would be expensive, as any miner will tell you when you start him cutting under a pillar in the stope or under the rib at the top of a stope; I consider there would be more loss than gain.

I note from the paper that the joints of the planks are made tight. Is this not a wrong policy? Would it not be better to leave the joints open for half an inch or so, and use narrow boards with canvas (brattice cloth) inside? It seems to me that by closing up everything that the water will find its way into the cracks and fissures in the roof and floor, and probably put pressure on the ground ouside the barriers; whereas if allowed to filter through brattice cloth, the water would get away easier and possibly reduce the pressure. The cost would not be greater, as the barriers and cloth could be recovered.

Mr. Laschinger, in his remarks at the last meeting, mentioned that the pressure would be the same whether the barriers were closed or left a little open so that the water could get through. Personally I think the pressure would be less if the joints were left open as stated above, for, we know that it is harder to get water to percolate through sand the greater the depth of sand. What is the depth of sand that will stop water percolating through it? Is it less or greater than that due to the length of backs of the drives? In some of the new works the distance will probably be shorter to the side barrier than from the top of the stope to the bottom. Again, under which system will we be able to pack the greater quantity of sand in the same space?

Mr. Laschinger is under a misapprehension if he thinks that all the collieries in Germany have flat seams. At the Shamrock Mine, I. and II. Colliery, Herne, Westphalia, the depth is 1,870 ft., the dip in one portion 43°, thickness of seam 5 ft. 3 in., horizontal distance in which filling is taken 1,530 ft. The Myslowitz Colliery passes, I believe, about 3,000 tons per day into the mine: the thickness of one seam here is 8 ft. to 21 ft., and the thickest seam 28 ft. to 37 ft.; the sand is obtained from a bank 20 ft high and 1,500 ft long, steam dredges and locomotives are used here in handling the sands on the surface, two boreholes are used 780 ft. and 1,100 ft. At this mine wood linings are used as described by the author, these linings passed 40,000 to 45,000 tons of sands before being worn out. The lining was put in by hydraulic rams at a cost of, say, 8d. per ft. Iron lining costing 4s. per ft. did not pass so much. At the Heinitz Mine, the filling is passed through a borehole: to which it is sluiced by a giant working under a pressure of 150 lb. per sq. in. It is passed through a grating with 3½ in. apertures, and the pipes in use are of 7¼ in. diameter.

In referring to sand-filling over two years ago* I said: "On the Rand we could get our tailings to the shaft head much cheaper than to the dump, especially where the cyaniding tanks are higher than the shaft head, as in this case the sands could be flushed out of the tanks into launders, in other cases where the shaft head is higher a tunnel could be driven from the shaft head to the tanks, and in other cases a borehole could be put down on the site of the cyanide works."

Dewatering apparatus could be used at the shaft head, the sands being sent down with only sufficient water to carry them to their destination.

At the Deutscher Kaiser Colliery, where they fill from 1,200 to 1,300 tons per day into the mine, experiments were carried out with different kinds of lining. In the use of white porcelain, costing about 4s. 6d. per ft., about 90,000 tons were put through, and on examination they appeared to be about half worn. A glass lining, costing 3s. per ft., was worn out after 60,000 tons had passed through it: and with a lining of earthenware, after passing the same quantity, the lining was also worn out.

The Cleophas mine does not use any lining, but pipes are patched by the use of the oxy-hydrogen flame, a practice which might be adopted here with the pipes in place.

The Ilsede Hutte Ironstone mine, Peine, Germany, uses flushing, thickness up to 60 ft.

The Gute Hoffnung lead mine, sends its tailing into the mine through a hopper in the mill.

Flushing is also being used at Cleator Moor in an iron-stone mine.

At the Ferdinand mine, near Kattowitz, the seam is 8 ft. thick, dip 25°.

At the Hedwigswunsch Colliery, the depth is 650 ft., and the horizontal distance 3,300 ft.

I visited in 1908 the Fanny Pit, Patience Colliery Co., of Beauyong, near Liège, Belgium, on purpose to see the flushing method in operation, where there were two columns at work, one 6 in. diameter, 460 ft. deep, pressure at bottom 90 lb., and carried along the level (drive) for a distance of 2,110 ft., where all the pressure was used up, then 5 in. pipes were used for running to the dip into the working places: the other column was 1,110 ft. in depth, with 50 lb. to 60 lb. pressure at bottom, carried along the drive for 650 ft., down an incline for 400 ft., then on to the drive until a total horizontal distance of 2,000 ft. was reached when they would not deliver any further, the piping so far was 6 in. diameter, then pipes of smaller diameter were put on and a distance of 3,600 ft. was attained, rise of drives 1 in 100. I have no note of the pressure after putting on the smaller pipes. I found on sounding the pipes that rather solid plugs were going through, at times due to poor mixing on the surface. This instance shows that pipe diameter has quite a controlling influence on the distance to which the sand can be carried on the drives. About three years ago experiments were being tried with air flushing at Kattowitz, but how this system eventually worked I do not know: it seemed then as if it would be successful. Pipes of 4 in. diameter were used and the filling was crushed dolomite of about nut size, with, of course, the fine in it. It was found most suitable to have the pipes in the shaft spiral instead of straight in the vertical course. The air pressure was slight and it was found possible to fill up-hill for some distance; much dust was made, but this could probably be kept down with sprays at the outlet, it does not, however, pack so closely as water-borne sand. This system has been mentioned here by Mr. Weston. I have given these particulars so that members may know a little of what has been done elsewhere, for a knowledge of the distances that sand can be carried, height of places being filled, dip of seam, etc., is just as well gained from other countries as from the Rand itself.

The miners' trouble will be the barriers, and I would suggest the use of a certain number of jack bars as props, not necessarily machine bars, for handiness of putting up and taking down, these to be used in the side barriers, timber posts only being used where the joints of boards come. The putting up of barriers will be one of the greatest expenses, but some of the expense could be saved by those whose blocks of ground are about 400 ft. or 500 ft. long, by working them alternately as mentioned in my reply to the discussion on my paper last month. In this way no side barriers will need to be put in, as the sands pack against the solid in one case and against the sands in the other case. In this way also there is little to fear from any rushes or breaking in of water.

The President: I am quite sure we thank Mr. Johnson for his interesting contribution, and you can all see that he speaks with authority on this subject.

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* See this *Journal*, Vol. [illegible], March, 1908, p. 259.

GRADING ANALYSES AND THEIR APPLICATION.

(Read at May Meeting, 1910.)

By H. Stadler (Associate).

DISCUSSION

Mr. W. H. Jane (*Member*): Part II. of Mr. Stadler's paper, par. 23, contains a very striking example of the possibility of theory unaccompanied by practical application leading to false conclusions.

In considering the application of this truth, we must consider the force applied in the mortar boxes of stamp mills, and the forces applied in tube milling. In the mill mortar box we have an impact, accompanied by a small amount of attrition, which increases as the number of apertures in the screen increases. Consider a battery crushing through a 9-mesh screen, the ore washed by a stream of water carrying off all particles sufficiently fine to pass the apertures of the screen. Here we are very near to the conditions necessary to the truth of par 23. Consider now the same forces operating in a mortar box, crushing through 1,600-mesh screening, with a proportionate amount of water to ore. In this case we have a mass of pulp in the box, the elasticity of which is sufficient to appreciably reduce the energy of the falling body, before it overcomes the elasticity of the pulp, and causes the cohesion of the particles to be destroyed. In this case the conditions under which the energy is applied falsifies the conclusions of the paragraph, since no account is taken of the loss of energy, through the elasticity of the pulp. In a tube mill we have different and I venture to think unknown forces at work. We surmise that reduction of the particles is caused partly through impact, partly by attrition, and also by compression under the weight of the load. A tube mill, in Rand practice, reduces cubes from 4 in. in dia., to sand which will pass through a 200-mesh screen, attaining its highest efficiency at a point at present unknown to the writer, but which is known to be higher than the standard of present Rand practice. In tube milling we probably have crushing, shearing and tensile stresses at work unimpeded by masses of resilient matter, since the specific gravities of different particles cause classification and separation in the tube mill. Operating on the pulp from stamps, by tube mills, we probably find we are very near the truth of paragraph 23 In other words it is only true when the conditions conducing to its truth are observed.

While admitting the truth of paragraph 24, we must consider other factors in connection with it or our conclusions will be untrue. Consider gravity stamps operating on pulp with 1,600 mesh screening and operating on reef with 9 mesh screening.

In the first case there are factors to be considered such as the expulsion of the slimy particles from the mortar, a factor which is practically absent in the second. In paragraph 2[illegible] the author states "Kick's law." It is sufficiently clear to enable us to discard it at once in connection with the reduction of Rand banket. Even when the law is applicable, we must still take other factors into consideration. Knowledge of this law is useless, if we do not apply correctly the mechanical force necessary to its truth. For instance we can shear a cube of iron with a pair of hand shears, but we could not shear another cube if we applied ten times the energy required if we did not use a differently constructed pair of shears. The same applies to stamp and tube mills. What we are doing is using machines differing as the volumes or weights of the bodies to be reduced. The author gives tables showing the relative efficiencies of stamps and tube mills, and in paragraph 32 says "under the conditions of the above examples taken from practical work, the efficiency per h.p. of tube mills (17·42) is only ¼ that of the stamps (70·40)."

We do not call it "practical" work nowadays to put only 290 tons through a tube mill, or judge our efficiency on one unit Is the statement, one of practical work! When the tube mill was newly lined, with 6 in. liners its internal diameter was 4 ft. 3½ in. and it consumed probably 85 h.p. After 66 days run internal diameter was 5 ft. 1 in. and its power consumption 125 h.p.

Did the author feed in 290 tons all through the run! Judging the efficiency of tube mills on the results obtained by reducing the pulp in one passage of the mill will lead to an erroneous conclusion, since the tube is not working at its highest efficiency.

Mr. E. H. Johnson (*Past-President*): Like most of Mr. Stadler's critics, the particular proposition I should like enlightment upon is the paragraph No, 32, in which he states that the mechanical efficiency of the tube-mill is one quarter that of the stamp. This is the more necessary since the author's ingenious development of Prof. Kick's views is, so far, unsupported by any foundation of experimental proof that such views are applicable to banket ore and the conditions under which it is crushed. Until such proof is forthcoming, conclusions based upon Prof. Kick's law with regard to local crushing practice are naturally hypothetical, though of mathematical interest.

A further practical consideration bearing upon the question is that apart from the theoretical efficiency of stamps and tube mills, it is commonly found profitable to crush banket ore to the fineness of 3% +60 mesh, or finer, by the combined use of both appliances. No evidence is available to show that the stamp alone is capable of reducing ore to such a fine state of division, and it is improbable that this could be done except with so trivial a stamp duty as to be exceedingly costly.

For several months past I have been conducting crushing trials at the Cason and Angelo mills, and the following details of two typical trials recently carried out, have considerable bearing upon the relative efficiency of stamps and tube-mills:—

Trial 1. (Double stage crushing).—10 stamps, 1,680 lb. running weight. Screen used 9-mesh per sq. in. Aperture 0·27 in. 1 tube-mill 5 ft. 6 in. + 22 ft. 4 in. liner. 27 revs. per min. Duty obtained 20·03 tons per stamp per 24 hours. Gradings:

	Aperture.	Screen.	Entering Tube.	Final Pulp.
+ 6	(·1212)	18·4	24·0	—
+ 10	(·08)	8·0	9·6	—
+ 20	(·034)	16·0	22·0	—
+ 40	(·013)	14·2	18·4	—
+ 60	(·01)	6·4	8·4	2·3
+ 90	(·006)	8·0	8·6	9·8
+ 200	(·003)	7·8	7·4	23·2
− 200	—	21·2	1·6	64·7

Trial 2. (Single stage crushing). 10 stamps, 1,680 lb. running weight. Screen used 1,600-mesh per sq. in. Aperture ·015 in. Duty obtained 5·88 tons per stamp per 24 hours. Gradings:

+ 60	(·01)	8·7
+ 90	(·006)	16·5
+ 200	(·003)	17·1
− 200	—	24·9
Slime	—	32·8

Trial 1 is of a combination unit consisting of 10 stamps and 1 tube-mill, taking the following horse-power as per ammeter:—

Stamps	...	45 h.p.
Tube-mill ...	...	95 h.p.
		140 h.p

This combination has a capacity of 200 tons per day reducing the ore from zero (mill feed) to the final pulp given.

Trial 2, stamps only, requiring 4·5 h.p. per stamp, and reducing the ore in one operation to the gradings given.

To crush 200 tons per day to equal the capacity of the combined unit, the single stage crushing would require 34 stamps using 153 h.p. The surface of fracture of the final pulp in each case works out as follows:—

Double Stage Crushing.

143310 sq. in. surface of fracture.

Single Stage Crushing.

129782 sq. in. surface of fracture.

(the − ·003 product was taken in each case as a mean of ·0015 in., as elutriation showed 32·8% slime in either case. As regards the other grades, the arithmetical mean was taken to obtain surface of fracture.)

It would appear from this, therefore, that the combination of stamp and tube-mill produces 9·5% more surface of fracture and required 8·5% less horse-power. This hardly supports Mr. Stadler's dictum that the tube-mill only has one-quarter the mechanical efficiency of the stamp. Calculating on the principle of the production of tons of −90 grade per h.p.—which Mr. Stadler considers a fallacious method of calculation, we have:—

Combined mill and tube-mill...	1·25 tons per h.p.	
Single stage crushing	... 1·006 ,,	,,

It would appear, therefore, that the empirical formula expresses the position more accurately than Mr. Stadler's theoretical development of Kick's law.

In these tests the tube-mill was placed at a disadvantage owing to the structural work being incomplete, and the means of returning the tube-mill effluent to the classifier was not in operation, so that only one passage through the tube-mill was obtained. The tube-mill was not, therefore, working to its full capacity and could have taken a materially heavier feed with advantage. As the respective gradings of mill screen product and tube-mill feed show, the tonnage fed to the tube-mill would not exceed 160 tons per day under these conditions. To remedy this there are three alternatives:—(1) Increase the number of stamps, (2) use a larger screen aperture, (3) take advantake of the suggestions put forward in the paper read by Mr. Beaver in 1906 (this *Journal*, Vol. 6, p. 215) of separating the fine ore prior to stamp-milling and deliver to the tube-mill. It is the last alternative that is the most attractive as it involves practically no further power consumption. There are a number of means by which this may be carried into effect, *e.g.*, trommels, grizzles, etc. Mr. Beaver proposes to effect the separation at the delivery of the mill ore belt, Mr. Martin uses grizzles between the ore chute and feeder. My own preference is by means of the feeder itself, using the "Comet" type of feeder which is readily adaptable to the purpose, and which automatically governs the supply of fine ore to amount of coarse ore crushed, thereby maintaining the tube-mill feed constant.

The following grading is of a large sample of ore feed :—

+ 6	(·1212)	67·7
+ 10	(·08)	5·9
+ 20	(·034)	7·8
+ 40	(·015)	6·1
+ 60	(·01)	2·1
+ 90	(·006)	2·3
+ 200	(·003)	2·4
– 200	—	5·7

This shows that approximately 33 of the ore is finer than the screen used in the mill, and that, therefore, a total duty for the ten stamps and tube-mill combination may be anticipated of approximately 300 tons of ore per day, giving the tube-mill a quantity of feed more in accordance with its capacity.

En passant, I might here emphasize the point I endeavoured to make at the time Mr. Stadler read his paper with reference to the stamp getting credit for work that is already accomplished before the ore reaches the stamp. The grading here given shows that 10% of the ore was already finer than 60 mesh, and for this the stamp receives credit as for work done.

The figures of duty given by Mr. Beaver, in the paper referred to, encourage the belief that the stamps will accomplish the same tonnage with the fine ore eliminated as when taking the total mill feed, and the 33 fine eliminated will be a nett gain of duty. The mechanical energy required to produce a pulp of a given grading value, under these conditions, should show a material reduction.

Tests on these lines are being conducted, details of which I hope to be in a position to give by the next meeting.

The proceedings then closed.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Carbon Monoxide Poisoning by Explosion Gases.—"The gaseous products obtained by the decomposition of different explosives in an explosive-bomb contained the proportions of carbon monoxide and dioxide shown in the following table :—

Explosive.	Carbon monoxide.	Carbon dioxide.
	per cent.	per cent.
Nitrocellulose powder ...	46·87	16·8
Gelatin-dynamite ...	34·00	32·68
Carbonite	36·0	19·2
Picric acid	61·05	13·46
Trinitrotoluene	57·01	1·93
Ammonal	23·74	6·09

It was found that the gases collected after blasting with picric acid and trinitrotoluene, and after firing nitro-cellulose powder with a normal cartridge, contained the same percentages of carbon dioxide and monoxide as the products obtained in the explosion-bomb. If, however, on firing, a muzzle-flame was produced, most of the carbon monoxide was oxidised to dioxide. Physiological experiments with the explosion gases showed that these produce typical carbon monoxide poisoning.

Under abnormal conditions (*e.g.*, in blown-out shots), oxides of nitrogen may be produced by the explosion of nitro-products. In experiments with 0·5 grm. of explosive in an iron tube open at one end, 68 c.c. of nitric oxide were obtained from dynamite, 32 c.c. from picric acid, and 25 c.c from trinitrotoluene. Small quantities of hydrocyanic acid were also detected in the gases from picric acid, trinitrotoluene, and carbon-carbonite ('Kohlen-carbonit')."—L. Lewin and O. Poppenberg. *Zeit. ges. Schiess. u. Sprengstoffw.* 1910, 5, 4-7, 25-29.—*Journal Society of Chemical Industry*, Feb. 15, 1910, p. 177. (J. A. W.)

A New Method of Determining Copper.—"The method is based upon the fact that, when copper ferrocyanide is dissolved in a solution of potassium cyanide, the completion of the reaction is accompanied by a very sharp change of colour. The copper solution must be exactly neutral and free from iron, lead, zinc, nickel, cobalt, manganese, and ammonium salts. The solution in nitric acid is, therefore, evaporated to dryness with sulphuric acid, any lead sulphate is filtered off, after adding cold water to the residue, and the copper is precipitated as cuprous sulphide, by acidifying with sulphuric acid and boiling with 10 to 20 c.c of a 50 solution of sodium thiosulphate. The sulphide precipitate, washed with boiling water, is dissolved in nitric acid, the excess of acid is boiled off, and the solution made up to a known volume. An aliquot part is titrated with standard alkali, using methyl orange as indicator and another portion is then exactly neutralised, without the addition of an indicator, this solution, after adding potassium ferrocyanide, being ready for titrating with potassium cyanide. The potassium cyanide solution is made by dissolving 6·5 grms of the salt per litre and is standardised by titration against a solution containing 1 grm. of copper per litre; for this purpose 1 c.c of a 10 solution of potassium ferrocyanide is added to 10 c.c. of the copper solution, followed by the potassium cyanide solution, until the reddish-brown colour suddenly changes to greenish yellow. The determination is not affected by the presence of tin, arsenic, antimony, or organic acids, but, in the last case, phenolphthalein is used as indicator, in the place of methyl orange, and a very dilute solution of copper is employed."—J. A. Sanchez. *Bull. Soc. Chim.*, 1910, 7, 9–17. *Journal Society of Chemical Industry*, Feb. 15, 1910, p. 178. (A. W.)

Ferric Iron Estimation.—"It may not be generally known that ferric iron may be more rapidly estimated by an iodometric method (which does not involve reduction) without taking any special precautions. To obtain the best results, it is doubtless necessary to follow the method of Mohr as stated by Treadwell (Analytical Chemistry, Treadwell and Hall, ii. page 532, and in 'Sutton's Volumetric Analysis'). In this, the ferric solution is placed in a stoppered bottle, the free acid nearly neutralised, and the air displaced by a current of carbon dioxide.

Solid potassium iodide is then added, the bottle closed, shaken and allowed to stand in the cold for 20 minutes. The liberated iodine is then titrated with decinormal thiosulphate. Treadwell states that the results obtained by this method are satisfactory. Moderate accuracy may, however, be obtained without taking any of these precautions. The ferric solution is acidified with hydrochloric acid (the amount used does not seem to be of any importance), a few grams of potassium iodide added, and the iodine liberated titrated at once with standard thiosulphate. If the substance under analysis is, as is almost invariably the case, insoluble in water, it is dissolved in strong hydrochloric acid, to which a few grams of potassium chlorate are added if any ferrous iron is present, in which case the liquid is boiled almost to dryness before titration. The thiosulphate may be standardised with iodine, or with a ferric salt of known purity such as iron alum (de Koninck, *Bull. Soc. Chim. Belg.*, 1909, p. 222).

In this laboratory starch solution is never used as an indicator in iodometric processes unless the solution being titrated contains a coloured compound other than ferric or bichromate. We find the end point just as sharply by watching the disappearance of the last trace of yellow colour due to the iodine.

As illustrations of the degree of accuracy to be expected in rapid work, the following examples are given. The thiosulphate solution used was standardised against pure iodine. Thiosulphate is quite a convenient standard for general use, as if carefully made and standardised in first instance, it can be kept unchanged for months. In this respect it is more satisfactory than permanganate. 1 c.c. of decinormal thiosulphate is equivalent to 0·005585 grm. of iron.

I. Analysis of iron alum, containing 11·58% iron: (*a*) 11·44%, (*b*) 11·55%, (*c*) 11·65%, (*d*) 11·59%. Mean 11·56%.

II. Analysis of iron oxide, containing theoretically 69·94%. Found 69·45%.

III. Analysis of a sample of ferrous sulphate:—By bichromate, 20·68%: by thiosulphate, 20·76%.

IV. Apparent value of iron wire:—By bichromate, 100·17%: by thiosulphate, (*a*) 100·16%; (*b*) 100·52%. Mean 100·10%." A. F. Joseph. *Journal Society of Chemical Industry*, Feb. 21, 1910, p. 187. (A. W.)

East African Soda Lake.—"It now appears certain that the recovery of soda from the soda lake at Magadi in the East Africa Protectorate will be undertaken, recent experiments having given good results. The lake is about 93 miles from the Ku station on the Uganda railway. The proposed railway between Ku and Magadi will have an important bearing on the question of the working of the soda lake in German East Africa, which is only about 31 miles from the English lake, and is four times as large as the latter."—*Chem. Zeit.*, 1910, 34, 103.—*Journal Society of Chemical Industry*, Vol. xxix., No. 4, 210. Feb. 28, 1910, p. 210. (J. A. W.)

Zirconium, Preparation and Properties.—"Zirconium was prepared both in the amorphous form, by the reduction of potassium-zirconium fluoride by sodium, and in the crystallised form, by using aluminium in place of sodium. The former process yielded a product containing 98% of metallic zirconium and a small quantity of oxide. The latter gave an alloy containing about 27% of aluminium, removable only by volatilisation by long continued heating at a high temperature. As the crystallised metal was found to be no more suitable than the other for the further experiments, the amorphous metal alone was used. This was formed into rods by pressure, which were first heated in hydrogen in a quartz tube to the highest temperature obtainable in a combustion furnace, then 'fritted' by the passage through them, *in vacuo*, of a sufficient current of electricity. These fritted rods were then used as the poles between which an electric arc was struck, either *in vacuo*, in nitrogen, or in ammonia. In each case globules of the fused metal were obtained, practically pure; no nitride was formed in nitrogen or ammonia. The fused metal has a white colour and lustre like white pig-iron, a hardness of 7 to 8 on Mohs' scale, and is very brittle; it emits flashes of light whilst being powdered in a steel mortar. Its density is 6·400 at 18° C., corresponding to an atomic volume of 14·2; its specific heat between 20° and 100° C. is 0·0804. In compressed oxygen in a bomb calorimeter it burns completely to dioxide, with evolution of 44·56 calories for 1 grm. equivalent (¼ atomic weight). When heated strongly in air it oxidises; the product is never pure dioxide with the oxide Zr_2O_3, which is produced by the combustion of the hydrogen compound (see below). Heated to redness in chlorine, the powdered fused metal slowly reacts, forming tetrachloride, with sublimes. Bromine and iodine are without sensible action on it. Hydrogen at a red heat is slowly absorbed by the metal, with formation of a black powder, of formula, ZrH_2. This when heated in oxygen, yields water and the greenish-white oxide, Zr_2O_3. The metal is very indifferent towards acids, being readily dissolved only by hydrofluoric acid." L. Weiss and E. Neumann.—*Z. anorg. Chem.*, *Journal Society of Chemical Industry*, Feb. 28, 1910, p. 218. (A. W.)

Tungsten.—"An elaborate and detailed examination of the various methods of obtaining metallic tungsten. Wolfram reduced by aluminium gave a metal containing 80 to 90% of tungsten, and by heating this metal, out of contact with air, with hydrochloric acid, the tungsten content could be somewhat increased. Zinc failed to effect reduction to metal, as did magnesium. Wolfram heated with carbon in the electric furnace gave a metal containing 81% of tungsten and above 2% of carbon; and this metal when re-melted under tungstic acid and fluorspar had its carbon reduced to 0·24% and its tungsten raised to 86%. Tungstic acid was then prepared from wolfram, the best method being to fuse or frit with calcium carbonate and salt or calcium chloride; the product dissolves completely in hydrochloric acid, and the separation of tungstic anhydride is then effected by ordinary methods. Tungstic anhydride mixed with zinc-dust in excess (30 to 35%) and rapidly heated to redness, enters into reaction, and yields a mass from which hydrochloric acid dissolves away the excess of zinc, leaving a residue which, when washed and dried, consists of practically pure tungsten. Electrolysis of fused tungstic anhydride gave a product containing 90% of metallic tungsten; and when the electrolysis was conducted under cryolite, some of the melted globules contained 96% of tungsten, but no considerable quantity of melted metal could be obtained. Barium tungstate electrolysed under barium chloride gave a metal of 94·5% purity, and calcium tungstate and aluminium one containing 93·4% of tungsten. Tungsten trisulphide heated with lime in the electric furnace, in proportions corresponding to the equation, $WS_3 + 2CaO = W + 2CaS + SO_2$, gave tungsten of 99% purity, with a little carbon and silicon. By none of these methods, however, was a really melted metal obtained, and after many failures to melt the amorphous metal, success was finally attained on lines similar to those adopted for zirconium (see preceding abstract). The fritted rods were held horizontally, and the arc between them concentrated by means of a magnet; the beads of melted metal dropped into a zircon crucible placed below. The fusion could be carried out either *in vacuo*, in nitrogen or in ammonia—no nitride was formed. The metal has a mercury-white lustre, showing a fine granular fracture (hardness 7·5) when rapidly cooled, and a conchoidal fracture (hardness 6·5) when slowly cooled. It is very brittle when rapidly cooled, but after annealing from redness is moderately mallable and ductile. It is non-magnetic. Its density is 18·72 at 20° C., and its specific heat 0·0358. Burnt in the bomb calorimeter it evolves 32·4 calories per grm. equivalent (⅙ atomic weight). It is practically unaffected by atmospheric air, and when heated in it, becomes superficially oxidised, showing a fine play of colours in the process. Heated in oxygen it burns completely to tungstic anhydride. It is hardly affected by acids, the only solvent for it being a mixture of nitric and hydrofluoric acids, which form a mixture of fluoride and oxyfluoride. Heated in chlorine it is slowly converted into chloride; bromine, iodine, and hydrogen sulphide have practically no action on it."—L. WEISS.—*Z. anorg. Chem.*, *Journal Society of Chemical Industry*, Feb. 28, 1910, p. 218. (A. W.)

GAS VOLUMETRIC DETERMINATION OF HYDROGEN.—"When a mixture of solution of sodium picrate and colloidal palladium (Paal and Amberger, this *Journal*, 1904, 208) is brought into contact with hydrogen, the gas is absorbed, and then reduces the picrate to triaminophenol. Thus, if a sufficiency of picrate be present, a very small amount of the palladium solution will serve for the absorption of a very large quantity of hydrogen. The authors utilise this reaction for the absorption and determination of hydrogen in gaseous mixtures. The absorption is carried out in the usual way, save that the pipette is an inverted cone with a very slightly convex base, so as to afford as large an absorbing surface as possible, for the absorption is not very rapid. Oxygen and unsaturated hydrocarbons must be first removed, for in the presence of palladium, hydrogen reacts with them to form water and paraffin hydrocarbons respectively; and carbon monoxide should also be first removed, as it acts as a 'poison' on the catalyst, and greatly retards the absorption. The solution retains its power of absorption unimpaired for many months."—C. PAAL and W. HARTMANN.—*Ber.*, 1910, *43*, 243—258. *Journal Society of Chemical Industry*, Feb. 28, 1910, p. 236. (A. W.)

METALLURGY.

PRACTICAL HINTS IN THE PRECIPITATION OF SILVER FROM CYANIDE SOLUTIONS. "The precipitation on zinc shavings of silver from cyanide solution is a function of the richness of the solution. The success of the precipitation and recovery of the bullion depends largely on this very important factor. The richer the solution in silver, the more easily precipitation is effected, and the simpler the subsequent recovery operations become. It is therefore important in the manipulation of cyanide solutions, before passing through the zinc boxes, for the metallurgist to arrange the flow of his solutions in such a way that the values are concentrated in a given amount of solution, or at least, in a certain or certain solutions before passing through the zinc boxes. When this method is employed, precipitation gives little or no trouble and a higher grade zinc silver product is obtained. With very high grade solutions in many instances 75% or even 85% of metallic will be found in the zinc silver precipitate, and the precipitation will occur almost wholly in the top compartments of the box. The consumption of zinc is much less on high grade solutions, and the deposition of base metals and minerals does not take place to such an extent, particularly in the top compartments of the zinc box.

It is no longer the best practice to move the zinc of the bottom compartments to the head of the box in filling with new zinc. The zinc should be disturbed as little as possible, and sufficient new zinc put into the compartments, the corners and sides being packed firmly, but not tight enough to cause channelling, and the new zinc being placed on top of the old. This should be attended to daily with as little disturbing of the zinc as possible. Every compartment should be gone over carefully and repacked daily.

Instead of moving the zinc from bottom to top, it is far better to depend upon the manipulation of the solutions, aiming to get a high grade solution that will give the successful precipitation desired by the metallurgist. At all times this method will give a higher grade of precipitate than will result from distributing the zinc from bottom to top.

The most important item in precipitation by zinc shavings is to have an ample zinc box capacity. Certain features of construction must also be carefully attended to. The box should be constructed in such a manner that the flow of solution from the head to the tail of the box will be very sluggish and uniform. There should be no currents or channelling through the zinc. To obtain this result the width of each compartment should be not less than 18 inches. It should have a depth no greater than the length of a man's arm. The length of the box and the number of compartments will depend on the volume of solutions that have to be run through in a given time. The width of the box will depend on the individual taste of the operator. A single unit box should be of such width that a man on each side of the box can work comfortably and clean up. A double unit box must be constructed so that one man can clean up from one side. The space underneath the screen upon which the zinc shavings rest should be not less than four inches deep on one side, tapering from 8 to 12 inches on the other side. The bottom of each compartment should be connected to the common launder by means of a plug or valve, preferably the former.

In present practice the zinc shavings should not be washed outside the box. To permit of this the compartments of the box should be of ample capacity, and should receive a small moveable trommel which is set up in the first compartment of the zinc box where the clean up begins, and the washing should be done in the solution of the box itself. After all the compartments of the box have been washed in this manner by moving the trommel into each compartment successively, the washings carrying the silver zinc precipitate are run out through the launder to the clean up tank. The compartments of the box are then thoroughly hosed out, the box filled with water, and the zinc returned.

It is unnecessary to describe the method of filling the box, as every metallurgist is acquainted with this procedure. The washing of the zinc boxes should be conducted once or twice, and with rich ore, possibly four times a month, and the resulting product should be collected in the clean-up tank. It is unnecessary also to describe the method of recovering the zinc silver slime from the clean-up tank. Either decantation or filtration may be used, the object of either method being merely to get rid of surplus water.

Although it may be used with advantage in many cases, it is not essential to use the acid method of refining, silver being too cheap a metal for this process. In some silver cyanide problems where the ore is particularly free from base metals, and the precipitation has been conducted by the metallurgist in a way that gives a high grade precipitate, it will be found that smelting with borax is the only refining operation necessary. The amount of borax used is for the operator on the plant to determine, but it should not run over 40%. It is quite unnecessary to dry the precipitate. In cases where a little base metal is present, it is sufficient, providing the solutions have been of comparatively high grade, to smelt the moist precipitate with borax, soda, sand and 2% nitre. A tilting furnace of the Monarch type should be used: the precipitate should be charged into a nearly cold pot, and the melting should be pushed rapidly with a good hot fire to finish.

For refractory precipitates to prevent roasting and incidental loss, I give the following method, bearing in mind that silver precipitate at most plants will not stand roasting on account of silver losses, and the low price of the metals:—

Place moist precipitate in a large iron pan, and add water up to from 30 to 50% of the total weight. Add, as fluxes, borax, soda, and 2% nitre. The borax and soda should be kept to the minimum that will permit of good refining. Add a very small proportion of glass powder or fine sand; mix the fluxes with the precipitate in the pan, and shovel the mixture about well. It will be found that this product, resembling in physical appearance a soft mud, can be smelted without roasting. Moreover, fluxes are utilized to better advantage, the base metals are more easily got rid of, and the pots will stand longer. Use a somewhat slow fire to start, rapidly increasing the heat to a hot and strong finish. After pouring, allow the charge to cool thoroughly before disturbing. Should the precipitate contain arsenic, antimony, or other base metals, or if lead acetate has been used in the treatment, there will probably be a matte or speiss, or a mixture of speiss and alloy of lead and antimony or other base metals formed on top of the bullion. If the button does not break off clean after a proper length of time for cooling there is a certain amount of alloy formed on top of the silver. The best way to deal with this is to re-melt the bullion from the entire clean-up in a hot furnace and pass a current of warm air free from excess in moisture, through the molten bullion to oxidise the base metal. Finish up with nitre refining flux. This flux should be made of borax, soda, and 5% nitre. It should not be added until the bullion has been formed, the slag removed, and the bullion re-smelted or the entire charge skimmed without pouring. By this method there will be practically no silver losses in refining, and the resulting bullion should not be less than 950 fine if the operation has been carried out by a careful manipulator. With a little patience it can be sent away from the plant 990 fine, or better, without adding to the cost of plant refining. I do not recommend roasting silver precipitates for the elimination of base metals, except possibly in a few exceptional instances. The method given above is the best that is known at present. It will be found on a little practice that the silver clean up can be accomplished with a very small percentage of flux. The total percentage of flux on the method outlined should not exceed 50% of the weight of the precipitate smelted for comparatively low grade precipitates, with corresponding less amounts for higher grades."—R. B. LAMB.—*The Canadian Mining Journal.*—Dec. 15, '09, p. 746. (A. R.)

CAPACITY OF VACUUM FILTERS.—"As the capacity of a vacuum filter varies considerably more than that of a filter press, the following notes on the estimation of tonnage will probably be of interest to users of these filters.

Estimation by Agitator Measurement.—Tonnage can always be estimated from the specific gravity of the pulp in the agitators (see table Vol. II., p. 374, of our *Journal*).

Estimation of Filter Vat Charges.—For the whole month's run of a plant, the gravity test, as just described, is certainly the easiest way of estimating the tonnage; but at times it is required to have a quick way of estimating the weight of slime which actually adheres to the frames in a filter vat charge; this is very necessary in order to know how much wash must be given in order to get complete displacement of the gold-bearing solution. Too much wash means loss of time and too little wash means loss of gold.

The usual method of finding the weight of a charge is to cut off a square foot of the cake and determine the weight of solid in it by drying, but this method is inconvenient and not very accurate, as different filter frames vary very much in the weight of cake carried according to the length of time that the cloths have been in use. I prefer, therefore, to take the moisture on a mixed sample from several frames, and also get the average thickness of cake by measuring several of them; from these data the weight of dry slime can be calculated. For it will be seen that if one short ton of slime be taken to occupy 12 cub. ft., and one short ton of water to occupy 32 cub. ft., then the number of cubic feet of filter cake to contain one ton of dry slime will be:—

$$\frac{(\text{Per cent. of moisture} \times 32) + (\text{Per cent. of slime} \times 12)}{\text{Percentage of slime.}}$$

This allows nothing for voids in the cake due to air drying, but this amount has been found by experiment to be negligible. For convenience in working, Table II. has been prepared, giving the capacities with varying percentages of moisture and thicknesses of cake. For instance, say a filter vat has a filtering area of 2,880 sq. ft., and it is required to know the capacity with a 2 in. cake carrying 26% moisture. Referring to the table, it is found that it requires 140 sq. ft. of cake of this nature to contain one ton of dry slime, and the capacity of the vat will therefore be $\frac{2880}{140} = 20{\cdot}6$ tons. To find how much wash will be required for perfect displacement of the moisture it is only necessary to divide the dry tonnage by the proportion of water to slime in the cake, which figure is given in the second column of the table. Thus, in the example $\frac{20{\cdot}6}{0{\cdot}35} = 5{\cdot}9$ tons of wash required. Table II. will also be found very useful in the designing of plants for a required capacity. The third, fourth, and fifth columns have been added to assist in this connection. The data given in the sixth column, viz., the number of cubic feet to contain one ton of slime, have also been found useful in estimating the tonnage in very moist slime dams.

TABLE II.—VACUUM FILTER-CAKE TABLE.

1 ton = 2,000 lb. = 12 cubic feet slime - 32 cubic feet water.

Percentage of moisture in cake.	Proportion of water to 1 of slime.	1 cubic foot of cake weighs lb.	1 cubic foot contains lb. slime.	Cubic ft. cake to weigh 1 ton.	Cubic ft. to contain 1 ton slime.	Number of Square Feet to make 1 Ton Dry Slime for Thicknesses of Cake as under.									
						¾ in.	1 in.	1¼ in.	1½ in.	1¾ in.	2 in.	2¼ in.	2½ in.	2¾ in.	3 in.
20	0·25	125	100	16·0	20·0	320	240	192	160	137	120	107	96	87	80
21	0·27	123	97	16·2	20·5	328	246	197	164	141	123	109	98	89	82
22	0·28	122	95	16·4	21·0	336	252	202	168	144	126	112	101	92	84
23	0·30	120	93	16·6	21·6	345	259	207	173	148	130	115	104	94	86
24	0·32	119	91	16·8	22·1	354	266	212	177	152	133	118	106	96	88
25	0·33	118	88	17·0	22·7	363	274	218	182	156	136	121	109	99	91
26	0·35	116	86	17·2	23·3	373	280	224	186	160	140	124	112	102	93
27	0·37	115	84	17·4	23·9	382	287	229	191	164	143	127	115	104	95
28	0·39	114	82	17·6	24·5	392	294	235	196	168	147	131	118	107	98
29	0·41	112	80	17·8	25·1	402	301	241	201	172	150	134	121	110	100
30	0·43	111	78	18·0	25·7	410	309	247	206	176	154	137	124	112	103
31	0·45	110	76	18·2	26·4	423	318	253	211	181	158	141	127	115	105
32	0·47	109	74	18·4	27·1	434	325	260	216	186	162	144	130	118	108
33	0·49	107	72	18·6	27·8	445	334	267	222	191	167	148	133	121	111
34	0·51	106	70	18·8	28·5	456	342	274	228	196	171	152	137	124	114
35	0·54	105	68	19·0	29·3	468	352	281	234	201	176	156	141	128	117

T. B. STEVENS.—*Journal of Chamber of Mines of Western Australia*, Nov. 30, 1909, p. 326. (H. A. W.)

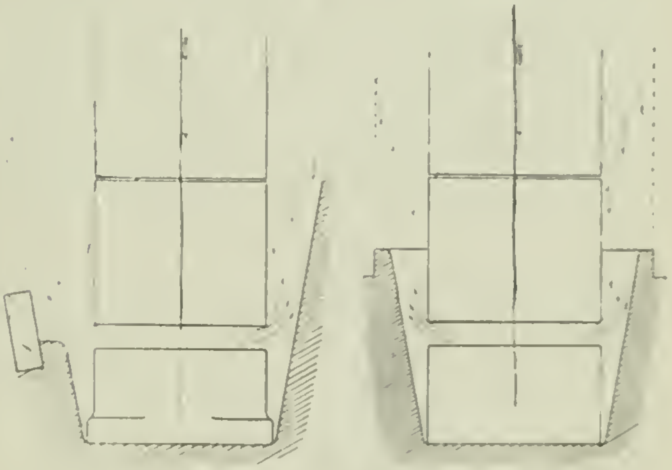
HAS THE STAMP A FUTURE? "It is not uncommon to see in print statements which, if seriously taken, would make the further building of stamp mills seem foolish and absurd. For forty years, however, there has not been a time but what some device has been lauded as being greatly superior to stamps for crushing ore, but throughout those many years the same vaunted rival has not continued. Still, the past is not proof of the future and revelations that revolutionize practice are not an unfamiliar sight. Successful means for crushing rock have multiplied and amplified within the last score or two of years and with them the stamp-duty requirement has been raised, but has it been raised enough to hold its own, and has it further capabilities?

The most primitive and natural means for man to crush a hard substance is by a blow, and to-day if one has occasion to fracture a small stone he almost invariably does it by that means. The crushing effect under proper conditions of a single blow upon rock has undoubtedly maximum effect with minimum loss of power by friction, or otherwise.

Action in the Mortar. The gravity stamp by its nature demands continuous feed and discharge and thus other factors are involved that do not enter that of crushing by a single blow. All oversize must get into the crushing zone and all crushed material must be got permanently out of it with minimum interference. Stamps are effective in proportion to the efficiency of this feed and discharge. Undersize material that is retained is not only a clog to the supply and discharge, but becomes a filling within the interstices, thus reinforcing the coarser particles against breaking. If the efficiency of the gravity stamp can be further developed, it is probably within the mortar and by increased weight of stamp.

A stamp dropping into a depth of liquid pulp effects a rapid and energetic displacement and by its quick rise creates a vacuum that must be filled by air if not by pulp. From the pulp expelled by the blow the fine should be passed permanently away and the coarse returned to the crushing zone. It is quite evident that the suction from the rise of the stamp should more desirably act upon pulp to facilitate supply to the crushing field with minimum disturbance to the ontrush. The stamp in the past has quite properly been called a slime-maker. The cause for this has been the tardy discharge of the fine.

Let us analyse the natural movement of pulp within the mortar. Fig. 1 is a cross-section of approximately an average five-stamp mortar, and Fig. 2 shows a circular mortar with an all-around discharge. In Fig. 1 the point nearest to the screen is seen. In operation in this mortar the ore will bank more or less forming somewhat dished pockets at each stamp, although the whole mass may be somewhat mobile. The initial direction of the splash is horizontal and is diverted upward perforce by its confines, more quickly in a narrow mortar than in a wide one and more vigorously upward from a deep pocket than from a shallow one. From each splash but a fraction of the material passes over the screen surface (indicated by the dotted line), and the rest must get screen exposure from other splashes, whence a cause for slime making.



Advantages of Circular Mortar. In the circular mortar with full encircling screen, and with die deep set as shown in Fig. 2, the outrush from the falling stamp is immediately diverted upward as the only escape from the falling stamp, and it is vigorous in proportion to the limit of cross-section area of its confining walls, and the vigour is a maximum at the periphery of the chamber. In the rapidly recurring splashes each individual splash will meet the material of its predecessor on its downward return, but being more forceful will keep to the periphery and the downward material will take the inner zone and become subject to the suction of the rising stamp. Thus occurs a systematic and regular rotation of all of the material of the splash and that which does not pass through goes back to the crushing zone with no place to rest, an accomplishment by no other means acquired and which yields minimum slime as a result and clearly it has maximum crushing efficiency.

Public demand has been very conservative with regard to weight of stamps. Forty years ago 650 lb. was a weight quite common for stamps, a little later 850 lb. was considered the correct size, and then after a number of years the ideal became 1,000 lb., which to-day is probably the most common, but a 1,250 lb. stamp is not uncommon on this continent. In African practice they have raised the standard faster and are now pinning their faith to a one-ton stamp, and why not? And why should not the stamp still further increase in weight, especially the individual stamp. Is it not possible that within a not distant future gravity stamps weighing tons will be in successful vogue? An object lesson for consideration is that of the ponderous steam stamps which have for years been in use for very coarse crushing of some copper ores, and which have a remarkable capacity. The force of blow which they give may be equalled by weight in a gravity stamp, but not in rapidity of blows which may not be a drawback, but if so, it is more than made up by simplicity and more economical use of power. The steam stamp creates too rapid a succession of splashes to get just the same operation described with Fig. 2, but more advantageously works upon a deeper bed of ore and the rotation as described is more by crowding of the solids rather than from the splash from the liquid, and for that reason is not so suited for the finer crushing. Has not the stamp a probable future?"—M. P. Boss. *Mines and Methods*, Jan., 1910, p. 190. (H. A. W.)

EFFICIENCY OF CRUSHING MACHINES.—"In the October issue of the proceeding of the Australasian Institute of Mining Engineers, R. W. Chapman, of Adelaide University, gives a method of calculating the comparative efficiencies of crushing machines. He calculates that the amount of energy absorbed in shearing a quantity Q of cubes with sides x to a series of cubes of smaller sides z is proportional to $Q\left(\frac{1}{z}-\frac{1}{x}\right)$. This formula would require the calculation of the actual size of the particles and of the reciprocals of these figures. It happens, however, that these reciprocals are practically double the number of meshes per inch in the Institution of Mining and Metallurgy's table of standard laboratory screens, so that the formula can be greatly simplified thus: Energy absorbed is proportional to $Q\,(n-m)$: where n and m are the meshes of the material before and after crushing. To apply this formula in practice it is necessary to have the grading analysis of the feed and product, and to multiply the various meshes by the corresponding percentages of material passing through each. Theoretically the analysis should be close, but this would involve too much work. For practical purposes it is sufficient to average at 30 mesh the material passing through 20 mesh, but not through 40, and so on, and to add say 20 to the finest mesh. As an example, suppose two machines, A and B give the following results:—

Machine A.

Grading.	Feed per cent.	Product per cent.
From 20 to 40	10·3	—
,, 40 ,, 60	18·7	—
,, 60 ,, 80	23·5	—
,, 80 ,, 100	12·2	4·3
,, 100 ,, 120	18·6	19·7
,, 120 ,, 140	10·4	28·6
,, 140 ,, 160	4·8	21·7
Through 160	1·5	25·7

We multiply the mean of each grading into the percentages, add up the results from the feed and product, and substract the latter from the former, thus:

Average Mesh.	Result of multiplying percentage of each by average mesh. Feed.	Product.
30	309	—
50	935	—
70	1,645	—
90	1,098	387
110	2,046	2,167
130	1,352	3,718
150	720	3,255
180	240	4,626
	8,345	14,153

The difference 14,153 − 8,345 = 5,808 is to be multiplied by the quantity of ore treated during a certain time, say, one day. If Machine A treated 20 tons the work done would be proportional to 116,161.

Now take another machine, B, which crushes 17 tons per day, with the following grading analysis:

Machine B.

	per cent. Feed	Product per cent.
From 20 to 40	0·5	—
,, 40 ,, 60	5·8	—
,, 60 ,, 80	29·6	—
,, 80 ,, 100	31·3	—
,, 100 ,, 120	14·7	6·2
,, 120 ,, 140	8·2	12·5
,, 140 ,, 160	7·9	47·2
Through 160	2·0	34·1

Working the result as before we get:

Average Mesh.	Result of multiplying percentage of each by average mesh. Feed.	Product.
30	15	—
50	290	—
70	2,072	—
90	2 817	—
110	1,617	682
130	1,066	1,625
150	1,185	7,080
180	360	6,138
	9,422	15,525

The difference between 15,525 and 9,422 = 6.103. This multiplied by 17 gives the 103,751. The crushing performances of the two machines, A and B, are therefore in the proportion of 116,160 to 103,751. The theory of this method is similar to that of Von Reytt, as quoted in Richards' 'Ore Dressing'."—*The Mining Magazine*, Jan., 1910, p. 64. (W. A. C.)

FERRO-TITANIUM RAILS.—"A curious peculiarity of the new ferro-titanium rails, which have been made experimentally in the United States for a little more than a year, is that they may be entirely free from titanium, and at most they contain but little. In this respect the 'alloy' differs from all others. The properties of nickel-steel, for instance, depend upon the presence of a material percentage of nickel and manganese-steel must have a carefully adjusted proportion of manganese. For the new metal, the steel is made in a converter and blown in the usual way, when it receives a charge of titanium alloy. This is a product of the electric furnace, and is a mixture containing 10 to 15% of titanium and 5 to 7% of carbon, the remaining portion being iron. Each ton of steel receives a charge of about 8 lb. of 10% alloy, the effect being to increase the slag removed from the metal in the ladle, while the ingots are free from blow-holes. The titanium unless in excess of what is necessary to remove the impurities, all passes off in the slag. The treatment gives special durability to rails for curves, and at the Grand Central Station in New York an ordinary rail lost 3·03 lb. per yard in four months, while a titanium alloy rail of the same composition was worn away only 1·01 lb. per yard in six months. The few dollars increase per ton of Bessemer rails is expected to be much more than offset by added durability."—*Canadian Mining Journal*, Jan. 1, 1910, p. 15. (A. R.)

The Electro-chemical System of Amalgamation and Cyanidation.—"In the recovery of precious metals the three methods most generally used by miners, are: (1) The gravity system, used in sluicing, rocking, panning, etc. (2) The amalgamation system, in which values are recovered in the form of amalgams: this system calls for mercury riffles, mercury baths, pan amalgamation or plate amalgamation. (3) The chemical system, in which values are dissolved in certain solutions from which they are recovered by various systems of precipitation. The leaching of values from ore is known as lixiviation and the solvent most generally used is cyanidé of potassium, although other solvents are known to metallurgists.

In the gravity system only the particles of gold and platinum heavy enough to be caught in riffles can be saved; the values in placer slimes, all microscopic and float gold, and all the values associated with the heavy sands are lost.

When we come to consider the saving of values by amalgamation we find hundreds of devices and machines which are designed to force sand, pulp, etc., into close contact with mercury: hundreds of patents have been issued for devices in which the gangue is passed through a mercury bath: but in practical milling operations only two devices are generally used—the copper plate coated with mercury, and the mercury riffle or mercury well. Only free, clean gold will alloy with mercury ordinarily, and only clean mercury has an affinity for gold. Gold in slimes will not amalgamate under ordinary conditions, and microscopic, rusty and coated gold cannot be saved by standard methods of amalgamation. Gold associated with sulphur, arsenic, lead, zinc, iron and tellurium amalgamates with difficulty or not at all, owing to the fouling and sickening of the mercury. Values in slimes cannot be saved on the standard mill plate by usual methods.

In milling operations when mercury wells are used the surface of the mercury often becomes covered with a coating of fine sulphides, which diminishes its alloying properties; for this reason mercury wells are not found in many mines in the United States although they are generally used in Australia.

In lixiviation we find endless difficulties: only when certain conditions are present can cyaniding be profitably employed: there are large areas, such as are found in Idaho, where there are hundreds of mines in which the ore is not amenable to cyanide treatment. While in lixiviation, theoretically, we have an ideal method of gold extraction, yet in practice we find the system hedged about by a multitude of exasperating troubles: generally speaking, cyanide plants are expensive to install, difficult to operate, and the average extraction is not over 85%.

In view of the above very conservative statements regarding the shortcomings of the three popular systems of gold extraction, it is not strange that investigators have long sought improved methods in the metallurgy of precious metals.

Electro-chemical science has devised improved methods for the reduction and refining of the useful materials and it is not unreasonable to suppose that the same science will point the way to even greater improvements in the metallurgy of gold.

We will suppose that both the positive and negative electrodes or wires of a battery or dynamo are immersed in water, the two electrodes being an inch apart. The positive electrode is known as the anode, the negative pole as the cathode, and the separating water is called the electrolyte. When the battery or dynamo is in operation, lines of electrical force pass between the anode and cathode through the water, and there take place in the electrolyte a number of reactions; in other words, electro-chemical conditions are present in the water, and these conditions form the key to the electro-chemical system of amalgamation and lixiviation. An interesting experiment can be tried by any one. Pour a few ounces of mercury into a flat glass or porcelain dish, pour over the mercury half an inch of water containing a pinch of salt or a few drops of acid; immerse the negative wire from a dry battery in the mercury. Now touch the water only with the positive wire and note the activity that at once is manifest. Try this experiment with dirty mercury and note the instantaneous brightening and cleaning of the mercury. Place the positive pole in the mercury and the negative in the water, and the mercury soon becomes foul. Restore the wires to their original positions and note that the mercury clears at once. Grind the mercury in a small mortar with some sulphur or oil and it becomes foul. Now return the foul mercury to the dish and apply the current as at first. Note the clearing of the mercury and see how quickly the detached globules coalesce. Then remove the mercury and add a pinch of bichloride of mercury to the water: immerse both wires and note how quickly there is a deposition of mercury on the cathode or negative wire. In this simple experiment is the germ of the electrolytic system for the recovery of precious metals. While the theory of electro-chemical action has been known for many decades it is only within the last few years that methods and machines for making a practical use of the theoretical knowledge have been devised.

A distinction should be made between electro-chemical amalgamation and electro-chemical lixiviation. In the first no solvent is used, and the values are recovered directly in the form of amalgam. In electro-chemical lixiviation the action of the solution is hastened by the presence of the electric current, and the values are deposited by electro-deposition on suitable cathodes: the cathode may be lead, copper, aluminium or mercury.

Electro-chemical amalgamation may be used in connection with (a) the standard mill plate, (b) mercury wells, or (c) the mercury bath (pan amalgamation). When the usual copper plate is used, anodes are suspended above it close enough to come in contact with the passing flow of water and pulp, which acts as an electrolyte. The copper plate is connected as the cathode

Electrolytic sluices with suspended anodes require more water than the usual mill plate. Mercury may be supplied as in the usual practice, or it may be supplied to the water in the form of a solution of bichloride of mercury. A solution of common salt added to the water greatly increases the efficiency of the device. High-grade graphite anodes give the most satisfactory results; there is a grade of graphite made especially for anodes which is superior to the ordinary carbon. A current density of one-tenth to one-quarter ampere per square foot of cathode surface is ample.

Mercury wells, such as are described in the second volume of Richard's work on Ore Dressing, can be connected electrolytically and their efficiency greatly increased; some tests were made some years ago in South Africa with mercury wells in which the baffle plates were connected as anodes, and I am informed that a similar construction is now being successfully used in California. There are several methods of constructing electro-chemical mercury wells, and many experiments have shown that the electrolytic mercury well is a very efficient gold saver; there is no fouling or flouring, and no scums or coating of sulphurets collects on the mercury. Electrolytic mercury wells will extract values from slimes and will recover gold that cannot be amalgamated by the usual methods; they require no skilled attention and possess many advantages over plates.

In pan amalgamation, anodes are suspended from the stirrers and the mercury is connected with the negative wire; various systems of pan amalgamation have been described especially in connection with cyanidation (Parks, Pelatan-Clerici, Molloy, Hannay, etc.) but there is no essential difference in these various processes.

Turning to electro-chemical lixiviation, the claim is made that under electro-chemical conditions the cyanide solution is more active than usual; values pass into the solution which are lost in the ordinary systems; the time of treatment is reduced and the extraction increased, while there is a decrease in the operating cost: precipitation by electro-deposition goes on simultaneously with the leaching process and the usual troubles are eliminated; a molecule of cyanogen after taking up a particle of gold, is released as nascent cyanogen, having greatly increased affinity for gold and silver. The solution is constantly regenerated, the loss of cyanogen being negligible. The larger particles of gold are amalgamated directly without passing into the solution. In electro-cyanidation the pulp and solution are agitated in a shallow tank, having the bottom covered with quicksilver (cathode): revolving stirrers cause the pulp to travel with a circular motion over the quicksilver for an hour or two; this is equivalent to passing the pulp over several miles of mercury surface. During this period the values are amalgamated directly or pass into the solution and are precipitated by electrolysis, the extraction averaging 97%. All ores, so far as known, are capable of successful treatment by means of electro-cyanidation.

Within the next few years, I expect to see the electro chemical cyanidation process perfected to such an extent that it will become automatic and continuous; pulp from the mill will continuously enter a series of tanks, and from the lower end of the system the tailings will be continuously discharged. In fact plans for such a continuous system have already been prepared.

At present electro cyanidation is in the experimental state, although it was once used quite successfully for several months at the De Lamar mine in Idaho; electro-deposition (Siemens-Halske process) is successfully employed in many mines in South Africa.*

Let us consider again electrolytic amalgamation, taking as a text the so-called base or rebellious ore. When low-grade pyritic ore is crushed to 100, 150 or 200 mesh, practically all the gold is released as free gold; but only a small percentage of the assay value can be recovered by the usual mill plate. Gold when in a very finely divided state quickly becomes coated with argillaceous, talcose, sulphurous, or arsenious coverings, and it will not amalgamate: very fine gold easily acquires coatings, gaseous or otherwise, which effectually prevent the amalgamation contact; values in slimes cannot be extracted by the usual mill plate: particles of gold in placer deposits sometimes become coated by local galvanic action, and we have rusty, silicious, or cupreous coatings which effectually prevent amalgamation. With electro-chemical conditions, all these difficulties are automatically overcome, and the broad statement can be made without fear of successful challenge that all values not encased are recovered, including platinum and the values associated with black sand.

In electrochemical amalgamation and cyanidation, a few pounds of common salt should be added to every ton of ore or pulp; by electrolysis the chloride of sodium is reduced to nascent chlorine and nascent sodium. Bichloride of mercury is also added to the water or solution; this salt under electrolytic action evolves nascent chlorine and mercury. As a result of these reactions, an alloy of nascent sodium and mercury is formed; this alloy absorbes nascent hydrogen produced by the decomposition of the water, and the resulting compound, known as hydrogen-sodium amalgam, has a powerful affinity for gold; in fact no compound known to science has a more powerful amalgamating action. The water and solutions become charged with nascent chlorine and hydrogen; these gases instantly destroy foul or greasy substances in the pulp; rusty or coated gold is quickly cleaned, the gases mentioned reducing all oxides. Nascent hydrogen reduces oxides at normal temperature. As a result of this cleaning and reducing action, gold particles have been freed from all substances that would prevent amalgamation, and amalgamation takes place at the first contact with the mercurial surface. Electro-chemical action keeps the mercury in a perfect state, and it also prepares the gold for amalgamation; in the mercury riffle or well we have the necessary fall to bring every particle of gold into intimate contact with the mercury surface. And as the quickened mercury surface is at all times exposed to the passing pulp, we have the most perfect conditions possible for effective amalgamation. We have therefore all the specifications which would be called for in an ideal amalgamation device—simplicity, low cost, ease of installation and operation, absence of skilled attendance, large capacity and high percentage of extraction.

In using the electrolytic system in placer and dredge operations, the auriferous material is at first passed through a series of screens until a 10 or 12 mesh concentrate is obtained; the coarser particles of gold will be saved by the usual riffle system. The 12 mesh concentrate, containing the values ordinarily lost, is then passed over the electrolytically connected plate or riffles of suitable length and 95% of all values not encased are recovered at a trifling expense. The successful application of the electrolytic system on gold dredges would double the net savings. Under the present system the fine tailings of the dredges

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[* This system is not now employed in South Africa. Ed. Com.]

generally contain greater assay values per ton than the amount of gold recovered; that is, on ground where 15 cents per yard is recovered it will generally be found that the assay value of the tailings is 25 or 30 cents per ton. In ten years I estimate that gold and platinum to the amount of $25,000,000 have passed through the tail sluices of Californian dredges and have been forever lost. Within the next ten years the value of the lost gold in the dredge tailings will exceed $40,000,000, unless some new system of successfully extracting these elusive values is discovered."—ELMER ELLSWORTH CAREY.—*Canadian Mining Journal*, Jan. 1, 1910, p. 18. (A. R.)

THE COMPUTATION OF CRUSHING EFFICIENCY.—"The rather lengthy contribution of Mr. C. O. Schmitt to the discussion on the paper 'The Computation of Crushing Efficiency of Fine Grinding Machines,' read by me a year ago before your Association, has hardly any connection with the purely theoretical subject dealt with, and contains very few material points which I consider worthy of consideration in my reply.

Very little will, therefore, remain to be said, as nearly the whole contents of Mr. Schmitt's contribution, tended to evoke the impression that in my paper the technical knowledge of Rand engineers had been belittled, and that the practical usefulness of tube-mills had been called into question. I cannot let this suggestion pass in silence.

Mindful of the words of Mr. Schmitt that 'theory should guide practice by showing the way to improvements,' I ventured, twelve months ago, to point out a weak point in our knowledge of a special line of engineering. Mr. E. J. Laschinger, the then President, in thanking me for the paper, said the statement made that the problems connected with grinding had not received that attention which other branches of the engineering profession had, 'was no less than a statement of fact,' and Mr. Caldecott mentioned that the suggestion of investigating the efficiency of tube-mills had already been discussed by various members of the Mines' Trials Committee. Shortly after, this certainly representative body of leading Rand engineers, recognising the *bona fides* of my statements, entrusted the writer himself with the undertaking of research work under the direction of sub-committee, composed of Messrs. S. H. Pearce, E. J. Laschinger, A. M. Robeson and W. A. Caldecott. This shows very clearly that Mr. Schmitt's position is opposed to that of other leading engineers.

To demonstrate the application and the practical value of formulæ evolved for the calculation of crushing efficiencies of grinding appliances, I compared, 'as an example,' the efficiency of a stamp with that of a tube-mill, and the conclusion then drawn from this calculation was, that '*from a purely mechanical point of view* tube-mills are very poor machines.' In quoting this statement, Mr. Schmitt omitted the words, 'from a purely mechanical point of view,' and added immediately after, this *may be true from a mechanical point of view*, but nevertheless, etc., and he then goes on to teach the Rand engineers that certain machines or methods of working, although of proved inefficiency, can in practice not be dispensed with. 'Why? Anyone can supply the reply,' are his own spirited words with which he winds up this kindergarten lesson.

Mr. Schmitt then proceeds to combat the data obtainable at the time when the paper was read with the figures of a recent test carried out at the Simmer Deep, Ltd., 'as the suggestion of Mr. Caldecott, following closely upon tests carried out some time ago on behalf of the Mines Trials Committee, and trusts I would not be 'horrified' to learn from him that to-day, 'owing to a considerable amount of further experience,' the right allowance of tube-mills provides for one tube-mill for every 15 to 20 stamps, as against 50 to 75 as before. I am rather 'horrified' that Mr. Schmitt carefully omits to mention that in consequence of coarser crushing, each of these 15 to 20 stamps has two to three times the capacity of the stamps working under the former condition with finer screening, and that he passes over in silence the fact that the progress made in tube-milling is precisely the outcome of the theoretical and experimental research work so strongly advocated in my paper, and in the meantime carried out by the Mines Trials Committee, may I say, with my assistance.

Crushing means reduction in size, respectively in volume, and the possibility of an exact size measurement of the materials dealt with before and after passing the machine is, therefore, the primary factor upon which the reliability of the computation of crushing efficiency of grinding appliances depends. The classification of the varying sizes obtained by sets of screens into series of standard grades, may be based on the largest *diameter*, on the *volume* (or weight), or on the *surfaces* (or fractures) exposed. It does not matter which of these functions of the particles be taken as a base for the classification into grades, as the final results of the calculations are bound to be exactly same for all classes, but once a system has been adopted we are compelled to stick to it by reducing all other functions to this same standard of measurement. It would, for instance, be utterly wrong to base calculations of crushing efficiency on the *surface*, and at the same time measure them by quite another rule based on the reduction in *volume*. Volume and surfaces vary with the coarseness of the particles at quite a different ratio and without converting the one into the other they cannot be measured by the same rule any more than the pressure in a steam boiler can directly be measured by degrees of temperature without conversion. Considering that after crushing the further ore-treatment by amalgamation or by cyaniding has to act through the surfaces, it has been suggested that the exposure of these surfaces, representing the 'metallurgical efficiency,' should be taken as a criterion of the useful work done in crushing. Under the condition that the work done represented by the exposure of surfaces is also measured by a standard of grades, based on equality of steps in regard to these surfaces, no other objections than those of practical inconvenience can be raised against this suggestion. However, as cyanide managers do not trouble about the surfaces exposed, and as also the assayers give the percentages of gold extraction per weight (volume) and not per fractures exposed, there is no reason why, in computing crushing efficiency, the reduction in volume should not also be accepted as representative for the metallurgical efficiency.

The most rational and logical system of classification for mining purposes is, therefore, undoubtedly to base it on the reduction of the volume of the particles, and with this view a table has been annexed to my paper, containing the theoretical values of the functions of a series of cubes determined by the diameter of the mesh aperture, and forming a graduated scale, based on a successive reduction in volume by one half from grade to grade.

It appears that the standard screens of the I. M. and M., in London, follow in regard to the mesh apertures the same curve of seriation, but unfortunately in consequence of unreasonable concessions made to the 'well accustomed' number of meshes and to the maintaining of the uniformity of the screen discharge area: the distance between the screens, or, better, the decrease of the grades in regard to the volume is very irregular and not based on any principle. By chance, however, it was possible to select from the very large number of these standard screens a restricted set with apertures corresponding fairly well to the mathematically correct scale, and complying, at the same time, with the requirements of equality of steps from grade to grade in regard to the reduction in volume (1 : 4).

The possibility of an accurate measurement of grinding products being ascertained, the next question arises of establishing from the gradings taken at the inlet and at the discharge of the machine a workable and reliable formula for the computation of the useful work done in crushing by the machine.

Even before the first tin shanty was erected on the Rand, technological and mechanical books referred to Rittinger's theory of the proportionality of the work done to the surfaces exposed by introducing it with the words: 'It was hitherto believed.' This question has been fully dealt with before another technical society, and would not have been taken up here again had not Mr. Laschinger, in reviewing some points of my paper, dropped the remark that Rittinger's law might be correct in case of pure shear. I cannot admit any exception, and, on the contrary, the punching of holes, for instance, affords a very good example of showing where the error of Rittinger's contention comes in. Supposing that a hole of a certain diameter is to be punched in an iron sheet of a thickness 'a,' and a second hole of double the diameter in another sheet of the thickness $a \div 2$: the area over which the adhesive power of the molecules has to be destroyed by shearing is then for both holes the same, and also the *force* required to produce this destruction, but in the case of the thicker sheet, the punch will have to run through a longer distance, or, if you like, the force has to act during a longer time, and the *work done* is, therefore, *not* proportional to the area of fractures but varies as to the thickness of the sheets.

By similar deductions it has been shown in my paper that the amount of work required to reduce the equivalent of 1 cubic inch from one grade to the next following, is constant, and the ordinal numbers of the grades represent, therefore, at the same time the relative values of the energy which had to be spent in reducing the unit of 1 cube inch to the respective grade. To obtain the mechanical value of *mixed sands* we need only multiply the percentages of the gradings with the number of these energy or crushing units of the respective grade and add the products. The relative value of the work done by any crushing machine is naturally determined by the difference between the mechanical values of the pulps at the inlet and discharge of the machine, and the relative mechanical efficiency is then the product of this value into the tonnage dealt with and divided by the value of the energy consumption.

In practice this method, based on Kick's law, has given great satisfaction, and all through the research work which I had to execute on behalf of the Mines Trials Committee, in tube milling, etc., it could be ascertained that this method is for coarse as well as for fine crushing, correct and reliable, and in perfect accordance with the results of practical experience. Based on this fact and backed by authorities, I have no hesitation in stating that with this method *we are now in a position to determine, with a high degree of accuracy, the relative merits of different crushing appliances or the mechanical efficiency of one and the same machine working under varying conditions.*

It would be going outside the scope of my paper to deal with results and facts obtained by the application of this method, and I only mention here that by the research work other weak points connected with tube-milling have been revealed and that even now there remains still ample room for further great improvements, especially in connection with the regulating and maintaining of the recognised most suitable conditions of the feeding."—H. STADLER.—*Journal of the South African Association of Engineers*, Jan., 1910, p. 141. (H. A. W.)

NOTE ON BULLION REFINING.—"The sulphur method of refining bullion dates from early times: its use now is probably exceptionally limited, and, perhaps, it is worthy of greater application. Recently the writer, having only the crudest appliances at his disposal, wished to raise the fineness of about 600 oz. of bullion assaying about 250 fine, the remainder being principally zinc and lead. The whole of the bullion was melted and finely granulated in the usual way by pouring into water. The granulated bullion, while still wet, was mixed with about one-half of its weight of powdered sulphur, and put into large clay pots in a dead fire; the temperature was rapidly raised to above the melting point of gold. The pots were then removed from the fire, and, when cool, were broken, the resulting bullion buttons were re-melted, stirred when melted with an iron rod to free them from sulphur, and poured into a bar. The bar assayed rather over 850 fine in gold. The matte, which was on top of the bullion buttons—which, as previously stated, resulted from melting the granulated base bullion with the sulphur—contained very little gold, and was cleaned with a very little iron in the usual way.

The same method of producing fair grade bullion was also used on cyanide gold slime, in a case where no manganese dioxide was available, and no drying furnace was available which would give anything approaching calcination: the slimes dried after acid treatment were accordingly fluxed with the necessary quantity of sand and borax, and, in addition, there was added a quantity of sulphur equal to about $\frac{1}{30}$ (or about 3%) of the weight of the gold slimes. The bullion obtained was just under 900 fine in gold: the slags and matte are handled in the usual way, and their gold and silver contents were normal.

It therefore appears that in out-of-the-way camps away from modern facilities the method might at times be of considerable utility."—H. T. DURANT, M.I.M.M.—*The Mining Journal*, Jan. 15, 1910, p. 53. (W. A. C.)

ERRORS DUE TO THE PRESENCE OF POTASSIUM IODIDE IN TESTING CYANIDE SOLUTIONS FOR PROTECTIVE ALKALINITY.—"The results of experiments recently made by me show that potassium iodide influences the testing for 'protective alkalinity' in cyanide solutions. As far as I can ascertain, this fact is not mentioned in standard text books now in use.

The experiments which I carried out show the influence exerted by potassium iodide to be important, especially with regard to cyanide solutions which contain no protective alkali, for in testing these in the presence of potassium iodide they show protective alkalinity.

Cyanide decomposes much faster in solutions which are deficient in protective alkalinity than it does when protected by an alkali; therefore, the method

I give of testing without potassium iodide will mean a considerable saving of cyanide on large cyanide plants, as the method generally used is to test the cyanide and protective alkalinity in the one measured portion.

In Clennell's "Chemistry of Cyanide Solutions" (p. 63), he says: . . . . So that the same measured portion of the liquid to be tested will serve for the determination of both cyanide and protective alkalinity."

The method which I have used for these tests is as follows:—

Into 10 c.c. of the test solution, using potassium iodide as indicator, run standardised silver nitrate until a deep brown coloration of iodide of silver shows clearly, and note number of c.c. of silver nitrate taken.

Now using another 10 c.c. of the test solution, without adding potassium iodide, run in twice the quantity of silver nitrate that was necessary for the above test, to make sure of getting rid of the alkalinity due to cyanide.

Add phenolphthalein and titrate with standardised acid. The result will be protective alkalinity.

A second method I have used, which I consider gives clearer results than the foregoing, and which also does away with the first part of the previous test, is as follows:—

Into the measured portion of test solution, to which has been added the indicator, phenolphthalein, run in silver nitrate, until the indicator colour remains constant, then titrate with standardised acid in the ordinary way. With a little practice this gives very good results.

These methods (potassium iodide being absent) will immediately show any deficiency of protective alkalinity, and this deficiency may be easily remedied, and so stop rapid decomposition of cyanide by the addition of a suitable alkali.

The chemicals I used for these experiments were:

Commercial sodium cyanide ...	(126%)
Pure potassium iodide ...	(10% solution)

The standardised solutions, using 10 c.c. of test solution, were:—

Silver nitrate ...	1 c.c. = 0·01%	KCN
Hydrochloric acid...	1 c.c. = 0·001%	Alkalinity
Caustic soda ...	1 c.c. = 0·001%	Acid.

Indicator, phenolphthalein.

For the tests a solution of the above-mentioned commercial cyanide was made up showing 0·117% cyanide, 10 c.c. of which were used for each test.

In the following table, 10 c.c. of the test solution were taken for each test, and tested for protective alkanity by the first of the foregoing methods. The results were:

	Protective Alkalinity	
	per cent	
I. Without adding KI	0·0033	Clean end reaction
II. With $\frac{1}{2}$ c.c. KI ..	0·0046	Indistinct ..
III. ,, 1 c.c. KI .	0·006	.. ..
IV. ,, 5 c.c. KI .	0·0095	.. ..

It will be noticed by the above table that the error in testing protective alkalinity increases with the increase of potassium iodide.

In the next table the same test solution was used as in that previous, but each 10 c.c. used was made slightly acid as regards protective alkalinity by adding 3·5 c.c. standard hydrochloric acid; 10 c.c. of the acidulated test solution were taken for each experiment and tested for protective alkalinity by the method given, when the following results were obtained:

	Protective Alkalinity	
	per cent	
I. Without adding KI	Nil	Clean end reaction
II. With $\frac{1}{2}$ c.c. KI .	0·002	Indistinct ..
III. ,, 1 c.c. KI	0·0037	.. ..
IV. ,, 5 c.c. KI	0·0068	.. ..

The results of the foregoing table prove that solutions which contain no protective alkalinity give tests for protective alkalinity after the addition of potassium iodide.

From these experiments it will be seen that potassium iodide must be absent when testing cyanide solutions for protective alkalinity.

I have obtained similar results from ordinary working cyanide solutions." — BEDE COLLINGRIDGE. *Institute of Mining and Metallurgy, Bulletin No. 64*, Jan. 13, 1910. (A. R.)

CYANIDING SILVER ORE IN HONDURAS. "At San Juancito, Honduras, the New York and Honduras-Rosario Mining Company has recently changed the ore treatment from pan-amalgamation and concentration to an all-slime cyanide system, which has proved successful.

The ore is a hard quartz containing gold and silver, and some lead, copper, iron, and antimony.

The ore passes over grizzlies, the coarse being sorted and then crushed to about 1-in. size in the usual way. The wash-water carries a large amount of slime in suspension, and to recover it, lime is added, the water being settled in vats. The overflow carries away a very fine and light slime, which is collected at the mill and treated apart.

There are fifty stamps, weighing 750 lb. each. The drop is 6 in., 100 to 102 times per minute. The mortars are of the double-discharge type with 30 mesh steel wire-cloth screens. The duty is 2 tons per head per twenty-four hours. The ore is crushed in cyanide solution, the pulp flowing in wooden launders to a 4-in. centrifugal pump, which elevates it to three conical classifiers. The coarse sand goes to two tube-mills, the slime overflowing from the classifiers to two circular-wooden settling vats with peripheral overflow. The clear solution is pumped to the mill tank, from which it passes by gravity back to the batteries. The sand from the tube-mills is returned to the classifiers, the final product, which consists of material of which 90% will pass a 200-mesh screen, going to the cyanide plant. About every hour the slime in the settling vats is drawn off and discharged by gravity into a wooden launder, which conveys it to the same plant.

The cyanide plant consists of seven circular wooden agitator vats, each of 50 tons, and two receiving vats, each of 90 tons capacity of dry slime. The pulp flows into one of the latter, and cyanide is added, until, when filled, the solution has a 0·2% strength. The pulp is then agitated in the former vats from forty to sixty hours. Aeration is accomplished by circulating the pulp with a 6-in. centrifugal pump one hour, twice a day.

Samples of both pulp and solution are taken from the agitator vats twice daily. These are filtered, the solution is titrated, and, if necessary, cyanide is added by suspending a perforated tin can filled with this salt in the pulp.

The very fine slime from the washing is settled in stone tanks below the mill, where it is de-watered, partly sun-dried, and returned to the mill. There it receives a preliminary treatment by agitation in solution in pans and settlers before going to the cyanide plant with the mill slime.

For filtering, a sixty-leaf Butters vacuum slime filter is used. There are two filter vats above the filter box. The pulp from the agitator vats is pumped to one of the former, from which it flows by gravity to the latter. When the filter leaves are covered, the vacuum pump is started and clean solution drawn out until a cake of the thickness of $\frac{3}{4}$ to 1 in. has formed on the filter leaves. A valve in the suction pipe is then opened, reducing the vacuum treatment 5 in., which is found to be sufficient to harden the cake on the leaves. The surplus pulp is pumped out of the filter box, returned to the vat, and the filter box is re-filled with barren solution. The valve is then closed, and the cake is given a 40-minute solution wash, which is followed by a 20 minute water wash, after which the vacuum pump is stopped, water is admitted to the filter leaves, the cake drops off, and is discharged through a 10-in. pipe from the bottom of the filter box. The charge varies from 12 to 18 tons, and is handled in two hours by one native workman per shift.

Precipitation is by zinc shavings in six-compartment boxes. The zinc in the head-box is packed and replenished from the next box daily, the zinc being moved up, and new zinc added to the lower box. The boxes are cleaned out every three or four days. The precipitate and the solution from the boxes flow in launders to the sump vat, into which they are washed through a 40-mesh screen. At intervals the agitator in the vat is set in motion, the pump is started, and the precipitate and solution pumped through a small 40-frame filter press.

The moisture in the precipitate is reduced to 3% or less in a drying furnace, and is then spread on a cement floor. The entire lot is passed over a $\frac{1}{8}$-in. mesh screen, after which it is weighed into lots not exceeding 770 lb. The different lots are carefully sampled, put into oiled canvas sacks, which are boxed and shipped to New York.

The plant produces about 4 tons of precipitate per month. Four labourers and one shift-boss attend to the zinc boxes, cleaning up, drying, sacking, and boxing of precipitate. The solution entering the zinc boxes contains an average of 14 oz. silver and 0·12 oz. gold. The outgoing solution contains a trace of gold and about 0·08 oz. silver.

In six months, ending Oct. 31, 1908, 14,529 tons of ore, averaging 40·05 oz. silver and 0·467 oz. gold were treated. The tailings averaged 3·33 oz. silver and 0·019 oz. gold. The difference between the actual and theoretical extraction was less than 1%. The cost of materials, including zinc, cyanide, lead-acetate, hydrochloric acid, and lime was slightly under $2 gold per ton. The large amount of cyanide (nearly 8 lb. per ton of ore) consumed was principally due to moisture in ore and slime (at times over 10%); to cyanide consumed in dissolving the precious metals; to the presence of cyanicides in the ore, the most active of which are antimony and copper; and to loss in precipitation (about $\frac{1}{2}$ lb. cyanide per ton of solution passing the zinc boxes)."—GEORGE E. DRISCOLL.—*London Mining Journal*, Jan. 29, 1910, p. 105. (A. R.)

MEASUREMENT OF PULP AND TAILING. II.—"In ascertaining the weight of solid in a vat filled with uniformly liquid pulp, such as slime in an agitator, a cubic foot or any convenient measured volume may be dried and the residue weighed, whence the weight in the entire volume is obtained by proportion. If the mixture is weighed before drying the percentage of solid in the pulp may also be found.

A much easier and more rapid method is to find the specific gravity of the pulp, either with a hydrometer or by weighing a litre or other convenient volume. The density of the dry solid must also be known, at least approximately. Knowing these two values: specific gravity of mixture (p) and density of dry solid (d), we can at once calculate the weight of dry solid per cubic foot, as well as a number of other factors and relations, by means of the data in Tables I. and II., or we may read them directly from a curve (Fig. I. at end of this article). The volume of pulp in cubic feet and the weight of solid per cubic foot, or the volume in fluid tons and the 'solid factor' (F: in Table I.) enables us to calculate the tonnage by a single multiplication.

For example, consider an agitator of 16 ft. diameter by 45 ft. deep, containing 7·200 cub. ft. or 225 fluid tons of pulp the gravity of which is found by hydrometer to be 1·182, while the density of the dry slime is known to be approximately 2·6. Substituting these values for p and d in the formula for F in Table I. we find the value of F to be 29·575 tons solid for 100 fluid tons, whence 225 fluid tons contain 66·5 tons. Or we may substitute the same values in the formula for S, the percentage of dry solid, which is thus found to be almost exactly 25%. Hence again $F = 25 \times 1·182 = 29·55$, which leads to almost exactly the same result. Also inspection of the curve for F in Fig. 1 shows at once that the solid factor for gravity 1·182 and density 2·6 is approximately 29·5.

If again, instead of determining gravity, we measure out exactly a cubic foot of the pulp, settle and siphon off the solution and dry the residue, we find it weighs 839 grm. or 1·85 lb. Then $7,200$ cub. ft. $\times 1·85 \div 2,000 = 66·6$ tons.

Where frequent determinations are required of dry tonnage in vats containing slime or ore of nearly uniform character (d = constant) much time is saved by drawing up a table with vertical columns corresponding to depth of pulp, and horizontal lines representing gravity of pulp, the total tons of solid being given at the intersections. A mark can also be established for a given number of fluid tons with a correction for each inch of variation in depth, the volume of pulp thus found being then multiplied by the solid factor corresponding to the observed gravity. A portion of a table used for several years in connection with such a datum mark is given in a later article. This enabled a workman, in less than five minutes, to determine within 2 tons the amount of slime in an agitator containing 100 to 150 tons.

In deep vats, or when agitation is imperfect, samples should be taken at several levels—say near top, middle and bottom—and the contents corresponding to the gravity found for each layer should be computed separately and then averaged. A less accurate result is generally obtained by using for the computation an average of the three gravity readings. Such layer samples can be drawn off by means of wooden beer-cocks, or simply from half inch holes closed by wooden plugs. Another method of taking them is to attach a bottle or can to a graduated stick, a string fastened to the cork being pulled at any desired level in order to sample the mixture at that particular depth.

The relations between the specific gravity of a pulp and the percentage of solid and various other factors have been worked out in some detail, and are given in Table I. annexed. Table II shows further in a simplified manner the relations found most useful in practical calculations, in forms which admit of fairly rapid computation by slide-rule or otherwise, if advantage is taken of the constants in Table III. and the relation of water ratio to percentage given in Table IV. For very many purposes the various values can be read with sufficient accuracy from a curve like Fig. 1, or interpolated from suitable tabulated data.

These formulae apply generally to all cases of water mixtures or emulsions, including slime and mill-pulp. The specific gravity of

TABLE I.—Pulp Formulae.

d = density or specific gravity of dry solid (ore, sand or slime).

p = specific gravity of pulp (mixture of water and ore, etc.).

S = percentage by weight of dry solid in pulp.
= grams in 100 grm., tons in 100 tons weight, etc.

R = water ratio of pulp.
= tons water per ton of dry solid.
= grams water per grm. of solid.

V = volume-percentage of solid in pulp.
= c.c. in 100 c.c. in pulp
= cubic feet in 100 cub. ft. pulp.

F = solid factor.
= grams solid in 100 c.c. of pulp.
= tons solid in 100 fluid tons or 3,200 cub. ft. of pulp.
= avoirdupois ounces in 0·1 cub. ft. of pulp.

k is a constant for any particular solid under consideration, used to facilitate calculation, and depending upon the density of the dry solid.

$$k = \frac{100\,d}{d-1}$$ (See Table III. for values of k.)

$$d = \frac{p}{1 - R\,(p-1)} = \frac{S\,p}{S\,p - 100\,(p-1)}.$$

$$p = \frac{R+1}{R + \frac{1}{d}} = \frac{100}{100 - \frac{S\,(d-1)}{d}} = \frac{k}{k - S}$$

$$S = \frac{100}{R+1} = \frac{100\,d\,(p-1)}{p\,(d-1)} = \frac{k\,(p-1)}{p}$$

$$R = \frac{d-p}{d\,(p-1)} = \frac{1 - \frac{p}{d}}{p-1} = \frac{100-S}{S}$$

$$= \frac{100}{S} - 1 = \frac{100\,p}{k\,(p-1)} - 1$$

$$F = S\,p = \frac{100\,p}{R+1} = \frac{100\,d\,(p-1)}{d-1} = k\,(p-1)$$

$$V = \frac{F}{d} = \frac{S\,p}{d} = \frac{100\,(p-1)}{d-1} = (k - 100)\,(p-1)$$

Volume-percentage of water in pulp

$$= 100 - V = 100 - \frac{F}{d} = p\,(100 - S)$$

$$= \frac{100\,(d-p)}{d-1}$$

TABLE II.—Water and Solid in Pulp.

	Weight Pulp.	Volume Pulp.	Weight Dry Solid.	Weight Water.	Volume Water.
One ton of pulp (2,000 lb.)	1 ton	$\frac{1}{p}$ fluid ton $\frac{32}{p}$ cub. ft.	$\frac{S}{100}$ ton $\frac{1}{R+1}$ ton $\frac{(p-1)k}{100p}$ ton	$\frac{d-p}{p(d-1)}$ ton	$\frac{d-p}{p(d-1)}$ fl. ton $\frac{32(d-p)}{p(d-1)}$ cub. ft.
One fluid ton of pulp (32 cub. ft., 240 U.S. gallons, 200 Imperial gallons)	p tons	1 fluid ton 32 cub. ft.	$\frac{(p-1)k}{100}$ ton	$\frac{d-p}{d-1}$ ton	$\frac{d-p}{d-1}$ fluid ton $\frac{32(d-p)}{d-1}$ cub. ft.
One cubic foot of pulp (7·5 U.S. gallons, 6·25 Imperial gallons)	62.5 p lb.	1 cub. ft. 0·03125 fluid ton	0·625 $(p-1)k$ lb. $\frac{(q-1)k}{3,200}$ ton	$\frac{62{\cdot}5(d-p)}{d-1}$ lb. $\frac{d-p}{32(d-1)}$ ton	$\frac{d-p}{d-1}$ cub. ft.
One ton of dry solid (2,000 lb.)	$R+1$ tons $\frac{100p}{(p-1)k}$ ton	$\frac{100}{(p-1)k}$ fl. ton $\frac{3,200}{(p-1)k}$ cub. ft.	1 ton	R tons $\left(\frac{100}{S} - 1\right)$ ton $\frac{d-p}{d(p-1)}$ ton	32 R cub. ft. R fluid tons.
One fluid ton of water (32 cub. ft., 240 U.S. gallons, 200 Imperial gallons)	$\frac{p(d-1)}{d-p}$ ton	$\frac{p(d-1)}{d-p}$ fl. ton	$\frac{1}{R}$ ton $\frac{d(p-1)}{d-p}$ ton $\frac{S}{100-S}$ ton	1 ton	1 fluid ton 32 cub. ft.

One U.S. gallon pulp = $\frac{(p-1)k}{12}$ lb. dry solid. One Imperial gallon pulp = $\frac{(p-1)k}{10}$ lb. dry solid.

Metric Values.—The relation between the metric ton (of 1,000 kgrm.) and the cubic metre (1,000 litres) is identical with that existing between the ton of 2,000 lb. and the fluid ton of 32 cub. ft. For instance, 1 cub. metre of pulp contains 0·01 $(p-1)k$ metric ton dry solid.

Tons of dry solid per 100 tons water

$$\frac{100}{R} = \frac{100\,S}{100-S} = \frac{100\,d\,(p-1)}{d-p}$$

Fluid tons pulp to yield one ton solid

$$\frac{100}{F} = \frac{100}{k\,(p-1)} = \frac{R-1}{p}$$

Water is taken as 1 and no corrections are made for temperature, air-buoyancy, etc. They are applicable to ordinary mixtures of cyanide solutions with slime, etc., except for very small percentages of solid, but do not apply to mixtures with sea water (sp. gr. 1·03), nor to heated or highly alkaline waters, except for purposes of rough approximation.

Very Thin Pulp.—For excessively thin mixtures, such as turbid water, or the overflow from efficient clarifying tanks or dams, and for all pulp of gravity less than 1·01, the gravity must be determined with the utmost care. When the percentage of solid is so very low its influence on the gravity may become less than that of ordinary variations of temperature, or of small proportions of dissolved salts. In such extreme cases, or where the proportion of dissolved salts is considerable, it is best to determine the percentage of suspended solid by filtering a measured quantity through an ashless filter-paper, drying and brushing off the residue, burning the paper and weighing the ash and residue together. The bulk of the solid must not be ignited with the paper or the result will be low, owing to decomposition or sulphides, carbonates, or hydrous silicates.

TABLE III.—Values Depending on Density of Dry Solid in Mixture.

Density of Dry Solids d	Values of $k = \frac{100\,d}{d-1}$	Values of 27 k	Values of $\frac{100}{k} = \frac{d-1}{d}$
2·2	183·3	4,950	0·545
2·3	176·9	4,780	0·565
2·4	171 4	4,630	0 583
2·5	166 7	4,500	0·600
2·6	162·5	4,390	0·615
2·7	158·8	4,290	0 630
2 8	155·5	4,200	0·643
2·9	152·6	4,120	0·655
3·0	150·0	4,050	0 667
3·1	147·6	3,985	0·677
3·2	145 5	3,930	0·687
3·3	143·5	3,875	0·697
3·5	140·0	3,780	0 714
4·0	133·3	3,600	0·750
5 0	125·0	3,375	0·800
6 0	120 0	3,240	0·833

It is noteworthy that in the case of extremely thin pulps, having gravity under 1·05 or weighing less than 66 lb. per cub. ft., the values of S, F, and A (and also of "sluicing efficiency" where this is applicable) are nearly identical and approach closely to the common value $k\,(p-1)$, which can be used to calculate them without serious error, provided the gravity of the pulp be determined with sufficient accuracy. We have then:

For values of p under 1·05
- S or percentage by weight of dry solid
- F or tons solid in 100 fluid tons of pulp
- A or tons dry solid in 100 tons of water and sluicing efficiency where applicable)

$= k\,(p-1)$ approximately;

the approximation being closer as the gravity approaches unity.

This approximation is often close enough for the overflow of certain slime-settlers, spitzkasten, cones, etc., but must be used with due caution.

Sluicing Wet Material.—When wet sand is sluiced out of treatment-vats and collecting-tanks, or slime-cakes from filter-presses, etc., by means of a stream of water or solution, calculations are complicated by the fact that the resulting mixture contains

TABLE IV.—Corresponding Values of Percentage and Water Ratio.

Percentage Dry Solid S or C	Water Ratio R or M	$\frac{1}{R}$	Percentage Dry Solid S or C	Water Ratio R or M	$\frac{1}{R}$
1	99·0	0·0101	33·33	2·0	0·500
2	49·0	0 0204	35·0	1 857	0·5385
3	32·33	0·0309	36·36	1·75	0·5714
4	24·0	0·0417	40·0	1·5	0·6667
5	19·0	0·0526	44·44	1·25	0·800
6	15·67	0·0639	45·0	1·22	0·8181
6·25	15·0	0·0667	50·0	1·0	1·00
6·67	14 0	0·0714	57·143	0·75	1·333
7	13·286	0·0753	60·0	0·667	1·50
7·69	12·0	0·0833	62·5	0·60	1·667
8	11·5	0·0870	65·0	0·538	1·857
8·33	11·0	0·0909	66·67	0·50	2·00
9	10·101	0·0990	70 0	0·428	2·333
9·09	10·0	0·100	75 0	0·333	3·0
10	9·0	0·1111	80·0	0·25	4·0
11	8·091	0·1236	83·3	0·20	5·0
11·1	8·0	0 125	85 0	0·176	5·66
12	7·333	0·1363	85·7	0 167	6·0
12·5	7·0	0·1428	87·5	0·143	7·0
14·28	6·0	0·1667	88·89	0·125	8·0
15	5·67	0 1765	90·0	0·111	9·0
16·66	5·0	0·200	90·91	0·10	10·0
20	4·0	0 250	95 0	0·0526	19·0
22·22	3·5	0 2857	95·28	0·05	20·0
25	3·0	0·3333	98 04	0·020	50·0
28·57	2·5	0·400	99·01	0·01	100·0
30	2·333	0·4286			

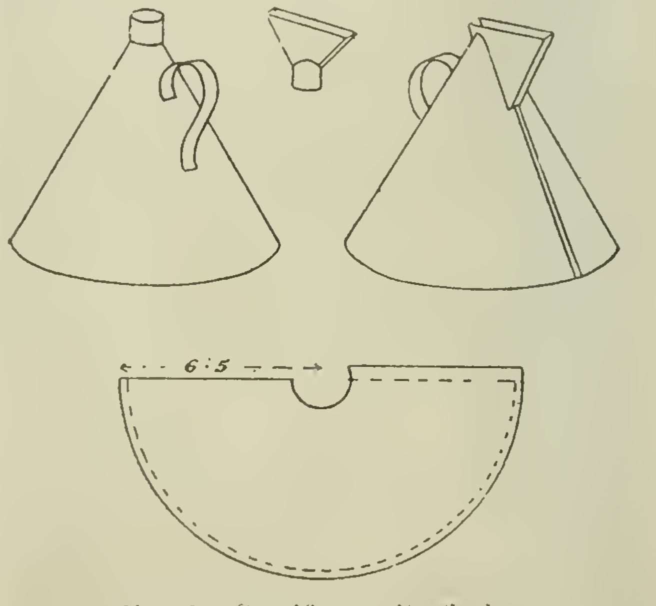

FIG. 2.—Specific gravity flask.

water from both sources. The additional data necessary for computing the proportion of added water to solids, etc., based on measurements of the final mixture, are given in Table V. Many other problems dealing with the addition to pulp of further quantities of water or solution can be solved by application of the same data in conjunction with the preceding tables.

TABLE V.—Formulae for Sluicing Moist Slime-Cakes and Residues.

C = percentage of dry slimes in cakes or of sand in drained residues, etc., prior to sluicing.

M = water ratio for material sluiced prior to sluicing.

$$= \frac{\text{percentage of water}}{\text{percentage of dry solid}}$$

$$= \frac{100 - C}{C} = \frac{100}{C} - 1.$$

E = net sluicing efficiency = tons of dry solid moved by 100 tons of added sluicing water.

R and S = water ratio and percentage of solids in the resulting pulp (see Tables I. and II.)

Q = tons sluicing water required per ton of dry solid in material sluiced, in order to yield a product of percentage S or water ratio R.

$$E = \frac{100}{R - M} = \frac{C - S}{C S}$$

$$Q = R - M = \frac{100}{S} - \frac{100}{C} = \frac{100(C - S)}{C S} = \frac{100}{E}$$

Tons added water per gross ton of wet material sluiced

$$= \frac{Q C}{100} = \frac{C(R - M)}{100} = \left(\frac{C}{S} - 1\right) = \frac{C}{E}$$

Density of Dry Solids.—The density of dry sand or slime is best found by means of a weighing-bottle. The weight of the bottle full of water at 15° to 20° C. (60° to 70° F.) is found once for all. A quantity of the wet sand or slime is then put in the bottle and covered with water. Air-bubbles are shaken out as far as possible and finally removed by suction: the bottle is then filled with water and weighed again. Finally it is emptied, rinsed out, and the sand or slime is collected, dried thoroughly, and weighed. Density (d) =

$$\frac{\text{Weight of dry sand}}{\text{Wt. bottle with water} + \text{Wt. dry sand} - \text{Wt. bottle full of sand and water.}}$$

If the sand is first dried and weighed, and a definite quantity weighed out for the test, the result is usually too low unless the mixture is boiled or given a long soaking. This has been fully discussed by Caldecott,[*] who seems to have been the first to note the difference. The result should be calculated to the second place of decimals.

It is important that the sand be well dried before weighing, as any residual moisture lowers the apparent density. If the true density is d and the material is weighed when containing n per cent. of moisture, the apparent density will be:

$$\frac{100\,d}{100 + n\,(d - 1)}$$

For instance, slime having a true density of 2·5 would, if finally weighed when containing 2% moisture, show an apparent density of $\frac{250}{103} = 2{\cdot}425$; similarly with an ore of density 3·5, the presence of 2% of water would reduce the apparent value to $\frac{350}{105} = 3{\cdot}33$

The size of bottle appropriate depends on the sensitiveness of the balance used: one of 25 to 50 c.c. capacity answers with an analytical or good pulp balance, one of 200 to 1,000 c.c. is suitable with scales only sensitive to a decigram. The tin litre flask described below can be used on a balance turning with a single gram, if a large enough weight of material is taken, say, at least 200 grm. of dry sand.

Estimation of Specific Gravity of Pulp: Hydrometer.—The hydrometer can be used in slime pulp when it is not extremely thick. It fails when the material settles rapidly, first on account of the upper stratum of liquid losing much of its suspended solid during the reading, secondly on account of particles settling on the bulb of the instrument. The glass instruments having a range from 1·00 to 1·50 can be used but it is preferable to obtain several of smaller range—say 1·0 to 1·2, 1·2 to 1·4, etc. For thin slime-pulp the very sensitive form known as the urinometer (range 1·0 to 1·060) can be used with advantage. A cylinder of galvanised iron is best substituted in the mill for the usual glass cylinder. Metal hydrometers might be used but the graduations are less easily read. It is advisable to test the accuracy of the instrument by immersion in water (sp. gr. 1·0) and in some heavy liquid, such as strong brine, the specific gravity of which has been determined by means of a standardized-hydrometer or by accurate weighing in a flask.

Flask.—The specific-gravity flask affords another rapid method, a measured volume of pulp being weighed. Glass flasks are satisfactory in the laboratory, but for mill-work it is best to use one of metal. A pattern that answers well is a conical vessel of tinned iron plate made to hold exactly 1,000 c.c. If of 60° angle the internal diameter at bottom must be about 6·5 in. and the side the same (see Fig. 2). For sampling uniform pulp the mouth is made small and round. For sampling a stream with a tendency to segregation it is best to add a narrow rectangular mouth piece or a similar funnel, the slot being long enough to cover the full horizontal cross section of the falling stream. The final adjustment of volume is made by trimming the mouth piece to a horizontal plane and 'peaning' the bottom with a small hammer, until the water contained when exactly full is within ½ grm. of 1,000 grm. A tare weight is made to balance the flask when full of water, additional grm. weights then indicate the fractional part of the gravity ($p - 1$). A balance weighing to one grm., such as Robervahl or trip scale, is used with this; it is not usually necessary to weigh closer than to the nearest 5 grms., giving the gravity to within 0·005. When not in use the flask is rinsed out and inverted to avoid accumulation of sand.

Cubic Foot Bucket.—For approximate work a bucket holding exactly a cubic foot, or up to 3 cub. ft., may be used, being weighed on a platform scale. It should be cylindrical, of stout sheet iron, with a heavy bail and stiffened rim. The larger sizes require two handles. A convenient size for 1 cub. ft. capacity is 14 in. diam. inside and 13 in. deep: a cubic foot of water fills this to about 11·25 in. from bottom. By making it 20 in. deep 100 lb. water can be accommodated, the level then being about 18 in. from bottom. Graduation is effected by weighing 62·5 or 100 lbs. water, boring a small hole at the surface mark and inserting a rivet or small bolt. A narrower form, say 12 in. diameter and 17 in. depth for 1 cub. ft., admits of more accurate measurement, but the additional height is inconvenient. A small tare weight should be made to fit the hanger of the platform scale, balancing the empty bucket. The

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[*] See this *Journal*, Vol. II., p. [illegible].

Table VI.—Pulp-Values.

For pulp ranging in specific gravity up to 1·60, and for solids ranging in density from 2·4 to 3·0.

	Density of Solid.	Factor.	Specific Gravity of Pulp = p = Hydrometer Reading.														
			1·01	1·03	1·05	1·07	1·10	1·15	1·20	1·25	1·30	1·35	1·40	1·45	1·50	1·55	1·60
Values of S = percentage by weight of dry solid in pulp $\frac{k(p-1)}{p}$	d	k															
	2·4	171·43	1·70	5·00	8·16	11·21	15·58	22·8	28·57	34·28	40·3	44·5	49·0	53·4	57·14	60·83	64·28
	2·6	162·5	1·61	4·73	7·74	10·63	14·77	21·2	27·08	32·50	37·5	42·1	46·5	50·4	54·13	57·66	61·00
	2·8	155·55	1·54	4·53	7·41	10·17	14·14	20·3	25·92	31·11	35·9	40·4	44·4	48·3	51·85	55·13	58·33
	3·0	150·0	1·485	4·37	7·15	9·81	13·64	19·56	25·00	30·00	34·6	38·9	42·9	46·6	50·00	53·22	56·25
Values of R = ratio of water to dry solid by weight $\frac{d-p}{d(p-1)}$	2·4		57·9	19·0	11·25	7·91	5·42	3·48	2·50	1·917	1·526	1·250	1·042	0·880	0·750	0·644	0·555
	2·6		61·2	20·1	11·9	8·40	5·77	3·72	2·69	2·075	1·666	1·373	1·154	0·983	0·846	0·733	0·611
	2·8		64·0	21·1	12,5	8·83	6·07	3·92	2·86	2·215	1·786	1·480	1·250	1·071	0·929	0·811	0·714
	3·0		66·3	21·9	13·0	9·18	6·33	4·11	3·00	2·333	1·889	1·572	1·333	1·148	1·000	0·878	0·778
Values of F = solid factor = tons dry solid in 100 fluid tons = grams solid in 100 c.c. $= k(p-1)$ *		k															
	2·4	171·43	1·714	5·143	8·571	12·0	17·14	25·71	34·28	42·86	51·43	60·0	68·57	77·14	85·71	94·28	102·8
	2·6	162·50	1·625	4·875	8·125	11·37	16·25	24·37	32·5	40·62	48·75	56·87	65·0	73·12	81·25	89·37	97·50
	2·8	155·55	1·555	4·666	7·777	10·89	15·55	23·33	31·11	38·89	46·66	54·44	62·22	70·0	77·77	85·55	93·33
	3·0	150·00	1·50	4·500	7·500	10·50	15·00	22·50	30·0	37·50	45·0	52·50	60·0	67·50	75·0	82·50	90·00
Values of V = percentage by volume of solid in pulp = c.cs. solid in 100 c.c. $= (k-100)(p-1)$		$k - 100$															
	2·4	71·428	0·714	2·143	3·571	5·000	7·143	10·71	14·28	17·86	21·43	25·0	28·57	32·14	35·71	39·28	42·86
	2·6	62·500	0·625	1·875	3·125	4·375	6·250	9·37	12·50	15·62	18·75	21·87	25·00	28·12	31·25	34·37	37·50
	2·8	55·555	0·555	1·666	2·778	3·889	5·555	8·33	11·11	13·89	16·66	19·44	22·22	25·0	27·78	30·55	33·33
	3·0	50·000	0·500	1·50	2·500	3·500	5·000	7·50	10·00	12·50	15·00	17·50	20·00	22·50	25·00	27·50	30·00
Values of A = tons dry solid per 100 tons water = grams solid in 100 c.c. water $= \frac{100}{R}$	2·4		1·726	5·25	8·89	12·63	18·5	28·7	40·0	52·2	65·5	80·0	96·0	113·6	133·3	155·3	180·0
	2·6		1·635	4·97	8·38	11·9	17·3	26·9	37·15	48·1	60·0	72·8	86·6	101·7	118·2	136·4	156·0
	2·8		1·563	4·75	8·00	11·33	16·5	25·5	35·0	45·2	56·0	67·6	80·0	93·3	107·5	123·3	140·0
	3·0		1·508	4·67	7·70	10·88	15·8	24·3	33·3	42·9	52·9	63·6	75·0	87·7	100·0	114·0	128·6
Number of fluid tons pulp to yield one ton dry solid $= \frac{100}{(p-1)k}$ †		$\frac{100}{k}$															
	2·4	0·58333	58·33	19·44	11·66	8·33	5·833	3·889	2·916	2·333	1·944	1·667	1·458	1·296	1·167	1·060	0·972
	2·6	0·61461	61·46	20·48	12·29	8·78	6·146	4·096	3·073	2·458	2·048	1·756	1·536	1·365	1·229	1·117	1·024
	2·8	0·64286	64·28	21·43	12·86	9·18	6·428	4·286	3·214	2·574	2·143	1·836	1·607	1·428	1·286	1·169	1·071
	3·0	0·66667	66·67	22·22	13·33	9·52	6·667	4·444	3·333	2·667	2·222	1·905	1·667	1·481	1·333	1·212	1·111

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* These values multiplied by 0·625 or divided by 1·6 give pounds dry solid per cubic foot of pulp.

† These values multiplied by 32 give cubic feet of pulp to yield one ton dry solid.

platform scale should have a top lever to release all knife-edges. The cubic-foot bucket gives gravity when the net weight in pounds is multiplied by 0·016, while it may be read directly from the 100 lb. sampler.

Graphic Solution of Problems.—The curves illustrated in Fig. 1 embody the principal factors for pulp containing solids of density 2·6 and 2·8. If plotted on a large sheet of squared paper they may be used to read off the values with only a small percentage error except near the extreme ends, or between the abscissae representing specific gravities 1·05 and 1·50. Numerical data for plotting similar curves for solids of density 2·4, 2·6, 2·8 and 3·0 are given in Table VI., covering pulp of any specific gravity up to 1·6 : the values given being plotted as ordinates on abscissae representing the specific gravity of the pulp. For material of any intermediate density the ordinates of the curves are obtainable with sufficient accuracy by interpolation. Table VII. contains similar data for a curve of sluicing values, starting with filter-cake or residues containing 33·3% water : further data in Table VIII. enable the extension of this to material carrying 25 to 40% water.

Photographic reproductions of curves for ore of a particular gravity can be used for ordinary approximations and can be carried in a pocket-book. The most frequent use of such a curve is to read certain constants of a pulp, such as the percentage of solid, starting with the specific gravity. It can be used for computation in many other ways, and most of the numerical problems bearing on pulp can be solved graphically by means of it. In general the first step is to find the specific gravity of the pulp if this is not given.

If it is required to find the amount of slime that can be handled in a vat that will hold 500 tons of water, the dry slime having a density of 2·7 and being mixed with twice its weight of cyanide solution, we first proceed to find the gravity of pulp that corresponds to a water ratio of 2·0. Noting in Fig. 1 where the horizontal line corresponding to 2·0 cuts the water-ratio curves for density 2·6 and 2·8 ; midway between these would be the intersection of the 2·7 curve, at a point corresponding nearly to the abscissae for specific gravity 1·265. Following a vertical line through this point to where it cuts the curves for the 'solid factor,' we see that for a density of 2·7, midway between 2·6 and 2·8, the intersection would fall close to the horizontal line through 42. That is to say 100 fluid tons of pulp would contain 42 tons of solid matter under the conditions named, or a vat of 500 fluid tons capacity would accommodate 210 tons. With this would be twice its weight of solution, or a total of 630 tons of pulp, of a gravity 1·265 which corresponds to about 79 lb. per cub. ft."

W. J. SHARWOOD. *The Mining Magazine*, p. 297. (W. R. D.)

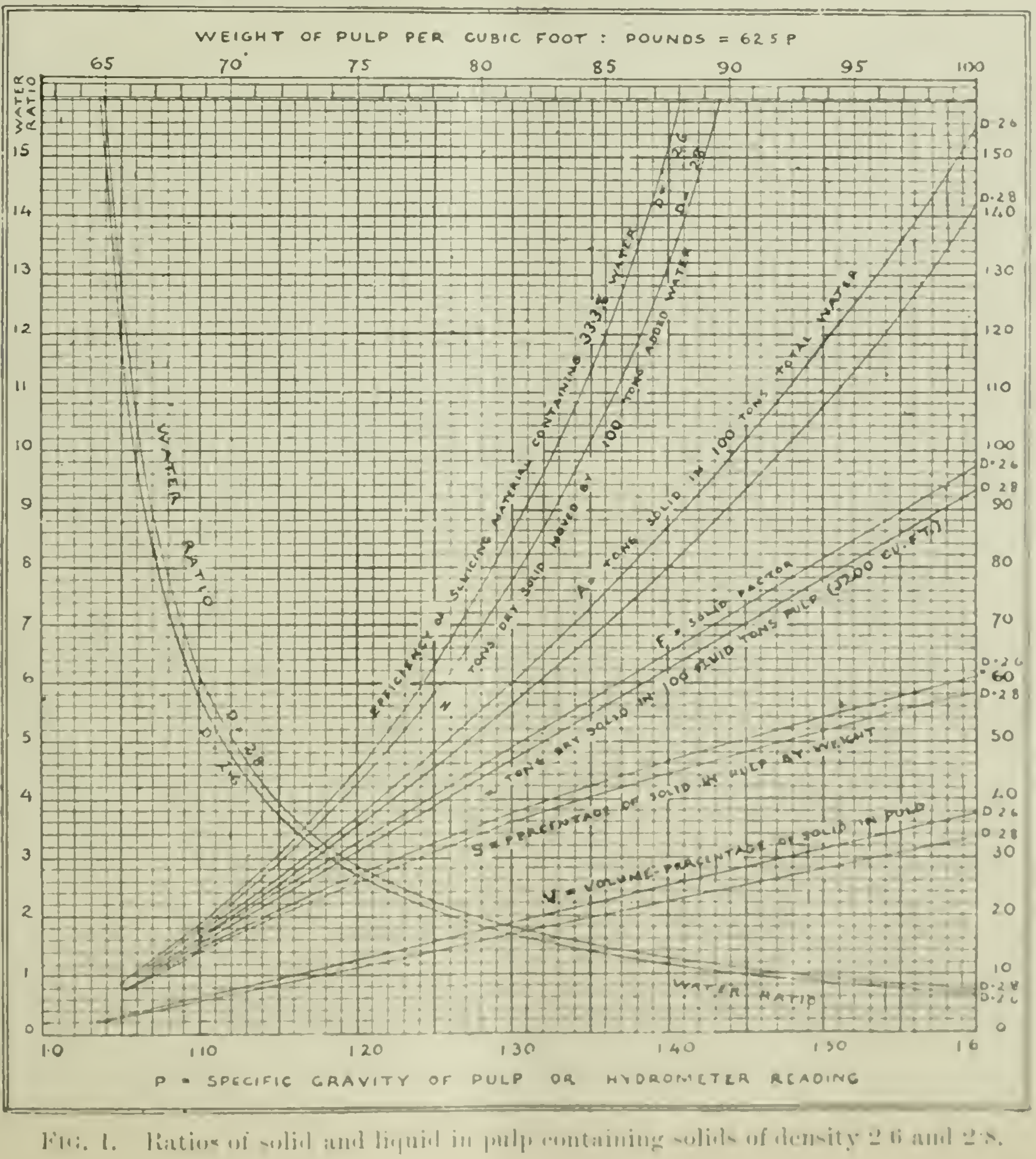
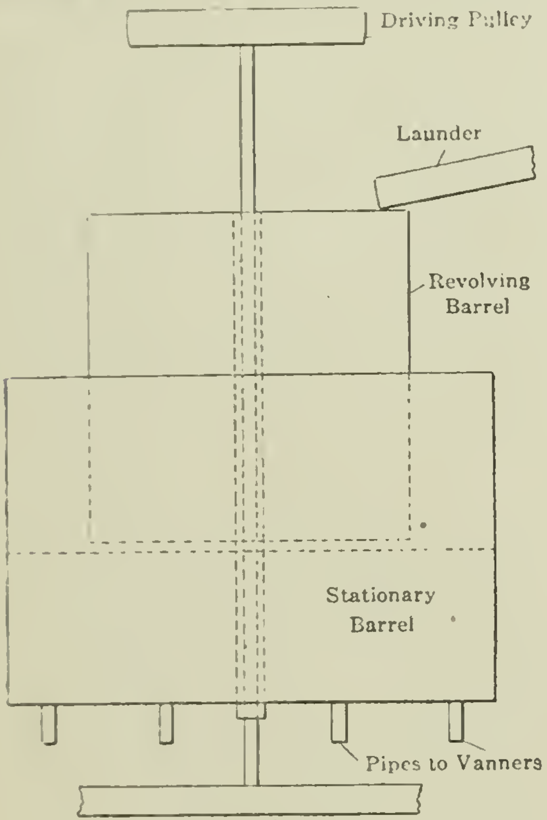


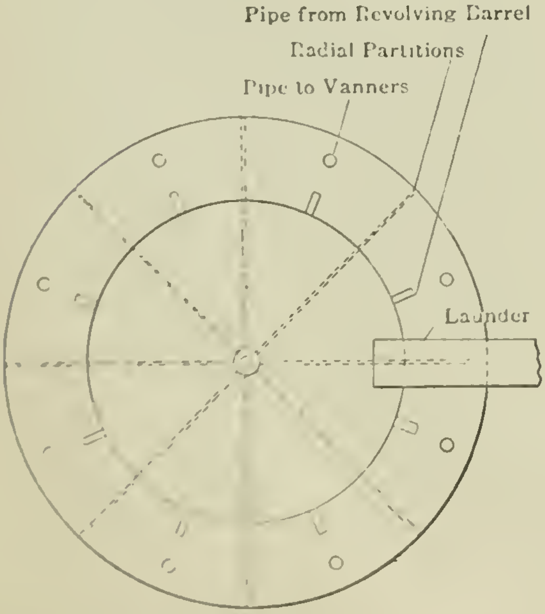
FIG. 1. Ratios of solid and liquid in pulp containing solids of density 2·6 and 2·8.

PULP DISTRIBUTER.—"The following description of a Pulp Distributer, taken from *The Engineering and Mining Journal*, may prove useful in helping to solve the problem of procuring an even distribution of pulp to classifiers, shaking tables, etc. :—

The accompanying sketch shows a pulp distributer in use at several mills. The machine shown is driven by a belt from a vanner line shaft, but by turning the spouts horizontally it may be made automatic, although the feed will not be positive and the machine will stop when the pulp discharge is low. The machine consists of a barrel, the bottom of which is divided into as many radial segments as there are vanners or tables to feed. Inside this barrel another barrel revolves, carrying in the side near the bottom as many radial pipes as there are divisions in the outer stationary barrel. This is not arbitrary, since as many pipes may be used as necessary. The pulp is fed into the inner barrel, which revolves at about 50 r.p.m. and distributes the pulp evenly into the compartments of the outer stationary barrel. This arrangement will satisfactorily feed 10 tables." (J. E. T.)



Elevation



Plan

DETECTION OF GOLD IN COUNTRY ROCK.—"There is a certain amount of bravery demanded in bringing forward notes on negative results, and Mr. Andrew in doing so, in this instance, is to be congratulated. His results are negative because he set himself a task which most of us have recognised as almost hopeless and beyond the sphere of practical economy. So much is this the case that his conclusions are adopted by many mining engineers, who prefer the use of the word *nil* instead of *traces* in their reports. Mr. Andrew's results would greatly tend to justify their action.

I have assayed many samples of country rock in the form of bore-hole cores, and in using two assay tons for the charge and fluxing with litharge, soda, borax, and reducer, I have detected no gold in parting. In parting a number of these together, say 20 to 30, however, a fine powdery gold is seen which is practically unweighable.

The presence of this would not justify any assayer in reporting any of the results as traces, since the occurrence of gold in minute particles in litharge is known, but more especially because such a report is apt to be confusing to non-practical men, and to my certain knowledge has led to the spending of further sums of money in what could well be regarded as hopeless cases.

The impracticability of obtaining an absolutely pure litharge has long been known and most assayers are now quite content to obtain this flux with a guarantee of uniformity of the silver and even gold contents, though, of course, the quantity of the latter present is not sufficient to influence the result. The variation of 10% in the silver-gold contents of litharge is, of course, large; but I have not infrequently found much larger differences than this. The author is not very clear as to whether his results were obtained by micrometer measurement of the beads. If so, then his conclusions are apt to be confusing. The method may be applicable with samples fairly rich in gold, but would be useless in the measurement of the very small beads with which the author deals."—A. McA. JOHNSTON.—*Inst. of Mining and Metallurgy, Bulletin No. 65*, Feb. 10, 1910, p. 49. (H. A. W.)

TREATMENT OF COPPER ORES.—"The method relates to the treatment of copper ores for the extraction of the metal, in which the ore, if necessary after the addition of alkali, is treated with a solution of sulphuric acid or sodium bisulphate together with an oxidising agent. By the present method the crushed ore is treated in shallow vats with a solution containing 1 to 3 per cent. of sulphuric acid according to the amount of copper. To this solution is added a calculated amount of potassium permanganate, say 3 to 6 lb. per ton of ore when the latter contains 2½ to 3 per cent. of copper. When the acid solution is placed on the charge, one-half of the permanganate is added. After the action has proceeded for about twelve hours, the solution is drained away; eventually, after the addition of half the remaining permanganate, it is used for the treatment of a fresh body of ore and then passed to the precipitating boxes, the remainder of the permanganate being added beforehand. Meanwhile the original charge is treated with a fresh quantity of solution together with a corresponding amount of permanganate. After twelve hours this solution is also drawn away and brought in contact with the second charge in exactly the same manner as before. Sulphide ores are roasted before treatment with the solvent solution.—C. H. N. WILLIAMS. *Journ. Soc. of Chem. Industry*, Vol. xxix., No. 4. Feb. 28/10. p. 220. (J. A. W.)

THE ASSAY OF INDUSTRIAL GOLD ALLOYS.—"The author deals mainly with the standard English gold alloys used for jewellery, etc., and considers in detail each of the operations comprised in the usual methods of assay, the latter involving the removal of the oxidisable metals by cupellation with lead, inquartation with silver, and subsequent parting with nitric acid. The methods adopted in sampling the various commercial forms of such alloys are given, the proportions of lead required for the removal of the base metals from them being next considered. The composition and uses of checks, the losses of precious metal occurring in the various operations, and the surcharges on assays made under varying conditions are likewise fully discussed; while examples of the usual methods of reporting the assay results, and a considerable amount of tabulated experimental data are also included."—E. A. SMITH. *Inst. of Metals*, Jan., 1910. *Journ. Soc. of Chem. Industry*. Vol. xxix., 4. Feb. 28/10, p. 215. (J. A. W.)

MINING.

AN INSTRUMENT FOR DETERMINING VERTICAL DEPTHS IN DIAMOND DRILL HOLES. "In the construction of the new aqueduct of the city of New York, which crosses the Hudson River just north of West Point, it became necessary to determine accurately the vertical depths of the narrow diamond drill bore holes which are being sunk under the Hudson River at various angles and which have already reached depths of 1,500 ft.

This problem has been solved by Professors G. N. Lewis and H. T. Kalmus, of the Massachusetts Institute of Technology, who have devised a self-recording pressure gauge which has a diameter of less than ⅞ in. and can therefore be placed directly in the terminal section of the drill rod. Since the bore holes are constantly filled with water, the maximum hydrostatic pressure recorded is a direct measure of the vertical depth. It seems probable that this instrument may be of service in other engineering and mining work where it is desirable to determine the vertical depth of bore holes used in prospecting.

The pressure gauge proper consists of a very thin strip of tempered steel bent into the form of a hollow rectangular tube approximately ¾ in. wide, [illegible] in. thick, and 18 in. long. The edges and the lower end of the tube are welded together with the oxy acetylene flame. Into the upper end is welded a small steel tube which in turn is sealed to a straight glass tube, the upper end of which enters an air tight chamber. The gauge and connecting glass tube are filled with mercury and when subjected to an external pressure the thin steel walls of the gauge undergo considerable temporary deformation, thus diminishing the total volume of the gauge and forcing the mercury from the upper end of the glass tube into the surrounding chamber. When the pressure is released the gauge resumes its original volume and the mercury level in the glass tube falls through a distance which measures directly the pressure to which the apparatus has been subjected. The mercury may now be returned to the glass tube from the surrounding chamber by a simple device.

The instrument gives remarkably reliable results and is able to record vertical depths up to 1,500 ft with an error of no more than 2 ft. or 3 ft. On account of the extremely small volume of the thin steel gauge, the effect of temperature changes, even amounting to 30° or 40°, is entirely negligible."

Canadian Mining Journal, Dec. 15, 1909, p. 758. (A. R.)

PROSPECTING FOR GOLD.—"At first it was the general belief that gold was most likely to be found in veins that traversed the oldest formations of granite gneiss and slaty schists. Recently this belief has been changed because rocks containing gold have been studied, and gold has been found in rocks of more recent formation. If the placers are left out of consideration, gold is always found with quartz, alone or associated with some other rock-forming mineral, preferably a magnesium silicate. Most quartz veins are barren, but if there is any mineral at all in them gold will probably be associated with it, for gold is found with copper, iron, lead, and zinc in such deposits. At the surface the quartz will be stained to rusty brown by iron oxide and perhaps be pitted or honeycombed. Quartz of this description is always worthy of examination, and if upon crushing and panning it fails to show gold, a fire assay should be made of an average sample. There are rich and lean places along every vein, for which reason the prospector should look for the richest before staking his claim. There are not enough bonanzas or rich gold deposits to go around, and one can never tell when the lightning will strike in this game; however, from an examination of the rocks in which the lucky ones struck bonanzas, the writer has found them to be near eruptive rocks and in magnesian silicates. By eruptive rocks is meant those rocks which were forced from below upwards through fissures in the surface, and since they were putty like masses at the time of eruption they are called magmas. Such magmas were accompanied by solutions which carried minerals, and wherever there was some kind of material which would set up a chemical reaction, the metal minerals were deposited. Solutions carrying gold would on coming in contact with magnesian rock deposit the gold in a small area; consequently, bonanzas are not extensive. It is to be understood that there are some exceptions to this rule; for instance, the solutions carrying gold may come in contact with substances that will precipitate the gold, such as carbon, fluorine, iron pyrite, tellurium, etc., but such deposits are not as likely to be as concentrated as those near talc and serpentine. The magmatic rocks from which the gold solutions were derived are usually gray, while the talc and serpentine rocks are greenish, and have a soapy feel to their powder. If the country is comparatively flat in the immediate vicinity of those rocks, the surface soil will be coloured gray or reddish brown by iron oxide and be mixed with sharp quartz which will cut the shoe soles, also the gravel from the brooks and depressions will furnish black sand and gold on panning.

Aqueo igneous rocks that were shattered by earth movements and afterwards had their crevices cemented by quartz solutions, while low grade, as a rule have the gold more uniformly distributed over a larger area. Frequently such magmas have sulphide, telluride, and fluoride metal minerals as a component part of their mass, in which case the minerals are said to be 'accessory,' and frequently have gold associated with them. In some cases these rocks may be worked for gold, but not usually; however, as they disintegrate with comparative ease, solutions from them may form rich gold deposits. It is a good plan for the prospector who discovers minerals in this class of rock, to inspect the contact they make with other rocks, particularly with limestone or the magnesian rocks, talc, or soap stone and serpentine. As a rule the whitish and grayish coarse grained rocks are the magmas carrying the gold, while the contact rocks in which the free gold is likely to be found are greenish, blue drab, purple, white carbonates, blue-black slates, and any coloured sandstone. The latter rocks have the gold in crevices, veins, and pockets. In the East, chlorites and mica schists were crumpled by earth movements, and between the folds of the leaves quartz solutions carrying metal minerals were injected and hardened. The intercalated quartz carries gold. Somewhat similar formations are found in California and South Dakota where, instead of schists, the rocks approximate slate. Gold is frequently found with other metal minerals, but if in subordinate value it is considered a by-product, thus the silver-lead and copper smelters always produce some gold."—*Mines and Minerals*, Dec., 1909, p. 277. (A. R.)

TESTING FOR CARBON MONOXIDE.—"The authors, Drs. J. S. Haldane and C. Gordon Douglas, began by remarking that it was now well-known that the main dangers to life in connection with colliery explosions and fires arose from the presence of carbon monoxide (CO) in the air, and that a lamp gave little or no indication of this danger.

Serious colliery explosions, continued the authors, are nearly always dust explosions, and usually the main intake and haulage road is involved. The explosion sweeps along this road, blowing out separation doors, stoppings, and air-crossings, and displacing timbering, so that falls of roof occur here and there. The air displaced by expansion is to a large extent replaced almost immediately by fresh air sucked in; but along the affected roadway a mixture is left which, although it contains enough oxygen to support life, and enough often to support even lights, is extremely poisonous, owing to the presence of carbon monoxide.

Those who have seen quite fresh after-damp, as, for instance, in the experiments which Mr. Garforth is carrying out at Altofts Colliery for the British colliery owners, will know that it is exactly like very thick and 'dirty' smoke. It might, therefore, be thought that air which is poisonous from after-damp or smoke would always be recognised quite easily. That is a mistake. The particles which make smoke or after-damp visible, and easily recognisable by smell, subside gradually, leaving clear air which still contains all the poisonous carbon-monoxide of the original mixture. Dr. Turner said that he and those with him were deceived by the places where the air was clear after the Darran explosion. Another striking instance was that of the fire at Snaefell Mine, where a whole shift went straight down ladders into intensely poisonous air, not noticing any smell of burning or sign of smoke.

The Live Animal Test.—The authors continued: Enough having been said to show the practical importance of having some reliable test for air which is dangerous from the presence of carbon monoxide, we may now proceed to discuss and demonstrate the test which appears to be the most practical one for use in mines. This is the test by means of very small warm blooded animals. So far as we are aware, it was first actually employed in mines by Dr. W. N. Atkinson, then Inspector of Mines for Staffordshire, and by Mr. A. M. Henshaw, of Talk o' th' Hill Colliery. Either a mouse or a small bird, as preferred by some, may be employed. A mouse was originally recommended, as white mice are very easy to keep. There is still, however, a good deal of misapprehension as to the significance of the test and the mode of applying it.

The principle of the test is as follows:—In small warm-blooded animals the rate at which chemical changes occur in a given body-weight is enormously greater than in large animals. Thus a mouse weighing about half an ounce consumes about fifteen times as much oxygen as half an ounce of the human body would consume in the same time. A reason for this difference is evident enough. With bodies of the same shape and composition, but different sizes, the surface increases as the square of any corresponding diameter, but the mass as the cube. The larger an animal is, therefore, the less surface will it have for a given mass, and the less rapidly will a given mass of it lose heat to the environment; or the less heat, and consequently the less oxygen, will a given mass of it require in order to maintain the normal body temperature. Not only are the chemical changes in the small animal far more rapid, but the rate of respiration, circulation, etc., are correspondingly increased. It is difficult to count by the eye the rate of breathing in a mouse, and quite impossible to count its pulse-rate. By a photographic method of recording the electrical changes which accompany the heart beat, Miss F. Buchanan, of Oxford, has recently shown that the pulse-rate in mice and small birds is from 700 to 1,000 per minute.

It follows that the small animal will absorb any poisonous gas far more quickly than a man will, and will, therefore, show symptoms of poisoning far sooner. It can, in fact, be employed to show what will ultimately happen to a man if he remains in the poisonous air. This, and this alone, is the principle of the test. The small animal is not, in the long run, more sensitive than a man to a given percentage of carbon monoxide. Indeed, the opposite is almost certainly the case; for a man's central nervous system is in many ways more liable to harm than that of any lower animal, and an animal seems to recover much more readily and rapidly from the effects of the gas than a man does. We have never observed in animals any of the after effects which so commonly follow partial poisoning by carbon monoxide in man.

Physiological Results of Carbon Monoxide.—As is now well known, carbon monoxide exercises its poisonous action by combining, to the exclusion of oxygen, with hæmoglobin, the red colouring matter of the blood, and thus preventing the hæmoglobin from exercising its normal function of conveying oxygen from the lungs to the tissues of the body. With the comparatively small percentages of carbon monoxide met with in mines, it takes a considerable time for enough of the gas to be absorbed into the blood of a man to produce appreciable symptoms. Even with a poisonous proportion of carbon monoxide in the air it may take half an hour to produce serious symptoms, while a mouse or small bird is correspondingly affected in two minutes. With smaller, but still serious, proportions of the gas it may take as much as two or three hours for symptoms to be produced in a man. Thus with 0·1% of carbon monoxide in the air we have found that during rest it took about two hours before giddiness, etc., began to appear. Periodic analyses of the blood showed that absorption was proceeding steadily, and that another half hour would have sufficed to produce practical disablement. A mouse is similarly affected within 10 minutes.

With less than about 0·02% of carbon monoxide in the air, noticeable symptoms are never produced, since absorption ceases when the blood becomes saturated to a comparatively slight extent, although even this would take several hours during rest. Owing to the fact that equilibrium tends to establish itself between the carbon monoxide in the blood and that in the air breathed, and that this equilibrium may be established at various levels, corresponding (if any symptoms at all are produced) to various degrees of disablement, there is a very wide range between percentages of carbon monoxide which will cause sensible inconvenience and those which will cause death. Men may be disabled for hours or days by carbon monoxide in mines, and yet finally recover. We do not know accurately what percentage of the gas is required to cause death in a man within an hour or two, but usually about 0·5% is needed in an animal, although as little as 0·2% may be fatal. Analyses made of the blood of men who have been killed at various colliery explosions show that the hæmoglobin is usually about 80% saturated with carbon monoxide when death occurs. On the other hand, it is known from experiments that practical disablement occurs when the blood is about 50% saturated with carbon monoxide.

If a mouse is used for testing, the animal should be in a cage large enough for it to creep about: otherwise, there will be a difficulty in judging of the effect of carbon monoxide upon it. For the same reason, a bird should have a perch in its cage. Birds are more easily killed by carbon monoxide than mice, owing, probably, to the fact that they are apt to struggle if they lose their balance, and that any sudden exertion is dangerous. In one sense, therefore, the bird is a better indicator of carbon monoxide than a mouse. But if the animal dies, it ceases to be of use as an indicator, just as a safety lamp becomes useless if it is extinguished. From this point of view, the mouse is superior. A miner needs to know, not only whether he is in dangerous air, but whether he has got back to safe air. If animals larger than mice or small birds are used, it must be remembered that they take a correspondingly longer time to react to the carbon monoxide. A rat or pigeon will take about twice as long as a mouse, a rabbit twice as long a rat, and on up to a horse, which probably takes twice as long as a man. The rate of recovery from carbon monoxide poisoning is correspondingly slower in the larger animals.

A man who is moving about breathes much more air than a man at rest, but the rate at which he absorbs carbon monoxide does not increase in the same proportion. The carbon monoxide may, however, be absorbed about twice as fast as during rest. Even, however, if this is the case, it will take seven or eight times as long for the man as for the mouse to be affected. The safe rule, however, is to retire at once from air in which the animal shows any symptoms. It must also be remembered that with proportions of carbon monoxide which are not actually dangerous a man may be ultimately affected more or less, although a mouse or bird shows no very evident symptoms. The symptoms of the animal can only be relied on to indicate more or less dangerous proportions of carbon monoxide.

Practical Use of the Test.—The animal test is in some ways a rough one, but the authors believe it to be a practical one, for the animal, just like an ordinary miner's lamp, is constantly testing the air. The miner can watch the animal just as he watches his lamp for black-damp or fire-damp. The air in a fan drift, upcast shaft, and various roads in a mine can be quickly, and, if necessary, constantly tested by this means if danger is present, so that all that is possible without the use of rescue apparatus can be safely accomplished, and a rescue corps can be followed up and as far as is possible without danger. The animal can also be used for testing any kind of irrespirable air: and in case of urgent necessity men can go safely with a mouse or bird and an electric lamp into air where no lamp would burn—for instance, into air containing more than 5% of fire-damp, or more than 15% of black-damp.

A mouse may also be used for testing for very small percentages of carbon monoxide down to 0·01%. The animal is left for about 10 or 15 minutes in the suspected air, then drowned, and a drop of blood from its heart examined by day-light for the pink colour. This test may be very useful in cases where heating of coal, or an actual fire, is suspected, but cannot be localised. It is not only a certain, but also an extremely delicate test.

Questions.

At the conclusion of the lecture the Chairman invited members of the audience to ask questions, which would be answered by Dr. J. S. Haldane.

The following questions were put and answered:

(Q.) Supposing the blood of a person to be half saturated with CO, how long would it take for that person to reach a normal condition?

(A.) Very difficult to say, as the after-effects upon persons vary; in some it takes considerably longer than others to reach a normal state.

(Q.) How long does it take the blood to pass round the human body?

(A.) There is a controversy at present raging in the physiological journals on this point. One school of thought say about one minute, while the other asserts two or three minutes.

(Q.) How long would it take for a person to become effected in a percentage of 0·2?

(A.) The person would be safe for an hour if in a state of rest, but if working, half an hour. Upon a miner becoming affected it is very necessary for him to leave the place as quietly as possible; anything in the nature of hurry and over-exertion would be fatal.

(Q.) What is the lowest percentage of CO that can be detected?

(A.) 0·01%.

(Q.) What are the after-effects of CO poisoning in a pure mixture of air and the gas?

(A.) A throbbing headache and weakness, although in the case of Dr. Turner, he came round with no headache, and some people were not affected that way."—Drs. J. S. Haldane and C. Gordon Douglas, S. Staffordshire and Warwickshire Institute of Mining Engineers.—*Iron and Coal Trades Review*, Dec. 17, 1909, p. 959. (A. R.)

A New Method of Shaft Sinking.—"Mr. John Cummings gave an account of 'Sinking the John Shaft at Hamsterley Colliery through Sand and Gravel by Means of Underhanging Tubbing.' The colliery, he stated, was situated about ten miles south west of Newcastle. The royalties, which lay chiefly to the east, had a slight dip in that direction and were heavily watered. Most of the coal lay above the drainage level of the district, and had been drained by a water-level drift driven from the level of the River Derwent; but, as a large area of coal in the lower seams was below that level, it was decided to sink a shaft at the lowest point in the royalty and fit up a pumping-station in order to drain that area. The most suitable site was on the alluvial flat near the river, close to and on the dip side of a large fault which intersected the royalties. A borehole was put down on the site of the shaft, and proved the existence of a bed of sand and gravel extending from the surface to a depth of 25¾ ft. and of very soft and loose ground to a depth of 54 ft. 8 in. It was at first proposed to sink by means of piles and afterwards to tub the shaft; but, as the site was close to the river and a heavy flow of water was expected, it was decided to adopt the method of underhanging or suspended tubbing in order that the feeders might be at least partly tubbed off as met with. The finished shaft was 10 ft. in diameter, and the whole of the plant in connection with the sinking was driven by electrical power, which was generated by the colliery company at their power station about 1¼ miles distant, the current being three-phase at 550 volts and 50 cycles. A commencement with the sinking was made by excavating an annular trench in the sand to a depth of 4 ft., with an outside diameter of 19 ft. An annular concrete foundation ring having an internal diameter of 10 ft. 10 in., and an outside diameter of 19 ft., and 3 ft. deep, was then laid, the top of the concrete being 1 ft. below ground level. The concrete block was strengthened by scrap iron bars and rails laced across each other. Great care was taken to level the top of the concrete block, as that formed the bed for the cast iron hanging-ring from which the whole of the tubbing was suspended, and upon that the vertical condition of the shaft depended. The hanging-ring or curb was of cast iron, 11 in. deep and 19¾ in. wide in the bed. It was in six segments, bolted together to form a circle 10 ft. in diameter inside the flanges. Three of these six segments had holes sloping downwards through which the liquid cement was poured for grouting the courses of tubbing below. All the joints were machined, and lead sheeting ⅛ in. thick was placed between them to form a watertight joint. The hanging-ring was laid on the foundation with the inner flanges projecting over the edge of the concrete, the top of the ring being flush with the surface of the ground. As a further precaution against slipping and in order further to spread the weight, two baulks of timber 20 ft. long and 22 in. square, were laid across, one at each side of the pit and bolted to the hanging-ring. The sand was then taken out of the centre of the pit for a depth of 5 ft. That was done without difficulty, as there was a concrete lining for a depth of 3 ft. A course of tubbing was then put in and bolted to the bottom flange of the hanging-ring, strips of lead sheeting being inserted between all joints, both vertical and horizontal, before the bolts were put in. The bolts being thoroughly tightened up, the space at the bottom of the tubbing all round the shaft bottom was rammed tight with well-puddled clay, and liquid cement was poured into the grouting holes of the hanging-ring above, so as to fill up the space behind the tubbing. Long tapered plugs were driven into the bottom grouting, these having a twofold purpose—first, to prevent the liquid being wasted, and second to preserve the holes for grouting the next course below. The mixture of grouting was varied as required. Near the surface, where the sand was fairly dry, the mixture was two of sand to one of cement, but in the worst ground, where it was required to set quickly, pure cement was put in. Little or no difficulty was experienced in putting in the first course of tubbing, as the water was just showing in the pit bottom, in fact, the work was done by ordinary surface labourers with a mechanic to direct operations. The method of getting the closing segment of the circle into place while the adjoining ones were in position was by excavating the shaft-side behind the last segment, thus making room for it to swing endways in behind the adjoining segment and then bringing it forward into position. The tubbing was of cast iron 1 in. thick, with six segments to the circle. Eight courses of segments were 2 ft. 5½ in. deep for going through the sand and worst ground, and eight were 4 ft. 11 in. deep, the object of the deeper rings being to save joints, as these, being machined, were costly. Every alternate segment in the circle was provided with a grouting hole just above the bottom flange, which sloped downwards as in the hanging-ring. All the joints were made of sheet lead ⅛ in. thick, accurately cut to pattern, each strip projecting ⅛ in. beyond the face of the joint to allow for caulking, which was done after the face of the joint was completed. The bolts were all turned $1\frac{5}{16}$ in. in diameter, and, to prevent the threads from being damaged, ⅛ in. of it was turned off at the end of each bolt. In order to make the bolt-holes as well as the joints watertight a conical lead washer was put on next the flange at each end of the bolt, and a metal washer having an inverted cone to fit over the lead one was put on next to it. When the nuts were tightened up the soft lead washer was squeezed between the cast iron of the flange and the conical space in the metal washer and formed a perfectly watertight joint. Small protecting horizontal ribs were cast on the outside of the tubbing and formed supports for the tubbing in the cement grouting. On July 27 of last year the actual sinking was commenced below the first course of tubbing with four men in the pit bottom, who began by digging out the sand at one side of the pit wide enough to get in one segment of tubbing. That segment was made good and bolted to the course above, and being taken out for the adjoining segment, which was also bolted up, and so on until the circle was completed, when the bottom was rammed with clay as before and cement grouting poured in through the holes in the course of tubbing above. While excavating for that course the sand was somewhat difficult to keep back until the segments were got in, especially as the men were all new to the work, but it was finished without mishap by 6 a.m. two days later. For the next and subsequent courses short slabs of wood, 2½ ft. long, were used as piles. These were driven in in a sloping direction under the tubbing so as to keep back the sand until a sufficient depth was obtained to insert a segment, continuing round the shaft until the circle was complete. A great quantity of brushwood was also used for the same purpose, being especially necessary at the open ends of the segment until the next one was joined up against it. When sinking for the fifth course of tubbing the sand was so soft that the men had to have short planks to stand on to prevent them sinking into it. For the first three courses, the quantity of water was not great and was kept down by a hand pump worked by two men. At a depth of 12 ft., a small electric centrifugal pump, having a capacity of 150 gallons per minute, was put in. It was only found necessary to keep this pump going intermittently, as the quantity of water varied most remarkably. It was noticed regularly, while the men were sinking or putting in the segments, the shaft sides being more or less open, that the quantity of water increased considerably and the pump was kept going to its fullest capacity to enable work to be carried on at the pit bottom; but after each course was completed and the bolts were tightened up, the quantity was greatly reduced, especially after the grouting was put in. This was particularly noticeable when the boulder clay was reached at a depth of about 30 ft., as, when the course was completed, the pit was quite dry and the pump was temporarily taken out. It was, in fact, so dry that water had to be brought from the river for the purpose of mixing the cement for the grouting. Although the large sinking pump was at hand on the works, it was not found necessary to use it until the tubbing was finished and the pit was being sunk through the stone below in the ordinary way. At the surface, a course of the deep segments was put in, as the ground was fairly easy. The eight shallow courses were next inserted, the remainder of the deep segments being put in last. Before the sixteenth course was put in, the sinking had reached a good firm stone. The sinking was carried down for over 30 ft. into the post stone, when a walling bed was formed and the shaft was lined up with walling lumps 9 in. thick. The walling was carried up to within about 6 ft. of the bottom of the tubbing and a wooden crib of elm was made to suit and laid on the top of it, being wedged firmly from behind. The last of the tubbing was then put in and the joint between it and the crib wedged and made tight. While the sinking and walling was being carried out below it, the whole length of tubbing remained suspended from the hanging-ring at the surface, entirely without support at the bottom (except so far as it was self-clinging in the grouting) until the walling was joined up below it. The total weight of tubbing with bolts, etc., in the shaft was over 56 tons. The time occupied in sinking the shaft, putting in grouting and completing the tubbing to a depth of 55 ft. from the surface was twenty-eight working days, including the time occupied in laying the hanging-ring but not including the stoppage of six days after the tenth course was finished in the boulder clay due to a consignment of bolts being delayed in transit. The total cost of sinking, tubbing and finishing the pit for the above-mentioned depth was as follows: Materials, £627 2s. 10d.; labour, £141 0s. 3d.; total, £768 3s. 1d., or £13 19s. 4d. per foot. The work was executed without an accident to the workmen, and, in the writer's opinion, the economy and safety of the method of putting in tubbing should make its adoption general in all cases where ordinary wedged tubbing could be used. The initial cost of the machined tubbing with bolts, etc., was no doubt greater than that of ordinary tubbing, but that was more than compensated for by inserting the permanent lining as the work progressed, thus saving the cost of both materials and labour of putting in any temporary lining and also affording greater protection to the workmen below from falls of side. The actual cost of putting it in would also be less than for wedge tubbing. The only objection to its general use, so far as the writer could see, were: (1) The absence of a good foundation from which to suspend the tubbing; (2) the difficulty of getting the grouting against high pressures; and (3) the difficulty of securing a tight joint at the bottom of the tubbing courses to hold the grouting when the pressure was great." — *The Colliery Guardian*, Dec. 24, 1909, p. 1291. (C. B. S.)

SHAFT SINKING THROUGH SAND. — "In a paper read at the September meeting of the North of England Institute of Mining and Mechanical Engineers, E. M. Bainbridge and W. M. Redfearn give their experience with the Haniel & Lueg system of sinking shafts through sand and clay at the Newbiggin colliery, Northumberland, 15 miles north of Newcastle. The land is moor and bog, and is close to the sea-shore. Preliminary borings showed the presence of peat bogs below layers of clay. Finally a spot was chosen for sinking the two new shafts, about half a mile from the sea and where the peat-bog was not much more than a foot thick. The boring at this point showed the following strata: surface soil, 6 in.; yellow clay, 15 in.; peat bog, 15 in.; blue clay with stones, 5 ft. 6 in.; sand, 6 in., mottled clay, 5 ft., strong sandy blue clay with stones, 17 ft.; coarse sand full of water, 19 ft.; below this came a soft sandstone. It was soon found that ordinary methods of shaft sinking were useless, and after enquiry it was decided to adopt the Haniel & Lueg method. We give details of the operations on the first shaft sunk. The shaft was to be 20 ft. circular. A space 40 ft. square was first excavated, and a reinforced concrete structure built up to fill the space to a depth of 6 ft., with a central aperture with the same diameter as the intended shaft. An anchor ring consisting of stout iron segments bolted together was embedded in the concrete structure during construction, as were also anchor-bolts which extended upward. After the completion of this structure, brick walls were built on top, 30 ft. high and 6 ft. thick, with the above-mentioned bolts embedded in them. On top of this wall was placed the pressure-ring, made in segments similar to the anchor ring, and the anchor bolts fastened to it. On the under side of the pressure-ring, near its periphery, were attached ten hydraulic jacks. There were also guide bolts fixed between the anchor ring and the pressure ring for the purpose of directing the action of the jacks. The rams of the jacks worked adjustably on the top of the segments of tubbing. Below the first segment was placed the cutting shoe, which consists of 12 segments of cast steel 2¼ in. thick and 3 ft. 3 in. deep, the lower circular edge being bevelled at 45°. Around three sides of the brickwork that supported the pressure-ring was built another but thinner wall to support a circular track for a crane used in handling the tubbing, and timbers were erected for supporting the headgear of the excavator. A scaffold was also hung from the pressure ring to accommodate the men supervising the work of the rams and the placing of the tubbing. When work was commenced the shoe was placed upon planks at the bottom of the circular hole in the middle of the concrete block and the necessary connections were made. The planks were then withdrawn, and the cutting edge, being released, began to sink by its own weight for a depth of 10 ft. Pressure was applied to sink the shoe further and the tubbing was gradually built in. Until the sand was reached the excavation was carried out in the usual manner, but at this point water came in and filled the shaft to a level of 10 ft. above the surface of the ground. It was then decided to use a grab for removing the sand. When the shoe had passed through the sand and came to the heavy blue clay some difficulties were encountered. The ten jacks at 5 tons pressure each were hardly sufficient to force the shoe downward, and the grab was ineffective as an excavator. Eventually the brickwork pillar gaped slightly, showing that the strength of the structure between the concrete block, anchor ring, bolts, and pressure ring was not sufficient to withstand the strain when the jacks could not force the cutting shoe downward through the heavy clay. Modifications were then made by adding further bolts and by heaping sa and clay on top of the concrete block; also by altering the position of the jacks to bring them adjacent to the bolts instead of equidistant between each two. Instead of using the grab, the shaft was unwatered and men set to excavate. The latter was soon found to be a mistake, as inrushes of water kept taking place, and finally the grab was brought back. The operations went on slowly, but, eventually, three months after starting, the sandstone was reached at a depth of 76 ft. Owing to the greater bogginess of the land on one side of the operations the structure had gradually subsided, and the tubbing had not been sunk perfectly vertical, but the deviation was trifling. Work at the bottom was completed by cutting the stone away to a depth of 16 ft. below the sand, setting a curb on the level floor of the rock, bolting a matching piece to the curb and to the cutting-shoe, and filling the back between the curb and the rock with concrete. When sinking the second pit provision was made to strengthen the structure. The concrete block was made 50 ft. square instead of 40 ft. and the brickwork pillar made stronger on the side farthest away from the bog; also the anchoring-ring was embedded at the bottom of the concrete block instead of near the top."—*Mining Magazine*, Dec., 1909, p. 320. (A. R.)

ELECTRIC SHOT-FIRING.—"The primary object of the electric shot-firing system is to ensure greater safety to the men working in the mine. The method consists in firing simultaneously all the shots when all the workmen are outside the mine, by closing for a second a switch in a cabin on the side of the hill away from the pit-mouth. The system has been installed at Dawson, New Mexico, for somewhat over a year, and has given entire satisfaction.

The current is carried into the mine on the trolley-wires in the main haulageways. From the ends of these trolley-wires No. 8 insulated wire is used to branch off into the various side entries. No. 14 wire then being used to conduct the current to the face of each room. These auxiliary wires were strung up as far as possible from the tracks and near the roof, and are held by three-wire porcelain cleats, the two outside grooves being used.

The cleats are fastened to props, legs, caps of timber, or wooden plugs driven into the roof or rib, the supports being placed sufficiently close to ensure that the wires will not come into contact with each other, should the insulation become worn off.

There are two switches in the entry where the No. 8 wire is taped on to the trolley, One of these switches is in a box under lock and key, the box being constructed so that it cannot be closed and locked when the switch is thrown in; the other switch is in the open, and only acts as an auxiliary. Where two pairs of side entries are directly opposite each other, one set of these switches is used for the four entries, but otherwise there is a set for each pair of entries. Just inside the pit mouth there is another set of switches, one being between the outside and inside trolley-wires, another between the outside and inside feed-wires, and a third between the outside feed wire and the wire to the shot-firer's cabin on the side of the hill, and only the first of these is locked up. The main switch, by which the firing is done, is in the shot firer's cabin.

Mr J. E. Sheridan, the New Mexico Territorial Mine Inspector, writes as follows in the Transactions of the American Institute of Mining Engineers, with regard to electric shot-firing:—

'The shooting is done by electricity after all the men are checked out of the mine. As the men enter the mine, they are required to deposit a metallic check at the shot firing house outside, near the entrance to the mine. These checks are placed on a checkboard and returned to the men as they come from the mine. A record of the working place of each check-number is kept in the shot-firing house, and in case any check is uncalled for the shot-firer makes a search for the man until he is found. No shots are fired until it is known positively that no one is in the mine.

'To ensure safety against accidental discharge of the shots by electricity, there are two or more locked switch-boxes in each mine, with throw-off switches, one at the mouth of the mine and another at one or more stations inside the mine. After inspecting the inside-connections with the shots to be fired, the shot-firer *en route* from the mine makes connection at each of the switches mentioned. He then goes to the shot-firing cabin to turn on the electric current, but before doing so he turns on an electric signal light in a red globe, to warn all persons to remain away from the vicinity of the mouth of the mine; so that should an explosion occur within the mine no one outside could be injured by flying *debris*. The shot-firing system has proved a success; the safety of the men from disastrous dust-explosions due to blown-out shots is assured; the miners make better wages, and the production of coal is proportionately greater per man employed. A record is kept of the number of shots fired, showing less than 2% of missed shots. The missed shots are left for the next day's shooting, and are either re-primed or a new hole drilled to perform the work intended for the original shot. Very little fire-damp has been encountered thus far in the mines; but a supply of Wolf safety-lamps is kept ready for use.'"—JAMES DOUGLAS, North of England Institute of Mining and Mechanical Engineers.—*The Science and Art of Mining*, Dec. 24, 1909, p. 226. (A. R.)

CAUSES OF HANG-FIRES. — "Hang-fire holes, meaning those that do not explode immediately, but do so after a longer or shorter space of time, are due to two main causes. Most of them are due to a broken or damaged train of powder. This may be due to originally defective fuse or to sudden bending of the fuse, which often breaks the train of powder, or to too tight compression of the fuse by crimpers, which may cause the powder to burn less rapidly. In all but the last case fire travels along shreds of the covering until finally the powder train burns again. The time that the fire takes to travel past the damaged place determines the length of time that the explosion of the charge is delayed. Another, but less important, cause of hang-fires is the tendency of the miners, after splitting the fuse several times in a vain attempt to split the hole, to hold the end of the fuse in the flame of the candle and then to run away before they know whether the tar covering of the fuse has caught fire or not. Frequently this tar covering will have caught fire, but this is not noticed by the miner in his haste, and the covering smoulders for some time before the powder train is ignited. This is quite apt to occur, especially with contractors whose work next shift depends on all the holes' going and the round's breaking well. Anxious not to have this work delayed, contractors go back to spit the "missed" hole, and in the drift, murky with the smoke from the other holes, the miner fails to see in time to save himself the smoke from the smouldering fuse. After a wait of half or three-quarters of an hour there is little danger in going back, provided that the smoke has been blown out of the drift or raise, but even then there is some danger. A miner should never go back to a missed hole inside of half an hour, and when he does go back he should keep a sharp watch of the hole to see if he can see any indication of a slight smoke from the fuse. When the fuse is reached, it should be jerked out of the dynamite as quickly as possible.

Another class of work in which hang-fires often occur is in blasting chutes. This is due to the fact that the primer used often has to be fastened to a pole and pushed up the chute against the ore. Frequently this primer is tied to the pole by a couple of half hitches made in the fuse itself. The resulting sharp bends damage the train of powder. If nothing but fuse can be found with which to tie the stick of dynamite to the pole, then use a different piece of fuse than the one with the cap on it.

When capping a fuse at least an inch of the end of the fuse should be cut off, as the powder draws dampness. The places in which missed holes are reported should be written on a bulletin board so placed that each miner will see it when he goes to work, but the shift boss should also make out a report, intended personally for the other boss. This other boss should notify each man on his shift working in such a place of the missed hole." —*Mines and Methods*, Jan., 1910, p. 190. (H. A. W.)

AN AUTOMATIC SAFETY CLUTCH FOR CAGES.— "At the recent meeting of the North Staffordshire Institute of Mining and Mechanical Engineers, held at Stoke-on-Trent, J. H. Ruthven read a paper on his invention of a safety clutch, designed to prevent a cage from falling in the event of the rope breaking.

The apparatus he proposed consisted of a notch bar which was thrown up into action by means of a spring, this spring being held in tension by the hoisting gear. Notch bars had been used before, but usually attached to the cage by a fixed pivot. In this clutch the pivots on which the notch bar moved were not fixed, but were movable, being at the end of links, and the other end of each link had a pivotal joint connecting it with the attachment to the cage. The various parts and joints bore some resemblance to the human arm, the movable attachment to the cage corresponding with the shoulder joint, the pivotal attachment to the clutch bar being the elbow joint, while the gripping notch represented the grasping hand. By means of these various parts, the grip possessed a power of accommodating itself to any oscillations of the cage, and when once the grip was secured, there was nothing to cause it to release itself until the cage was once more lifted up.

The weight of the cage was the power that held it in position, and the heavier the cage the more powerful the grip. In high-speed cages descending, the velocity imparted a tremendous force which had to be reckoned with, and a too sudden stop would be disastrous, not only to the occupants but to all the surrounding structures. On the breaking of the hauling rope, an instantaneous action must take place by means of which the speed was first of all retarded and finally stopped. The force applied must get gradually stronger in proportion to the force required. As the momentum decreased, the force required to stop the cage was also decreasing, but the aim was to arrange the grip so that its power became relatively stronger. The power of the grip depended upon leverage, and this leverage varied according to the distance apart of the gripping surfaces as compared with the distance between the notch and the pivotal attachment of the notch bar to the links.

By separating the gripping surfaces the leverage was reduced, and therefore the gripping power, and the grip was arranged to be just sufficient to retard the cage but not to stop it. These gripping surfaces consisted of a series of plugs, two or more, of soft metal, fitted into shoulders on the notch bar. The first pair to strike must be furthest from each other, and this pair would take the force of the first impact and would be smashed and bent: if made of a readily fusible alloy the balance would melt away by means of the heat generated by the friction during the safety slip, and become ineffective as a grip. During this process a considerable amount of retardation occurred. The second pair, which were closer together, then formed the grip, and in this case the leverage was stronger; therefore, the retarding force was greater, and it was now safe to apply this greater force because the momentum had diminished. As the momentum all the time was becoming less, when these were finally melted away, the grip proper, that is, the iron notch bar itself, came into use. Where tremendous weights and extremely high velocities were dealt with, a still further safeguard might be introduced to minimise the first impact, which is the dangerous one, by employing oil buffers instead of the plugs of soft metal or fusible alloy' the buffers occupying the same positions as the plugs, and coming into action in the same order." —*Mining World*, Jan., 1910. (C. B. S.)

HUMIDITY OF MINE AIR.—"One of the questions now before the Belgian Chamber of Representatives, in connection with the proposed remodelling of the law respecting the working of mines, is the regulation of the hours of labour. The Belgian Colliery Owners' Federation recently addressed a letter to members of the Legislature calling attention to the need of recognising the importance of the humidity factor in any proposal to limit the hours of labour in high-temperature mines. They contend that the suggested limitation to eight hours of work in seams having a temperature of 28° C. (82·4° F.) should not apply unless at the same time the moisture reaches 85 %. According to the proposal of the Minister of Public Works, as given in a special supplement to the *Echo des Mines et de la Metallurgie*, the whole of the various regulations for working mines in Belgium are to be codified into a single Act, embodying several important modifications, of which the question of the restriction of hours of work is at the moment the most prominent. The original proposal, as stated in Article 103, was to forbid altogether the working of seams, except in the case of absolute necessity, where the temperature reaches 35° C. (95° F.) on the dry bulb thermometer or 30° C. (85° F.) on the wet-bulb thermometer; and where the wet-bulb temperature exceeds 25° C. (67° F.), it was proposed to limit the hours of productive work to six. This restriction, if carried out, would be most serious for the colliery owners. We are not in possession of the average reading of the wet-bulb thermometer in Belgian pits, but from the information given by M. Denoel, before the Royal Commission on Coal Supplies, it may be gathered that the above-named limits of temperature and moisture are approached, if not exceeded, in many cases. Applied in Great Britain, such a restriction would close some of our most productive pits.

From a report of a recent conference of the Socialist Parliamentary Party, we understand that these drastic measures have now been delayed by the action of the Centre. The whole question of the temperature and humidity of working places is to be made the subject of an enquiry by a special commission.

It is satisfactory to find that the Belgian Government has not been rushed into hasty legislation upon so important and so difficult a point as the physiological effects of working in high temperatures. It is a question to which much attention has been given in our own collieries, and there has been a revival of interest in it in connection with the suggested rules for compulsory watering as a palliative of the coal-dust danger. In the second report of the Royal Commission on Mines emphasis was placed upon this aspect of the case. we may, for the present, dismiss the influence which moisture undoubtedly has upon the development of *ankylostoma* in order to dwell upon the purely physiological effects of working in a hot atmosphere, the humidity of which approaches saturation. In moderate temperatures, experience shows that the humidity of the air has little effect upon bodily health or capacity to work. It is different, where the wet bulb temperature reaches 80° F., or thereabouts. We are there approaching the limit at which the human constitution can bear excessive fatigue without exhaustion.

The temperature of the dry bulb thermometer appears to make far less difference upon human powers of endurance. A dry temperature of 100° F. is not in itself injurious to health, so long as the wet bulb thermometer is below 80° F. But if the humidity is such that the latter is above 80° F., exertion causes a serious rise in body temperature. The fact that heat apoplexy is almost unknown amongst miners is perhaps to be explained by the fact that those who are habitually subjected to conditions under which such a result might supervene, are careful to avoid over-exertion, doing less work, with more frequent periods of rest in cooler air. But the death of John Welsby at Hampstead Colliery, under circumstances where duty impelled him to overtax his powers of physical endurance, as well as the frequency of heat stroke amongst mine horses, show that the possible physiological effects of saturated air at high temperatures are not to be lightly dismissed.

Interesting information bearing upon this matter has been recently published in the November *Bulletin* of the American Institute of Mining Engineers, in which Mr. G. J. Young describes the ventilating system at the Comstock Mines, Nevada. These mines are notably hot, and the relative humidity in the working places is often high. Yet the miners appear to suffer no detriment to health. Professor P. Frandsen, of the University of Nevada, has made a study of the physiological effects of these atmospheric conditions, and states that the most striking feature is the uniformly high hæmoglobin index of the miner's blood, the number of leucocytes being normal. Much of the healthiness of these men is attributed to the care with which the miners observe the precautions prescribed by the management. Especially is this necessary with regard to exposure to sudden changes of temperature. Frequent rests, plentiful bathing, with shower baths, hot and cold, on leaving work, and other practices, tend to neutralise any harmful physiological tendency exerted by their employment.

We learn, in fact, a practical lesson from these mines, where no legislative restrictions have been found necessary to prompt both miners and management to take proper precautions. The Royal Commission on mines have stated in plain language that, in their opinion, no good object would be served by prescribing a limit of wet bulb temperature for the carrying on of work in mines. And this conclusion will be endorsed by all practical men.

Humidity is not always easy to control, since not only the condition of the mine, but also the degree of saturation of the ventilating current, comes into play. A hard and fast rule restricting the working of seams showing a certain degree of saturation of the mine air would result, in some cases in the curious position that such seams could sometimes be normally worked and at other times would be legally restricted.

It will be highly interesting, therefore, to watch the proceedings of the Belgian Commission, and to examine the conclusions to which that body may eventually come. These conclusions will, in due course, be published in the columns of the *Colliery Guardian*." — *The Colliery Guardian*, p. 1297. (C. B. S)

Humidity and Ventilation. "The Comstock has been considered the hottest group of mines in the world, a water-temperature as high as 160° F. having been reported from the Ward shaft, which has reached a depth of 2,500 ft. Of late years, considerable attention has been paid to ventilation, and underground conditions have been much improved. The paper contains a full record of air and water temperatures taken over a considerable period at varying depths. With the exception of the Ward shaft, all of the ventilation is effected by upcast shafts, supplemented by small fans for local distribution of the air in drifts, raises, stopes and winches. The Ward shaft is divided by a brattice, and one compartment (the pump) is used as an upcast, the other two compartments serving as a downcast. An exhaust fan is attached to the upcast compartment. The total amount of upcast air measures 235,835 cub. ft. per minute, and the amount of downcast air measures 216,687 cub. ft. per minute.

Perhaps the most interesting section of the paper is that dealing with the question of humidity. At the Comstock the air entering during the day in summer is of low and in winter of moderate humidity, but the air-current begins to rise in temperature and absorb moisture as soon as it enters the underground workings. An idea of the amount of water taken up by the air-current may be gained by considering the total amount of the moisture discharged with the air at the upcasts. The total amount in all upcasts is 519,788 lb. per twenty-four hours, of which it is calculated 391,798 lb. are taken up by the air-currents. It is a well-known fact that the control of humidity and heat is a matter of considerable importance as affecting the efficiency of the miners. Upon the Comstock the carrying of air through sheet-metal conduits of moderate cross-section has been found to be the only practicable way of delivering air-currents of low humidity to working places where a high temperature prevails, and this method is in general use. The quantity of air varies from 500 to 1,200 cub. ft. per minute in ordinary cases, up to from 2,000 to 4,000 cub. ft. per minute in cases in which an excessive temperature is to be overcome. The humidity of the discharged air depends upon the humidity of the air entering the pipe or fan and the temperature of the discharged air, and for this reason the air should be drawn from a current low in both temperature and humidity. The relative humidity of the discharged air ranged from 20 to 41%, while the relative humidity at the face, usually 20 to 25 ft. beyond, ranged from 22 to 74·4%. In one abnormal case in the 2,350 ft. level the moisture charged the air-current to saturation, and raised the temperature 27·6° F. in a very short distance. Some observations on the cooling effect of large volumes of air were made. The most striking results were in the case of the hot drift above referred to. Before the connection was made the maximum temperature of this drift varied from 126 to 130° F., and this in spite of forcing 4,160 cub. ft. of air per minute into the face. After making the connection and before boarding off the walls, the temperature dropped to 110°, with a volume of air passing of 10,350 cub. ft. per minute. After snow-shedding, the temperature dropped to 98·6°, with 11,667 cub. ft. per minute passing and a relative humidity of 98·8%.

In conclusion, Professor Young has an interesting chapter in which he discusses the physiological effects of working in high temperature and humidity. The main facts brought out by previous observers have been that the result of such conditions is a notable rise in body temperature: that all the bodily organs with the exception of the stomach are stimulated: that prolonged exposure to such conditions undoubtedly lowers vitality, but that intermittent exposure produces no permanent ill-effects: that these conditions cannot be considered an important cause of lung diseases; and lastly, that under abnormal conditions loss of mental control and serious bodily disturbances result. Local physicians have informed Professor Young that the average life of the Comstock miner approximates twenty-five years, and that the miners do not show any greater susceptibility to any particular disease than the residents of the town. The result is due in a large measure to the fact that the miners observe certain precautions, and that the mine management provide the necessary facilities. Miners are careful not to expose themselves to cold draughts, since they work stripped to the waist. On passing from a hot to a cold place a heavy coat is used to protect the heated body. Wet clothes are either removed before going to the surface, or trousers and coat worn over them. All miners take hot and cold showers after coming off shift. Frequent drinking, and the bathing of the hands, wrists, arms and head in ice-water are resorted to in all hot workings. Frequent rests are taken in special cooling-rooms, so placed in the workings as to receive the freshest and coolest air. In exceedingly hot workings cold water from a hose is turned upon the miner while at work.

The effect of the underground conditions upon the blood has been made the subject of a study by Professor P. Frandsen, a note by whom is given in an appendix to the paper. His conclusion, given tentatively, is that the condition in the workings of the Virginia City mines do not have any particularly detrimental effects upon the composition of the blood. The main permanent effect appears to be an increased hæmoglobin content of the individual red corpuscles. Professor Young, as the result of his own underground experience, comes to the conclusion that moderately high temperatures, from 95 to 105° F., with moderate humidities, from 50 to 70 relative humidity, and with air-currents of velocities from 200 to 300 ft. per minute, do not prevent work nor are particularly uncomfortable. A higher temperature, from 110 to 115°, together with the same conditions, decreases efficiency to a considerable extent. A high temperature, from 110 to 115°, with high humidity and moderate velocity air currents, very greatly impairs the miners' efficiency, and a still higher air-velocity, under the same conditions, renders work more bearable, but miners cannot work very long at one time. A moderately high temperature, from 95 to 105°, in a saturated atmosphere with no current, becomes very trying. Prolonged exposure with much exertion is dangerous. A moderate temperature, from 90 to 98°, and saturated air-currents of a velocity of from 400 to 500 ft. per minute, with more or less vitiated air, are conditions which are very trying, and give a low labour efficiency. Vitiated air will impair labour efficiency to a greater extent than a high temperature."—G. J. YOUNG. *The Colliery Guardian*, Jan. 28, 1910. (C. B. S.)

MISCELLANEOUS.

PROFESSIONAL ETHICS.—"This subject is considered under the several heads of the positions likely to be occupied by engineers—namely, as authors, employees, agents, or private or public advisers.

1. *Authors.*—The author is party to an implied contract with his reader, in which he covenants that he has something to say, and will say it clearly, honestly, and fully. He ought, therefore, to avoid all obscurities or ambiguities of style: 'mixed figures' of rhetoric (which are essentially insincere, because they present, as seen by him, pictures which could not possibly have been sanely seen by anybody —as when Wendell Phillips said the time would come when Liberty would 'stand by every new-born child, and drop in its cradle the school-house and the ballot-box!'): misuse quotation marks, by making them include anything else than *verbatim* quotations; second-hand references to authorities not actually verified by the author, without specific statement of the fact, etc. And a technical author ought, in good faith, to furnish his reader with all the aids to further detailed enquiry, which he himself possesses. In other words, he is bound to give, not only his own conclusions, but also such a statement of his ground and authorities as will fairly show the range of his own knowledge.

2. *Employees.*—Under this head, both duty and policy dictate a loyal devotion to the interest of the employer, and a consideration of duties rather than rights. Even from a selfish stand-point, loyalty commands to-day the highest price in the market.

3. *Agents.*—The obligations of this relation are as clearly defined by law as by personal honour. No agent may have secret personal relations, affecting the interests of those for whom he is openly acting. The payment of commissions to purchasing agents is a case in point. Sometimes, by reason of trade-agreements, a manufacturer cannot sell to a customer below a certain price, yet is permitted to pay the agent, who procures the customer for him. If that agent is at the same time the agent of the purchaser, he cannot honourably receive such a commission without the knowledge of his principal. But, if the rules of trade make this the only way of securing a reduction of price, he may accept such a commission and turn it over to his employer, in which case he should insist upon receiving it in a check to his order, and indorse the said check to the order of his employer, so that there may be no doubt in any quarter as to the disposition which he has made of it.

4. *Private Advisers.*—Under this head, perhaps the most frequent function is the writing of letters of recommendation. Do not give general letters of this kind, if you can avoid it. A letter which the candidate carries in his own pocket, and of which the contents are known to him, has little weight the first time it is exhibited, and less and less weight as time goes on; and it becomes dog-eared and dirty through frequent presentation or pocket wear. Moreover, a letter of personal recommendation is really a piece of advice to an unknown client; and it cannot possibly be made as clear and effective as it ought to be, if it

is drawn in a general form, without knowledge of the particular requirements of the said unknown client. Even for the benefit of your best friend, it is much better to give a letter, authorising him to use your name as a reference, and promising to answer, confidentially and fully, any inquiry that may be made of you concerning him. Such an enquiry will come only from some party seriously considering the engagement of the candidate. When it does come, it can be answered with intelligence and freedom; and if the man is, in your opinion, a good man for the place, he will be ten times as likely to get it, upon such a statement from you, as upon the strength of your vague general recommendation, presented by himself. But in all such cases it should be remembered that the enquirer, not the candidate, is your client: and, even in giving the most favourable opinion, you should carefully state the extent of the knowledge upon which it is based. You have no right to try to get a place for a friend by misleading or incomplete advice to your client, his prospective employer.

Under the head of private advice falls the advice given to clients as to investments. Here honour requires, in the first place, that the client should not be deceived as to your ability to deal with his particular problem. The numerous instances in which scientific experts in one line have undertaken, with disastrous results, to give opinions in another line, furnish sufficient illustration of this proposition. There was a time when experts were scarce in this country, and a chemist was regarded as competent to pronounce judgment upon a piece of mining-ground. But the situation has been so changed that, nowadays, even a trained mining engineer cannot be trusted for such a judgment unless, in addition to his general knowledge, he has also a special acquaintance with the nature and history of the particular kind of mining and the particular district concerned. One of the most eminent mining experts within my acquaintance never undertakes to report upon an enterprise in a region not familiar to him without engaging at his own expense, a local expert of his own selection, to post him as to such matters as his own examination might miss.

I can recall instances of really able and unquestionably honest mining experts who, lacking such local aid, and ignorant of local conditions and history, have reported hundreds of thousands of tons of ore, on the strength of scattered 'float' from a comparatively small outcrop.

It is true, that a thoroughly trained mining engineer should be able, by keen and faithful study, to get at all the essential facts for himself; but he ought to be frank with his client beforehand, and let the latter clearly understand the situation.

Private advice is also dependent upon the nature of the question presented. The client may say, 'Here is the report of the owner of the property. I have agreed to buy it, if his statement be true; and I employ you to verify that statement.' Such a commission is entirely honourable; yet the discharge of it is not altogether agreeable and safe for him to perform it. For the condition stated is narrow, and its honourable fulfilment is liable to serious subsequent misunderstanding. If, for instance, the expert should find, that the representations of the vendor, which he was sent to verify, were true, but that other and equally promising mining property in the neighbourhood could be bought for a lower price, he could not properly include in his report any reference to that more profitable alternative. Yet his favourable report as to the vendor's statement submitted to him, might be construed afterwards as a recommendation of the purchase, which was not really submitted to him at all. This situation occurs, perhaps, most frequently under the English system of organisation of mining companies, according to which a company is organised on the basis of a vendor's statement, for the purpose of purchasing a certain mining property, provided that statement be endorsed as correct by a competent expert, selected to represent the company. Upon the report of that expert, the existence of the company itself, and all the profits of its promoters, depend; and it would be, of course, unprofessional for him to declare that, while the representations of the vendor were true, he had heard of another investment, which he thought was better!

For the mining expert himself, it is highly desirable to insist, if practicable, upon a wider authority, permitting him, not only to verify the vendor's report, but also to advise as to the purchase itself. Either duty, I think, may be honourably undertaken; but in the former case, the expert should take care that his task and responsibility are clearly expressed, especially if his report is likely to be published afterwards, and thus converted from private to public advice.

5. *Public Advisers.*—Under this head fall the expert reports, which are published in the prospectuses of mining companies, and the testimony of experts in court. Under the first head, I have only to say: (1) That an expert has the right to insist, that his report should not be garbled, by fragmentary publication or otherwise, so as to support a scheme which it did not contemplate or approve; and (2) that his protest against such misuse of his statements should be made promptly. As I said to the Electrical Engineers, 'don't come limping in, after a scheme has failed, to protest that you never meant to approve it in all respects. Speak quick, or shut up!'

Among public advisers are those who appear as expert witnesses in court. The difference between an expert witness and an ordinary witness is, that the former is expected, while the latter is forbidden, to give his opinions, and to state, among the grounds of those opinions, the facts and conclusions reported by others, as well as those personally verified by himself. The expert's position is, that he has been called, as *amicus curiæ*—a disinterested 'friend of the Court'—to assist judge and jury in the interpretation of the evidence, a part of which may be his own testimony as to facts observed by himself. But as to his statement, whether of such facts or of his opinions, the expert should not forget, that he has sworn to state the truth fully and only. It has often been urged that experts should be employed by the court, and not by either party. This plausible notion involves the selection of such expert advisers without knowledge of their theoretical views. I have had some experience of the practical working of that plan; and I can testify that it results in opinions, usually voluminous and vague, which need further expert interpretation before they can become useful guides to judge or jury. Moreover, the court may innocently select an expert who is a partisan of a particular scientific theory. On the whole, therefore, I am thoroughly convinced that our present American system, under which expert witnesses are presented by each party, to be cross-examined by the other party, is, with all its inconveniences and costs, best adapted to elucidate the truth and promote the ends of justice. It must be confessed that, under that system, the expert witness becomes an advocate; for neither party in a law-suit would produce such a witness without knowing beforehand, that his opinions were favourable to its contention. Yet it still remains true that the expert witness is, after all, a sworn adviser of the court. Under these circumstances, what is his professional duty?

I have answered this question by declaring:

a. That no expert can honourably serve in such a capacity, without having previously made sure, for himself and his clients, that he can honestly defend their contention.

b. That if, after having, on a preliminary retainer, studied the case, he cannot do this, he should decline to appear as a witness in court, either for that side or the other.

c. That his preliminary study of the case should be so thorough as to warrant him in propounding and defending in court his expert theory. He may fairly appear as a partisan of that theory, though not of the party aided by it. And, in defending his view, he is still under oath, and must not shrink from the candid acknowledgement of new evidence presented by the other side, not previously known to him, or deal with such evidence evasively or controversially, for the purpose of protecting his own reputation. He is justified in saying that he has made up his deliberate opinion upon all the facts observable or ascertainable by him, and cannot change it upon the presentation of new evidence, which he has had no opportunity to test and weigh.

Above all, the expert witness should not try to be smart or witty under cross-examination. In such an encounter his hands are tied; for he is under oath, while the inquiries and insinuations of his antagonist are unrestrained.

The best expert witness I ever heard courteously assumed, that the cross-examining attorney was sincerely in quest of information, and gave it to him with fraternal and pedagogical cordiality which simply paralysed his attack! I have seen many a witness beat a lawyer in repartee, and lose the case for his client; and I have seen many an honest witness confused and embarrassed under cross-examination—but the jury saw it also, and gave the verdict to his side!

It frequently happens that an expert is required to answer categorically, 'Yes' or 'No,' a question which cannot be thus answered without giving place to unjust misconstruction. In such a situation, the expert witness should not 'fence' or 'dodge' (thereby giving the impression that he is afraid of the inquiry), but simply give the categorical answer required, and then ask the privilege of further explanation. This the court will always grant, to his complete protection and the discomfiture of the ingenious cross-examiner.

It happens sometimes, especially in mining lawsuits, that an expert, who has expressed in court a certain opinion is forced by later developments to change it. For instance, he may have testified to separate occurrence of two lodes, while deeper mining has afterwards shown that these two, uniting, have become one—with lamentable and disastrous results, perhaps, to conflicting claims of ownership under the U.S. laws. Now, such changes of opinion upon later evidence are very common and perfectly honourable. And an expert, who has once honestly advocated in court a given opinion has a perfect right to retract that opinion afterwards. But if he be wise, he will not announce that change as an expert under fee, in another law-suit. Let him state it in a technical essay or lecture, if he thinks it worth publishing—but not before a jury, to whom it would have to be explained along with the unpleasant circumstance, that both the erroneous first opinion and its later correction have been paid for. Moreover, an expert offering such an explanation in court will be forced either to confess that he came to a wrong conclusion the first time, or else to assert that his opinion was correct then, but has been changed by later evidence—in other words, that he might change it again upon further evidence. Such declarations will inevitably have an unfavourable immediate effect upon the case of his client, and a permanent effect upon his own reputation. This does not apply with equal force to a changed opinion, which has been already publicly declared elsewhere. Yet, even in that case, it is not advisable to appear in court as an expert contradicting one's former testimony. To give voluntary expert testimony against one's former *clients* is, of course, dishonourable. But when this consideration is not involved, an unnecessary recantation, under oath and fee, is still inadvisable. In view of my former statement that the expert witness is a partisan, not for a *side*, but for an honest *opinion*, it might fairly be said that his opinion is really his client, and that if it has been overthrown he had better not become the partisan of another opinion before a judge and jury."—R. W. RAYMOND.—*Bulletin Trans. American Institute of Mining Engineers*, No. 37, Jan., 1910, p. 43. (J. A. W.)

PLATINUM RESISTANCE THERMOMETRY AT HIGH TEMPERATURES.—"The subject is treated very fully and a very large number of experimental data are given, especially in the appendix. Very detailed accounts of the methods of experiment are also included. The subject matter comprises methods of measurement, construction of thermometers, calibration of thermometers, determination of their constants, measurement and reproducibility of freezing and melting points, correction of scale for thermometers of impure platinum, calibration formulæ, changes in thermometer constants at high temperatures, comparison of resistance and thermoelectric scales, the palladium thermometer, the boiling point of sulphur and other matters. The authors arrive at the following conclusions:—Temperatures defined by the resistance thermometer of pure platinum calibrated by Callendar's formula at 0° C. 100° C., and the boiling point of sulphur (444·7° C.), agree with temperatures on the generally accepted gas scale over the range 0°–1100° C. to within the degree of reproducibility of the latter. Thermometers of impure platinum, calibrated in the same way, give a temperature scale, the divergence of which from the normal scale increases with the temperature. If thermometers of impure platinum are calibrated at a fourth temperature, such as the freezing point of silver, and the difference coefficient of the Callendar formula is written $d = a + bt$, then the scale so defined is in very close agreement with that of the normal platinum thermometer. The palladium thermometer, calibrated by the Callendar method, agrees with the platinum thermometer up to 600° C., to within 1° C. It requires a fourth degree equation to express its resistance-temperature relation from 0° C. to 1100° C. The following freezing points were determined:—Tin, 231·92° C.; cadmium, 321·01° C.; lead, 327·43° C.; zinc, 419·37° C.; antimony, 630·71° C.; aluminium, 658·0° C.; the eutectic mixture, Ag_3 : Cu_2, 779·20° C.; silver, 960·9° C.; copper, 1083·0° C.; the eutectic mixture, Cu : Cu_2O, 1063·2° C. The average deviation for the pure metals ranges in the several cases from 0·02° C. for tin, cadmium and zinc to 0·5° C. for aluminium and copper, and the reproducibility figure ranges from

0·06° C. for tin and zinc to 2·3° C. for antimony. The ordinary thermo-couples of platinum and iridium-platinum calibrated at three temperatures in terms of the formula, $E = a + bt + ct^2$, are in agreement with the platinum resistance scale, in the interval 200° C.–1100° C., to within 0·3° C. At high temperatures the constants of platinum thermometers undergo gradual changes, which are greatly reduced, but not eliminated, by annealing for several hours at a temperature higher than that for which they are to be used. After annealing, the changes in the constants are much less for thermometers of pure than of impure platinum: they are least for pure platinum wires of large diameter and free from strains. The freezing points of the metals are the most satisfactory calibration temperatures. The temperature of separation of the silver-copper eutectic is an entirely satisfactory fixed point, as is also that of the copper-cuprous oxide eutectic, if proper precautions are taken. The boiling point of sulphur at known pressure and under carefully specified conditions is a most satisfactory fixed point, reproducible to within a few hundredths of a degree."—C. W. WAIDNER and G. K. BURGESS. *Bull. Bureau of Standards* (U.S.A.), 1909, *6*, 149–230.—*Journ. Soc. of Chem. Industry*, Vol. xxix., No. 4., Feb. 28/10.; p. 205. (J.A.W.)

REPORT ON EXPLOSION, &C., AT KYNOCH'S WORKS, DURBAN, NATAL. DECEMBER 7TH, 1910.—"The explosions occurred in the weighing and mixing houses, three Europeans and eight natives being killed. 2,000 lb. of nitroglycerin and 100 lb. of collodion-cotton were present in the weighing house, and in the mixing house, 800 lb. of blasting-gelatin in the machines. Whilst nitroglycerin and collodion-cotton were being weighed off, the latter became ignited, and a few seconds later the nitroglycerin in the building detonated. This was followed, 15 or 20 seconds later, by another explosion in the mixing house situated 320 ft. away. The structural damage caused by the explosion was considerable. At the site of the weighing house a crater 3 ft. deep and 18 ft. wide was formed. Two craters, 5 ft. deep by 9 ft. wide, marked the site of the mixing house. All buildings within 1,000 ft. radius were affected, and in most cases the roofs had been sucked down and the walls bulged out towards the centres of the explosion. The probable causes of the explosion in the mixing house are reviewed. That spontaneous ignition of the collodion-cotton, due to imperfect purifying, was the cause, is negatived by the results of the tests carried out with samples from the same batches of collodion cotton and blasting-gelatin. The collodion-cotton gave a heat test of 27 minutes, and the blasting gelatin one of 21 minutes. Mercury compounds were proved to be absent. The inspector arrives at the conclusion that ignition due to friction on the scales was the cause of the explosion. The collodion-cotton was first weighed out, and nitroglycerin from the filter-tank poured on, until the beam of the scale rose with the correct weight. The operation caused the production of a considerable quantity of dry collodion-cotton dust. The beam was fitted with two sliding weights, one of them weighing 6 lb. There was very distinct evidence that ignition of the collodion-cotton took place during weighing. The beam or its graduations would be coated with collodion-cotton dust, and in the inspector's opinion, the friction between the sliding weight and the beam would have been quite sufficient to cause ignition of this dust. The explosion in the mixing house was undoubtedly due to projected *debris*. To prevent accidents of a similar nature, the following alterations have been made in the factory: (*a*) Nitrocotton is to be weighed in a special annexe to the dry cotton store, and sent to the hand-mixing house in flax bags. All weighing parts of the scale are as far as possible to be thoroughly and efficiently protected, and the platform suitably covered. (*b*) Nitroglycerin is to be measured out and not weighed. (*c*) Mixing tables are to be covered with cork-linoleum to minimise friction, and all collodion-cotton boxes and benches are to be covered with soft material for the same purpose. A special porch for collodion-cotton is to be added to the weighing-house, so that only minimum quantities of this ingredient will be introduced into the main building."—W. H. PAY. *The Natal Government Gazette*, 1910, *3803*, 38-41. *Journ. Soc. of Chem. Industry*, xxix., 4. Feb. 28/10; p. 235. (J.A.W.)

THE OLD GRANITES OF AFRICA.—"(1) In South and Central Africa there exists a fundamental granite-gneiss formation which form the nucleus or core of the continent, the base of the geological column on and around which the younger formations have been built up, and although later portions are intrusive in the Swaziland Series, nevertheless it underlies and supports it in much the same way that the fundamental or Laurentian gneiss of Canada is intrusive in and yet underlies and supports the Grenville Series. In Eastern Canada, however, the original basement upon which the first sediments were laid down has nowhere been discovered, having been everywhere torn to pieces by the Laurentian granite intrusions; whereas in South Africa later portions of the old granite are intrusive in both the fundamental gneiss and also in the oldest existing sediments, and there is no reason to assume the pre-existence of still older sediments; but, as we have seen, there is good reason to regard the old gneiss as the original basement on which the oldest existing sediments were built up. This fundamental granite-gneiss formation is especially well developed in the region of the Limpopo, in German South-West Africa, in German East Africa, in parts of the Congo Free State and of the Nile Basin, in Mashonaland and Matabeleland, and also in Natal and Zululand.

(2) It follows from (1) that the old granites of South Africa, although all derived from similar, probably the same, magma, are not all of exactly the same age. Thus it would seem that the old granite is intrusive in the Swaziland Series by after-protrusions due to the fracturing and crumpling of the old granite crust produced by secular cooling. These intrusions have occurred at various intervals extending over vast geological time. For example, Dr. Molengraaff has shown that the old granite boss of Vredefort is probably more recent than the Pretoria formation.

(3) The increasing prevalence of granite and gneiss as one travels northwards towards Central Africa has already been noted. To the south and south-east the sediments are well developed. The elevation of the more northern portion and the progressive subsidence of that to south and south-east (and consequent depositions and building up of the sediments of these latter areas) may probably be due to the same causes as those recently suggested by Dr. Adams to explain the progressive uplifting of the Laurentian Highlands of Eastern Canada and sinking of the great plains to the south, namely, the uprising of the granite-magma from the depths accompanied by its withdrawal from below the subsiding areas.

(4) The gneisses of this formation are of igneous origin, and are probably granites altered chiefly by regional metamorphism. Possibly, however, some of the later gneisses, intimately associated with the crystalline schists of the Swaziland Series, are paragneisses, that is to say, are of sedimentary origin.

(5) The sediments of the Swaziland Series are derived from this old fundamental formation, and possibly also in part from ocean-borne detritus.

(6) The Witwatersrand System has been derived from the Swaziland Series, and the old granite which is intrusive in it, but probably to only a very small extent, if any, from the further removed fundamental granite-gneiss formation."—C. B. HORWOOD, *Geological Magazine*; *S. African Mining Journal*, Feb. 19/10, p. 631. (A. R.)

Abstracts of Patent Applications.

(C.) 386/09. John Acheson Jones. An improved tube mill liner. 23.8.09.

This specification describes a form of tube liner with detachable and renewable projecting pegs.

(C.) 116/10. Carl Giesecke. Process and apparatus for dressing ores. 18.3.10.

The principal feature of this application is a scheme for ore reduction (without specific reference to the means employed for the actual extraction of the precious metal) as explained in applicant's claim 1, which reads:—

"A process of dressing ores of noble metals, particularly those containing gold and silver, characterised by the ores being ground to a thick paste or slime, consisting of exceedingly fine particles, in one uninterrupted single non-cyclic series of operations with an admixture of approximately 33⅓% of water in ball drum-mills, whose chambers, whether separated or united, diminish successively in size (stamps, sieves and amalgamating apparatus as well as intermediate classification on spitzkasten not being used). This slime has added to it at the conclusion of the grinding operation an admixture of approximately 500% to 1,000% of water, and being intimately mixed with the latter in devices before it is supplied to the lixiviation plant."

Claim 2 is for a rotating mixing drum forming part of the tube mill employed as in claim 1. Claim 3 is for the mixing drum referred to in claim 2 of a larger diameter than the body of the grinding mill. Claim 4 is for the partition wall between the main mill and the mixing mill, and claim 5 is for a collar-like extension to the mixing mill for conducting the mixed pulp into the tail launder.

The inventor claims the process of feeding a tube mill loaded with steel balls with dry ores broken to a size of 60 mm. (2⅜ in.) mixed with ⅓ of its weight of water and completing the whole reduction apparatus in the one machine at one pass without intermediate classification. Claims 2, 3, 4 and 5 are all subsidiary and dependent upon the main claim 1.

(C.) 168/10. Peter Arthur Walsh. Improvements in tube mill linings. 14.4.10.

This application relates to improvements in tube mill liners, which consists in interspersing pieces of steel or other metal throughout the liner of flints, pebbles, etc., which are cemented in position to form a smooth surface. A further claim describes the means of effecting this in using rings and boards to form a mould for the liner.

(C.) 182/10. C. G. Patterson (1), The Butters Patent Vacuum Filter Co. (2). Improvements in filters for metallurgical and other uses. 22.4.10.

This application relates to an improved filter which differs from existing forms in several essential points of construction. In the form submitted, the filter leaf takes the form of a number of units held in a frame. Each of these units consist of a "tubular septum" or filter bag distended with tubes (one of which is wedge shaped).

The advantages claimed for this construction are, that it allows of ready access to both sides of the filter and facilitates the discharge of the cake. Also the cake may be discharged by air instead of water, and provides for the interchange of a smaller proportion of filtering area for repairs, etc., than in the present form of leaf.

(C.) 214/10. John van Nostrand Dorr. Improvements in decanting apparatus. 6.5.10.

The inventor claims using a settling tank having either a flat or sloping bottom, with a central discharge opening; in this tank he places a central column with radial arms on which are attached scrapers, set at an angle with reference to the arm that carries them. These scrapers gently push or carry the settled slimes towards the central opening which discharges them.

(C.) 220/10. Byron Laport Morrison (1), Joseph Wesley Ady (2). Improvements in tube mills.

This application relates to improvements in tube mill construction, the subject of which comprises "a barrel with feed opening at one end and driving head at the other, the ends connected by a shell having a series of perforations, a lining comprising rectangular bars, notched strips between the bars and shell, said strips formed with flanges turned downward between the bars and having the notches therein registering with the perforations in the shell.

(C.) 224/10. William Morley Martin. Improvements in treating complex ores or their concentrates.

This is a process for the treatment of complex ores or their concentrates consisting in mixing the subdivided ore with an oxidising agent such as sodium nitrate, which may be in solution or as a solid.

This mixture may or may not be subsequently heated.

By the above treatment it is claimed that a better separation of the constituent minerals can be obtained by water concentration

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 270/10. Jehn Berry. Improvements appertaining to winding and hauling engines. 8/6/10.

(C.) 271/10. Frank Drew Winkley. Improvements in pumps. 8/6/10.

(P.) 281/10. James Hamilton. Improvements in pressure regulating devices for use in connection with air compressors and the like. 14.6.10.

(P.) 282/10. Edmond Erskine. A process for obtaining carbonate of lead from silica ores, sand stone, lime stone or other allied lead-bearing ore. 14.6.10.

(P.) 283/10. Herman Stadler. Improvements in pulp classifying, concentrating and dewatering apparatus. 14.6.10.

(P.) 287/10. Hermann Georg Spengel. Improvements in deep boring. 16.6.10.

(P.) 288/10. Hermann Georg Spengel. Improvements in rotary deep boring apparatus. 16.6.10.

(P.) 290/10. David Gavine Hunter (1), Thomas George Murton (2). Improvements in or relating to the mantles and crushing heads of gyratory rock crushers and the like. 18.6.10.

(C.) 291/10. Thomas Edwards. Improvements in dust preventing ore dropping apparatus and the like, for roasting furnaces. 20.6.10.

(P.) 292/10, Robert James Galt. Improvements relating to shaking tables for ore treatment. 21.6.10.

(P.) 293/10. Samuel Mann. Improvements relating to tube mills. 21.6.10.

(P.) 294/10. Donald Kennedy Adams (1), Alex. William Crichton (2). Improvements in acetylene gas lamps. 22.6.10.

(P.) 295/10. Rowley Alfred King. Improvements in machine rock-drill bits, and hand rock-drills. 22.6.10.

(P.) 296/10. William Worby. Appliance for collecting the dust from machine rock drills. 23.6.10.

(P.) 298/10. Siemens, Limited (1), Max Carl Guenther Schwabach (2), Ernst Gustav Wehausen (3). Improvements relating to electric hoists. 23.6.10.

(P.) 299/10. Hermann Georg Spengel. Improvements in cushioning means for reciprocating pumps. 23.6.10.

(C.) 300/10. Robert Crawford Lyness. Improved contact finger for electrical controllers and switches. 24.6.10.

(C.) 301/10. Joseph John Stocker. Improvements in blowers. 24.6.10.

(C.) 302/10. Charles Anderson Case. Desulphurising system for refractory ores. 24.6.10.

(P.) 303/10. Hans Charles Behr. Improvements in pneumatic ore stamps, hammers and the like. 27.6.10.

(C.) 304/10. Johann Wilhelm Krahe. Improved germ destroyer of all kinds of microbes or germs in animals, and human beings, on the face of the earth. 28.6.10.

(C.) 305/10. Janus Simpson Island. A process of extracting precious values from ores. 21.6.10.

(P.) 306/10. James Lester Clark. Improvements in pipes and launders suitable for conveying abrasive or corrosive material. 30.6.10.

(P.) 307/10. Kenneth Somerset Bell. Improvements in washing tables for ore and the like. 30.6.10.

(C.) 310/10. The Turbon Patent Fan Co , Ltd. (1), Bernhard Bomborn (2). Improvements in centrifugal blowers or pumps. 30.6.10.

(C.) 311/10. Lewis Condict Bayles. Improvement in rock drills and other direct acting engines. 1.7.10.

(P.) 312/10. John Rorrison Douglas. Carrying roller for steam winding and hauling ropes. 2.7.10.

(P.) 313/10. Donald McArthur. Improvements in the construction of amalgamating plates. 2.7.10.

(P.) 314/10. Frank Robinson. Concrete linings for metal pipes. 4.7.10.

(P.) 315/10. Bertram Spencer Cressy. Improved compound valve attachment. 4.7.10.

(P.) 317/10. Alfred George Bettington (1), Victor Liebman (2). An improved method of digging, cultivating, and pulverising the soil. 5.7.10.

(P.) 318/10. Goven Miller Meikle (1), David Hamilton Evans (2). Improvements in slime separators. 5.7.10.

(P.) 319/10. James Ford Cook (1), Frank Gowyne Alfred Roberts (2). Improvements in the decantation process or ore treatment, and in apparatus therefor. 6.7.10.

(C.) 320/10. Eberhard Brauer. Process for concentrating nitric acid. 7.7.10.

(C.) 321/10. Charles Sinclair Drummond. Improvements in steam generators. 7.7.10.

(C.) 322/10. Nobels Explosive Co., Ltd. (1), William Tuxford Evans (2). Improvements in electric igniting apparatus for fuses and the like. 7.7.10.

(C.) 323/10. Thomas Locker & Co. (1), Thomas Locker (2). An improvement in or applicable to sieves. 7.7.10.

(C.) 325/10. Ottokor Serpek. Improvements in electrical resistance furnaces. 8.7.10.

(C.) 326/10. Ottokor Serpek. Process for manufacture of aluminium nitride. 8.7.10.

(C.) 327/10. Ottokor Serpek. Process for the manufacture of pure alumina by means of aluminium nitride. 8.7.10.

(C.) 328/10. Otto Liebknecht. Improvements in the manufacture of sodium cyanide. 9.7.10.

(C.) 329/10. Robert Taylor. Improvements in ore stamp batteries. 11.7.10.

(P.) 331/10. Robert Seymour Benson (1), Head, Wrightson & Co., Ltd. (2). An improved coal washer. 12.7.10.

(P.) 332/10. Harry Gumpold. Composition for making fuse lighting torches and analogous uses. 12.7.10.

(P.) 333/10. John Ebbs. Draughting and surveying instrument. 12.7.10.

(C.) 334/10. Joseph Arthur Ford Glover. Improvements in gas and like cooking ovens. 12.7.10.

(C.) 335/10. Arthur Wilson Ferns. Improvements in acetylene gas generators and lamps. 13.7.10.

(C.) 336/10. Hugh McMenamin (1), Francis Allen (2). Improvements in ore feeding means for the stamps of stamp mills. 14.7.10.

(C.) 338/10. Vincent John O'Brien (1), Thomas Punshon (2), George Alexander Oliver (3), Friedrich Edmund Preiser (4). Improvements in valves for hammer drills. 14.7.10.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

COOK, B., *l/o* Florida ; P. O. Box 26, Roodepoort.

ELLIS, B., *l/o* Johannesburg ; New Rietfontein Estate Gold Mines, Ltd., P.O. Rietfontein Mines.

FORD, S. H., *l/o* Bristol ; 42 Dacre Road, Blackheath, London, S.E.

JOHNS, J. Harry, *l/o* Johannesburg ; Royal Colonial Institute, Northumberland Avenue, London, W.C.

JONES, H. E., *l/o* London ; Rhodesian Exploration and Development Co., Ltd., P. O. Box 213, Bulawayo.

LEAN, T. F. R., *to* Wrightville, New South Wales.

NIXON, G. H., *to* Post Office Vaalhoek, district Pilgrims Rest.

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Proceedings

AT

Ordinary General Meeting, August 27, 1910.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the South African School of Mines,* on Saturday evening, August 27th, Dr. James Moir (President) in the chair. There were also present :—

70 Members : Prof. G. H. Stanley, Messrs. F. F. Alexander, R. Allen, Tom Johnson, A. Richardson, G. O. Smart, H. A. White, Prof. J. A. Wilkinson, J. Littlejohn, A. McA. Johnston, E. H. Johnson, W. A. Caldecott, E. L. Adams, H. A. Adams, W. T. Anderson, A. J. R. Atkin, S. Beaton, G. H. Beatty, W. Beaver, B. V. Blundun, A. L. Chambers, J. Chilton, F. W. Cindel, A. A. Coaton, C. A. Corder, W. M. Coulter, M. Dodd, H. Freedman, R. Gascoyne, J. Gray, W. T. Hallimond, B. J. Hastings, A. Heymann, A. B. Inglis, Dr. L. G. Irvine, A. J. Johnson, A. T. Judge, G. A. Lawson, H. Lea, J. Lea, W. D'A. Lloyd, Dr. D. Macaulay, H. Stuart Martin, E. C. J. Meyer, H. Meyer, J. T. Mitchell, H. Morrell, P. T. B. Morrisby, S. Morison, S. Newton, E. A. Osterloh, E. Pam, S. Penlerick, F. D. Phillips, E. Pinckvos, H. S. Potter, O. P. Powell, W. S. V. Price, J. F. Pyles, C. B. Simpson, J. A. Smith, T. F. Smith, W. H. Smith, A. W. Sly, B. C. T. Solly, S. H. Steels, R. Stokes, J. Stroberg, J. A. Taylor, and C. F. Thomas.

30 Associates and Students : Messrs. A. R. Adams, M. S. Archibald, J. V. G. Clarence, C. J. Crocker, C. A. Damant, A. P. Davis, F. E. Doble, C. Greathead, D. W. Greig, B. de Haas, A. King, G. W. Leach, C. Lipschitz, H. E. Maasdorp, A. McAlister, H. J. v. d. Merwe, G. Musson, E. H. Nellmapius, W. S. New, N. N. Newland, G. H. Olivier, C. O. Palmer, F. J. Pooler, J. M. Robertson, J. R. H. Robertson, R. Sawyer, H. Stadler, M. O. Tillard, W. Waters and A. L. Wright.

About 120 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as printed in the July *Journal*, were confirmed.

NEW MEMBERS.

Messrs. W. Beaver and B. C. T. Solly were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows :—

Cocking, A. S., Messrs. Kynoch, Ltd., Umbogintwini, Natal.

Crowe, Thomas Bennet, Portland G. M. Co., Victor, Colorado, U. S. A. Superintendent, Victor Mill.

Druckes, Arthur Eilert, The Oriental Con. Mining Co,, Paracol, Korea. Consulting Metallurgist.

Freusberg, Adolph, West Sumatra Mijnen Syndikaat, Mangani via Pajacombo, West Coast Sumatra. Technological Engineer. *(Transfer from Associate Roll).*

Hughes, Arthur Robert, Simmer and Jack Prop. Mines, Ltd., P. O. Box 192, Germiston. Mine Foreman.

Hutchinson, James William, Goldfields Consolidated Mines Co., Goldfield, Nevada, U.S.A. Mill Superintendent.

Lawrie, W. W., Village Deep, Ltd., P. O. Box 1064, Johannesburg. Underground Manager.

Lofts, Horace Frederic, *c/o* P. O. Box 1183, Johannesburg. Reduction Works Manager. (*Transfer from Associate Roll.*)

Penlerick, Sydney, East Rand Proprietary Mines, Ltd., P. O. Box 57, East Rand. Joint General Manager.

Ramsay, James Douglas, Rand Klip, Ltd., P. O. Box 2, Benoni. Mine Manager.

Redfern, Alfred, Premier Coal Estates, Brugspruit P. O., Transvaal. Mining Engineer.

Richter, Ernest William, Village Deep, Ltd., P. O. Box 1064, Johannesburg. Shift Boss.

Rush, Frederick William, Geldenhuis Deep, Ltd., Cleveland. Cyanider.

Simpson, junr., George, M. M. Ketahoen, Lebong Soelit, Benkoelen, Sumatra. Metallurgist.

Stroberg, J., Village Deep, Ltd., P. O. Box 1064, Johannesburg. Mine Foreman.

Walton, Arthur John, P. O. Box 149, Johannesburg. Mining Engineer.

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* Formerly Transvaal University College.

Wilson, Harry L., Dolores, Distrito, Mina, Chihuahua, Mexico via Guadalupe *y* Calvo. Mining Engineer.

The Secretary: Since the last meeting of the Society the following have been admitted by the Council:—

As Associates.—

Branthwaite, Stanley, corner of Doran and East Streets, Jeppes Extension. Cyanider.
Courtenay, Benjamin Charles, Geldenhuis Deep, Ltd. (North Section), Cleveland. Cyanider.
Donaldson, Duncan Lane, Geldenhuis Deep, Ltd., P. O. Box 5, Cleveland. Cyanider.
Giddings, Donald Shirley, Portland Gold Mining Co., Victor, Colorado, U.S.A. Chemist, Victor Mill.
Guthrie, Gilbert L., c/o Butters Co., 333 Kearney St., San Francisco, California, U.S.A. Cyanide Engineer.
Howard, T. C. L., Penhalonga Proprietary Mines, Ltd., Umtali, Rhodesia. Analyst.
Jack, William Drummond, Geldenhuis Deep, Ltd., P. O. Box 5, Cleveland. Cyanider.
Jeudwine, Hugh Myddelton, Geldenhuis Deep, Ltd. (West Section), Cleveland. Cyanide Shift Leader.
Kemp, Herbert Sidney, Geldenhuis Deep, Ltd., P. O. Box 55, Cleveland. Junior Amalgamator.
Leach, George William, Geldenhuis Deep (East Section), P. O. Box 54, Cleveland. Cyanider.
Macer, Lionel William, Post Office, Pilgrim's Rest, Transvaal. Mine Surveyor.
MacKenzie, Alexander Wilkie, Elandsdrift G. M. Co., *via* Sabie. Miner.
Maine, Lionel Thomas de, Geldenhuis Deep, Ltd., P. O. Box 5, Cleveland. Cyanider.
Mathews, George Frazer, B.A.(T.C.D.), East Rand Proprietary Mines, Ltd., Box 159, East Rand. Assistant Chemist.
Moir, James Stanley, P. O. Box 62, Knights. Cyanider.
Moore, William, Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Assistant Amalgamator.
Pothecary, George, Sheba Gold Mining Co., Ltd., Eureka. Cyanider.
White, Stanley Rowe, Great Boulder No. 1, Ltd., Mt. Magnet, West Australia. Assayer.
Zipp, William Frederick, Geldenhuis Deep, Ltd., P. O. Box 39, Cleveland. Cyanider.

As Students.—

Allison, William, Geldenhuis Deep, Ltd. (East Section), P. O. Box 54, Cleveland. Mill Learner.
Archibald, Max Stanfield Eaton, Knights Deep, Ltd., P. O. Box 143, Germiston. Cyanide Learner.
Ball, Henry Herbert, Geldenhuis Deep (East Section), P. O. Box 54, Cleveland. Mill Learner.
Broad, Francis Rous, Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Cyanider.
Copeland, Anson William Shaw, Geldenhuis Deep, Ltd. (West Section), P. O. Box 54, Cleveland. Cyanide Learner.
Gibbs, Norman Murray, Geldenhuis Deep, Ltd. (East Section), P. O. Box 54, Cleveland. Learner.
Goldberg, Sam, Geldenhuis Deep, Ltd. (East Section), P. O. Box 54, Cleveland. Mill Learner.
Haas, Bruce de, Geldenhuis Deep, Ltd. (North Section), P. O. Box 5, Cleveland. Cyanide Learner.
Hasserüs, Baldur F. Christian, Geldenhuis Deep, Ltd. (North Section), P. O. Box 5, Cleveland. Sampler.
Hawkins, Thomas Henry, Geldenhuis Deep, Ltd. (East Section), P. O. Box 54 Cleveland. Tube Mill Learner.
Kemp, Norman, Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Cyanider.
Lindhorst, Richard, Geldenhuis Deep, Ltd., P. O. Box 5, Cleveland. Cyanide Learner.
McCaffery, Harold, Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Cyanide Learner.
Merwe, Hendrik Johannes van der, Geldenhuis Deep, Ltd., P. O. Box 5, Cleveland. Cyanider Learner.
Noël, Arthur de Jersey, Geldenhuis Deep, Ltd. (East Section), P. O. Box 54, Cleveland. Mill Learner.
Palmer, Cecil Owen, Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Cyanide Learner.
Shearar, George Linden, Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Cyanide Learner.
Tucker, Bernard Somers, Simmer Deep, Ltd., P. O. Box 178, Germiston. Cyanide Learner.
White, Percy, Geldenhuis Deep, Ltd., P. O. Box 1, Cleveland. Cyanide Learner
Witt, Arthur Bachelor de, Simmer Deep, Ltd., P. O. Box 178, Germiston. Cyanide Learner.

General Business.

The President: We have had letters from Lord Gladstone, who has been good enough to accept the Honorary Presidentship of the Society, and from our four Hon. Vice-Presidents, Hon. J. C. Smuts (as Minister of Mines), Mr. J. G. Hamilton, M.V.O. (President, Transvaal Chamber of Mines), Mr. Lionel Phillips and Mr. J. W. S. Langerman (Past-Presidents, Transvaal Chamber of Mines), accepting office.

With regard to the matter Mr. Crosse raised at the last meeting, as to the procedure at these general meetings, we brought the matter before the Council, and the Council was of opinion that the question had been sufficiently ventilated at last meeting, but that we ought to do our best to accelerate matters at our meetings so as not to waste time. Whenever we get original papers to hand in time before a meeting, the Secretary will try to send out copies to those he knows to be interested.

We have had to abandon the Social Dinner owing to the Union Club being unable to meet us in the matter. They could not guarantee us accommodation, and we could not find another suitable place to go to.

Another matter I want to mention is that we have joined with eight other scientific and technical societies in composing a Loyal Address to the Duke of Connaught, who is due here in about two months. The Secretary has a copy of the proposed Address, if anyone would like to see it. The Society is also going to renew its offer of the prizes to the South African School of Mines which we gave last year.

"PRIMUS STOVES."

Mr. F. J. Pooler (*Associate*): I want very particularly to draw the attention of the Society to a matter which I think we might well take up, namely, the question of the dangers in the use, misuse and abuse of "Primus stoves." It concerns perhaps three classes of people. There are assayers—men who can look after themselves—and the stoves are used very extensively in households, but what I am chiefly concerned with, they are largely used in schools. If this Society does not take the matter up, I do not see who is going to do it. Quite recently there was a serious accident in Jeppe Street, when a lady got very badly hurt. I think a sub-committee of the Council should be asked to take the matter up and report at our next "Chemical" meeting as to the possibility of substituting for Primus stoves, something equally suitable and of course no more or very little more costly, and at the same time perhaps they may be able to report on the general disadvantages of "Primus stoves" There is another point that might be worth considering in the interests of the children generally, whether the attention of the Education Department should not be called to the fact that they are a great danger. This country is not altogether over-populated, and there is no use in putting death traps in the way of the children, especially in our schools.

Prof. J. A. Wilkinson (*Member of Council*): I have much pleasure in endorsing the remarks, which have been made by the last speaker concerning the dangers attendant on the use of "Primus stoves," and am also of the opinion, that this matter may very rightly form a subject for the earnest consideration of the Council of this Society, if only on account of the large number of these articles, which are in domestic use. From the point of view of the heating appliances requisite for laboratories, the nature of the work to be performed must be carefully considered. The means at our disposal may be classified under the headings of solids, liquids and gases. For the prolonged maintenance of high temperatures, such as are required in gold assay work, coke, wood and coal are suitable, but for ordinary chemical work, such as is performed in analytical and teaching laboratories, solids are out of the question, and liquids or gases must be used. The choice in both cases is, however, extremely limited. The only liquids, which are feasible, are alcohol and petrol. The dangers arising from the use of these substances are accounted for by the fact that their vapours form explosive mixtures when mixed with certain volumes of air. Mixtures of gases required for heating purposes may be manufactured from carbonaceous substances, such as, wood, coal, oil, etc. There are few laboratories, however, which are endowed with means sufficient to instal an expensive gas plant for private use. During the last ten years, which has seen the rise of the motor industry and the excessive production of petrol, a cheap method of manufacturing a suitable gas for this purpose has been invented, obtained by passing air through petrol. This gas, known under the names air gas, petrolene, etc., can be used and piped in a manner exactly similar to coal gas, and, as the apparatus is automatic, it needs but little attention. For teaching laboratories more especially it is extremely convenient and useful, and the cost of maintenance small. In conclusion, I may say that I am glad to hear from our President, that the Council will give the matter its attention, and I will therefore defer any further remarks for the present.

The President: Well, gentlemen, I think we are in agreement with what Mr. Pooler says. These things are undoubtedly a danger, and I think the Council will be willing to inquire into the matter and see if we can persuade those in authority to find some better scheme. I should like to know if any assayers have anything to say as to their experience of these things.

Mr. Fraser F. Alexander (*Member of Council*): I agree, as the previous speakers have pointed out, that these things are an abomination, and I believe it would be wise on our part to recommend that the matter should form a plank in the efficiency platforms before the country to-day. No doubt we will find necessity become the mother of invention if legislated upon, and we shall then get something better.

STREET DUST.

The President: There has been some corre-

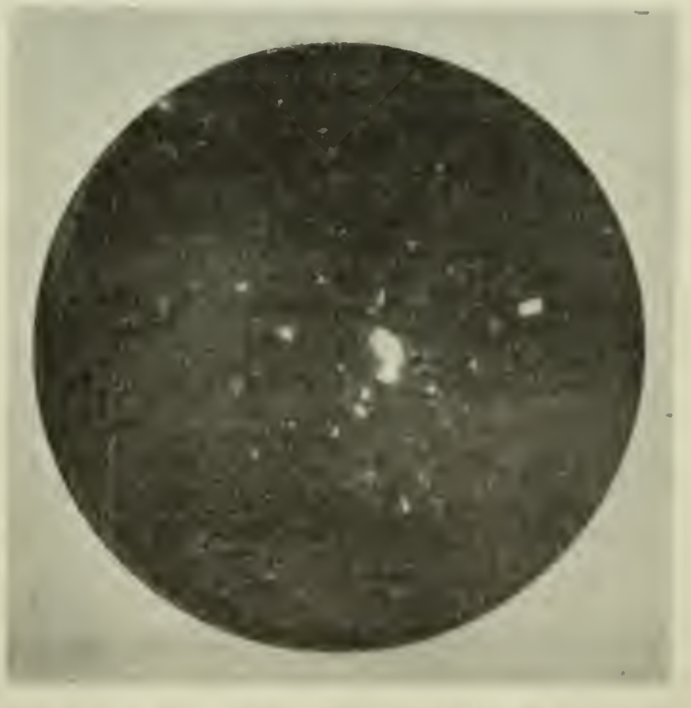


FIG. 1.—Dust from Eloff St.: very jagged particle.

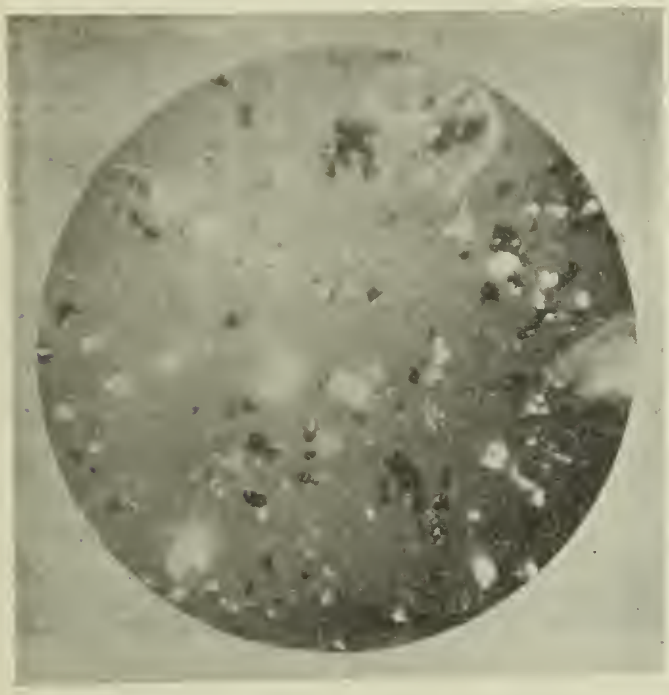


Fig. II.—Dust from Market Square, × 100, polarised. Many angular particles.

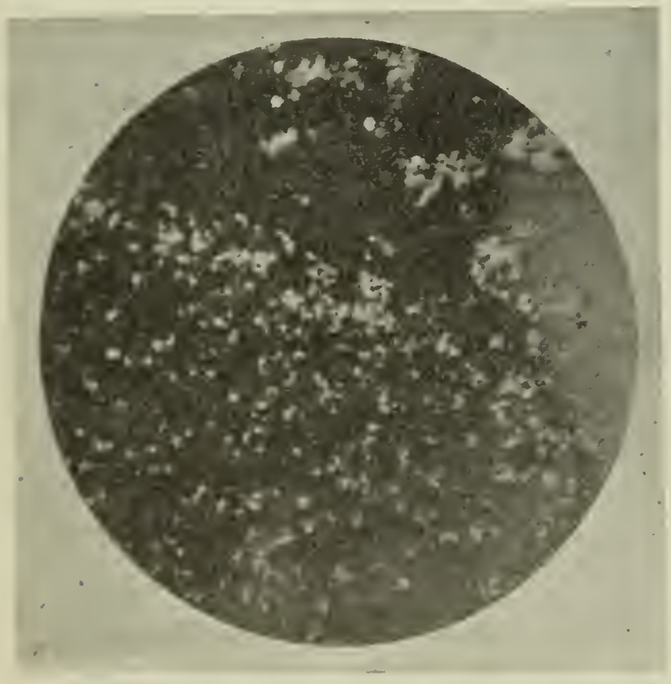


Fig. III.—Finest Dust from Laboratory dark-room (no window). No angular particles.

spondence in the papers as to street dust causing miners' phthisis amongst the inhabitants of Johannesburg. We exhibited some lantern slides of this dust in 1903, and I have some new ones prepared last week which I would like to show you. The street dust is mostly rounded quartz particles stained red with iron oxide, but there is also a proportion of clear angular stuff which has probably come from the tailings dumps.

THE EFFICIENCY OF LABOUR UNDERGROUND.

By Tom Johnson (Member of Council).

The interest I take in increasing the efficiency of labour and machinery must be my excuse for bringing this paper before you.

As you all know, there is plenty of room for improvement ; the trouble is, how to bring it about. I have at various times had a little to say about the matter, and many ways have been proposed, but we are advancing very slowly, especially in regard to underground labour. We hear a great deal about shortage of native labour, yet we are continually wasting it. For some years I have been advocating that more use should be made of *mechanical* means of handling rock, both in shovelling and tramming. I recommended motors for drives as they would be more "flexible" than any other mechanical means, but to make such a scheme come out at its best, we must produce more ore at each stope and on each level. These things are, however, intended to increase the efficiency of the mine as a whole, but to-night, on the other hand, I wish to point out how I think the efficiency of the men themselves could be increased.

Many managers of our mines have tried different methods of increasing the efficiency of the workers, particularly the underground workers, but in dealing with whites they do not appear to have had much success. The general system of paying a standard rate, whether to manager or workman, in my mind tends to decreased efficiency, for the reason that the good worker gets only the same pay as the inferior worker. A standard rate of pay is bad, as men are not standard. The employer does not always try to gauge the abilities of the worker or the amount of work he does, with the result that the good workman is not paid as much as he should be, and the inefficient man gets more than he should, a state of affair which dissatisfies the one and demoralizes the other. Of course, some might say "discharge the inefficient man," but that cannot be done here, for we have not enough good men to go round. Again, what is to become of the poor man? It is not always his fault that he cannot come up to high water level. Pay him according to his worth, then there will be a chance for the good man to get his worth also. Under the present system the good man has no remedy but to try and raise the general wage level. So unions are formed, and the idea is bred that—"they should take who have the power, and they should keep who can," and this is true of both sides—employers and employed. Now, this kind of spirit is not right, and as long as the interests of the employers and employés seem to be antagonistic, there will be conflict. Conflict is mostly unnecessary and we should try to do without it. If one side would not try to get the better of the other there would then be a chance of better doings. It is only natural that workmen should try to get as much money as possible for their work, as it is also for the employer to get as much work done as possible for the money he pays. Nevertheless it is our duty to try and harmonize these conflicting factors.

In day's pay work, if one man's wages are raised others are apt to claim a rise also, and in works where no accurate records are kept of the amount of work that the different men are doing, one cannot blame them. If records were kept the manager would know who deserved an increase, but having no records he does not know those who deserve a rise, and consequently decides not to make any. Accordingly the good man naturally gets dissatisfied and slackens his pace to something like the indifferent man's, that is, he does only just enough to keep his job. In this way the employer who pays at the poor worker's rate gets only, at most, the efficiency for which he pays, thereby losing on his good workmen, who are working below their capacity. It must be understood that by capacity I mean quality and quantity of work accomplished by proper effort, that is, not necessarily by sweating, as the man who makes the greatest fuss is by no means the one who usually accomplishes the most work. We very often buy stores on the same principle, quality being a secondary consideration so long as the goods bear the same name. It is a very wasteful way, especially with workmen, for, as a rule, the difference in quality of workmen is far greater than the difference in quality of material.

Below I try to show how we could make the interests of both employers and employed harmonize. Increased efficiency is the remedy, and this is to be gained by paying each man as near as possible by results, and not by the time actually put in. The manner of payment may be either by day's pay and a bonus for accomplishing a certain task, or contract and bonus; as much contract as possible for it puts a greater premium on individual effort, but it must be carried out in an honourable spirit or it will fail. It is true, foremen and shiftbosses cannot always tell which are their best men as far as turning out best results is concerned, so accurate records should be kept of every man's work as much as possible, then a fair task should be set and a bonus paid to all men doing the task, *also for any work done beyond this task.*

Now, it is also true that many of the young fellows who are "shiftbossing," etc., in our mines have not had sufficient experience to know when a man is doing a day's work, and if a man is backward they cannot show him how to advance. These are to be left out of the following scheme for efficiency, and are to be left to carry on the ordinary routine work. I cannot understand how employers expect to get increased efficiency by putting in as "shiftbosses," etc., young fellows who have not actually earned their living at the work they are put to supervise. Most people who understand handling labour know that the man immediately over the workman should be one who has gone through the mill and knows the details, like the N.C.O's. in the Army. To put college youths in as shiftbosses, etc., without their actually having had a few years at manual labour is the wrong way to get increased efficiency, and the results from mines where this has largely been done shows the folly of it. I hope that these young fellows, whom this description fits, will not take it as prejudice on my part, for it is not. Many of these young fellows have been spoilt by having responsibility put on them before they were fit to take it, and have therefore failed. They have a harder job to work up again than if they had not held such a position.

It often happens that the boss or foreman has friends, and he is anxious to secure them the best positions in the mine, regardless of their personal fitness. The result is that less work is performed, the others become disgusted with the conditions prevailing, and the cost of mining is thereby materially augmented. There are also foremen who hold the belief that only such miners as are born and bred in certain districts can be good men. These bosses are unfit for their positions owing to their narrow-mindedness. Mining is a branch of labour that can be acquired by any intelligent man, and intelligence is not confined to the aforesaid districts alone. All these things tend to decrease efficiency.

To work the proposed scheme expert teachers must be used and they should be drawn from the best workers in the class of work under review, using judgment in picking out men likely to be good teachers. These experts know the amount of work that can be done; they know the obstacles that stood in their way when simple workmen, and know how to remove them. Each of these experts should be put in charge of a few others, and teach them how to avoid the obstacles, and the best way of doing things.

Further, the expert quickly learns to develop better methods of avoiding obstacles (after a time the gangs of each expert could be increased). The expert would soon make his services valuable, and it is possible for him to make the *average* efficiency greater than the previous best. The expert after studying the different conditions of different portions of the mine, and getting the best and speediest way of doing the work, has still only gone half way; he will have to get someone to do the task, which will probably take some time, but the desired result will surely follow if the task is fairly set.

When the task is set, it must be adhered to, unless a quicker and easier way is found for doing the work.

I mentioned that the men should be paid a bonus for doing the task, also for work done over the task. In addition to paying this bonus to the workmen, the expert should receive a bonus for every man who does the task, and a further bonus if *every* man under him does the task;—this last bonus is intended to lead him to devote his attention to the poorer workers, otherwise he might give all his attention to those he was sure could get the task done.

Under this scheme, self-interest is brought into play with all, and I fancy that there would be a lot of hustling to see that there was plenty of steel and of a suitable kind, wedges, bars, etc.; that cars, shaking shoots and other conveyors, trucks, etc., were kept in order, and in time that much less supervision would be needed than at present in any mine which adopted this system.

Poor efficiency is, as a rule, not altogether the fault of the workmen but also of the management. I will try to show this, in regard to contract work also, as the same remarks apply.

Contract.—In contract machine work, I fancy most managers take the wrong way of procuring cheap work, and the strange part of it is that, in regard to *natives* they go the right way about it. We set them a task, and for anything done over that task, we pay them more per unit than for the task itself, whereas, in treating with whites, we penalise the best men by cutting the price if the wages are too high in our eyes; this is distinctly wrong, for the effect of penalising the good workman in proportion to his increased effort, is to discourage him, so that he learns to limit his output by that of the poorer worker; and how can we condemn him or be surprised?

There is one exception; in shaft sinking we generally pay a bonus for increased footage and find it cheapens the cost per foot, and we have not the sense to apply this to ordinary producing work, simply because our standing charges do not come all on one thing. It may be argued that the good man is not really penalised since he gets a good wage, but I hold that he is penalised unless he gets the same per unit as the inefficient. Another instance of what I mean is this. Some managers are paying a fixed rate per shift, no matter what is earned, plus a contract price per unit. For example, I know where a friend of mine, wishing that none of his men should go home without wages at the end of the month, gives them 15s. and 13s. 6d. per day—machine and hammer hands respectively—plus say 45s. per fathom contract price. Now, on the face of it, it looks good business, but how does it work out? If we take three men, with three machines each, breaking ground in the ratio of 1 : 2 : 3 then the price per fathom for the first is :=

$$\text{1st} = \frac{15\text{s.} + 45}{1} = 60\text{s. per fathom.}$$

$$\text{2nd} = \frac{15\text{s.} + 90}{2} = 52\text{s. 6d. per fathom.}$$

$$\text{3rd} = \frac{15\text{s.} + 135}{3} = 50\text{s. per fathom.}$$

so that the best man gets 10s. per fathom less than the worst. If machine costs are 14s. per shift the cost to the company is :—

$$\text{No. } 1 = 60\text{s.} \quad + \frac{14 \times 3}{1} = 102\text{s. per fathom.}$$

$$\text{No. } 2 = 52{\cdot}5\text{s.} + \frac{14 \times 3}{2} = 73{\cdot}5\text{s. per fathom.}$$

$$\text{No. } 3 = 50\text{s.} \quad + \frac{14 \times 3}{3} = 64{\cdot}0\text{s. per fathom.}$$

so that the man who saves the company money by his increased efficiency, is penalised for doing so. Is it any wonder that we do not get better results from our machinemen? Where is the encouragement for them to put forth their best efforts?

Take the cost per ton in the above examples:—
Stoping width 4 ft. = 12 tons per fathom.

$$\text{No. } 1 = \frac{102}{12} = 8\text{s. 6d.}$$

$$\text{No. } 2 = \frac{73{\cdot}5}{12} = 6\text{s. } 1\tfrac{1}{2}\text{d.}$$

$$\text{No. } 3 = \frac{64}{12} = 5\text{s. 4d.}$$

Now, if No. 3 can make a good wage at 50s., why should No. 1 get 10s. per fathom more than he. Even were No. 1 paid at 50s. per fathom (like No. 3) he still costs the company more than No. 3, for his cost would be $\frac{92}{12} = 7\text{s. 8d.}$, that is a higher cost than either of the others.

Instances of this kind prove that the efficient man at higher wages, is more profitable to the employer than the inefficient man at low wages.

It is no use saying that we have not enough of the No. 3 men. The remedy is to put experts on and make more; the increased efficiency will more than pay for the experts. The scheme of using teachers is applicable to contract work as well as to day's pay.

It must be remembered that efficient labour requires less machines, material, hoisting, etc., and also that it is not always the total amount of wages paid to men, but the *difference* of wages that stimulates exertion. A permanent increase of exertion will be created, and this by recognition of individual effort and not by compulsion, for it is a sure thing that the man who is driving his workers gets done down when the workers get the chance. Some bosses seem to like getting into a temper and putting their men into one also over every little thing, which is foolishness. There is a great deal of truth in the saying, "men are like tools; of little use when they lose their temper." A ranting, roaring way of giving orders, though much in favour with certain types of foremen and other bosses, never accomplishes one-half as much as the cool, collected, sure-of-himself commands of the official who, by his actions, shows that he knows his business.

Employment of men of different nationalities should be encouraged, as some miners delight in showing their working capacity over men of another race.

Keep men for long periods on the property if possible, so that they get to look at the place as home; they will then take an interest and pride in keeping the property to the front. Men thoroughly used to the mine will do more than newcomers. Bonuses in the shape of shares of the mine itself might be given, thus making them part owners.

Another great essential to efficiency on a mine is team work. By this is meant that every head of a department must co-operate with the other heads of departments; similarly with the men, each must perform his duties promptly and cheerfully. Orders are not issued for amusement; they are intended to fulfil a definite purpose, and it is only by everyone concerned doing his best that things can be kept running smoothly. It is the carrying out of the details, one man working with another, that makes the whole count.

To all, I would say, learn to take a real interest and pleasure in your work. During working hours think of nothing else. Make your work the real thing. Let pleasures and outside interests be on one side, and the results are bound to turn out pleasant.

In conclusion, I hope that members will give me a few hard hits, just to let us know that the mining members of the Society are alive: so come along and do your little share to keep up the reputation of the Society of being composed of the liveliest, hardest and straightest hitters known.

The President: I wish to thank Mr. Johnson on your behalf for bringing forward a paper on such a controversial subject. It will give us plenty of discussion. If I may say so, Mr. Johnson has in the past been perhaps mistaken for a visionary, but I think he is now beginning to come into his own. You will remember, for example, that he was the pioneer on the question of ventilation now so prominent in this country, having read a paper on it before us about the year 1903. In the present paper, one of the new things he has brought forward is the point as how trade unions originate. I fancy some of you will have something to say on that.

VENTILATION AND HEALTH CONDITIONS ON THE MINES OF THE WITWATERSRAND, WITH SPECIAL REFERENCE TO THE VENTILATION SYSTEM OF THE EAST RAND PROPRIETARY MINES.

By S. Penlerick (*Member.*)

"The three main factors directly incident to their occupation, which combine to produce disease amongst the miners on the Witwatersrand are dust, infective processes, particularly tuberculosis and in a lesser degree the fumes arising from the use (or abuse) of explosives."

This statement was made by Dr. Irvine in his evidence before the Mining Regulations Commission in May, 1908, and with that statement there must be universal agreement.

The causes of these diseases have been discussed at great length by numerous authorities, and my object here is to deal with the means to be adopted to remove these causes

Dust. —This constitutes the greatest danger which the underground worker has to combat. The sources of dust are:—(a) Rock drilling; (b) Blasting; (c) Handling of rock. It must be acknowledged that the proper use of water in conjunction with rock drilling effectually allays the dust generated during the operation. In the face of this acknowledged fact it is astounding that to-day this remedy has not, in many of our mines, been completely carried out. The proper use of water implies:—1. An adequate supply of clean water. 2. Efficient application of same.

With regard to these points the law already has power to insist on their fulfilment—Mines and Works Regulations, Act. 146 and Amendment published in Government Notice No. 1278 of 1908.

As there can be no possible excuse for non-compliance with the Regulations relating to this which non-compliance should really be considered a criminal offence, I would urge the rigorous enforcement of the law, and, if necessary, an increase of the penalty for offences.

I would here express my agreement with the opinion of the Mining Regulations Commission with reference to respirators. For continuous wear they are useless, although for occasional wear they are useful, *e.g.*, when returning to the face after the cut has been blasted.

To the same system of water supply mentioned above there must also be connected a device for spraying water on to the face after blasting to arrest and settle the dust generated at the time of the blast. For this purpose the "James Water Blast," or some similar device, is quite satisfactory.

On the East Rand Proprietary Mines we simply connect the $\frac{3}{4}$ in. water pipe to the 2 in. air pipe, fitting cocks on each behind the junction at a point where they can conveniently be operated. Care must be taken to prevent the water getting into the air pipe except when the water blast is required.

An ordinary hose pipe is sufficient for spraying water over rock which is being handled, but this spraying must be frequent throughout the operation,

For the prevention of dust in stopes we have nozzles (either directly on the water pipe or on hoses) at convenient points on the drives above, from which a stream is directed on to the face and also on to the top of stulls on which there may be an accumulation of fines from blasting.

The adoption of all these precautions will have changed the bulk of the dust into mud and settled it, but there is still a certain amount of very fine dust in suspension escaping the closest efforts at settlement. This dust must be constantly carried out of the mine and not allowed to accumulate at points where it cannot be attacked, but where periodically it can be stirred up so as to vitiate the atmosphere. This demonstrates one of the necessities for adequate ventilation.

Fig. 1 is an apparatus for taking dust samples. It consists of an aspirator through which the air is drawn by suction. The drum holds about 15 litres of water and the period of taking a sample can be regulated by the valve at the bottom of the drum. This is a convenient size for a two to three hours sample. These samples should be taken periodically to ascertain what has actually been accomplished. To Mr. C. Toombs, Experimental Chemist, E.R.P.M., belongs the credit for this neat and efficient arrangement.

Now so far I have described measures actually employed for dealing with dust, the primary source of our miners phthisis. The Government, employer and employé must all co-operate to eradicate the disease and prevent its ravages.

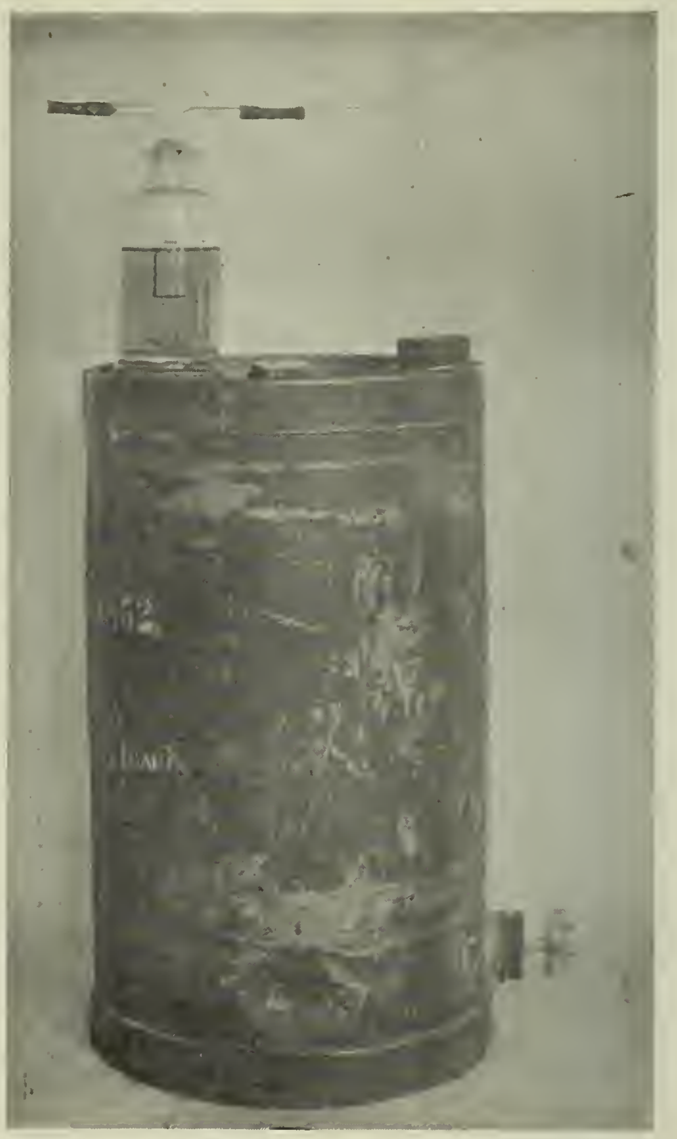


Fig. 1.—Apparatus for taking Dust Samples.

Tubercular Disease.— The incidence of this disease amongst miners is generally believed to be entirely due to the conditions under which he works. While recognising that this is to a certain extent true, it must be admitted that the conditions of private life have also contributed their share. The unfavourable condition still existing underground is impure air, and there is but one solution, namely, adequate ventilation.

It is necessary now to consider the conditions of living prevalent amongst the class of workmen who are the greatest sufferers. The actual period spent at work underground nowadays cannot in any way be considered excessive, and provided both employer and employé have fulfilled their respective obligations, the possibility of contracting this disease in the mine should be reduced to a minimum ; but even with a plentiful supply of fresh air circulating through the mine, one must recognise the existence of circumstances favourable to the development of disease in one predisposed thereto, *e.g.*, changes of temperature and humidity ; but surely common-sense pracautions on the part of the individual (such as cleanliness, suitable clothing and the proper use of the change-house, which latter must be efficiently equipped in the first place by the employer) would effectively protect the miner.

All the precautions already referred to cannot however have their full effect unless ordinary hygienic conditions of living be observed. Generally speaking, the housing accommodation is reasonably satisfactory and is unquestionably so on the newer properties. The responsibility now rests with the miner for the maintenance of fresh air constantly circulating through his room, for which purpose windows and ventilators are provided, and we think that every inducement should be made to enable him to engage in healthy outdoor recreation. Thereby his general health may undoubtedly be much improved and the spread of infectious disease minimised.

Vitiation of mine air by gases from the use (or abuse) of explosives.—The gases produced by the detonation, or more of less complete detonation, of explosives are well known. For our purpose we will merely consider that we have a quantity of vitiated air to dilute and displace, which is a question again of *adequate ventilation.*

I would here refer to the deplorable carelessness of some men, who, regardless of their own safety and that of their fellow workmen, neglect the most obvious precautions necessary in the handling of dangerous explosives, which neglect has been the cause of some of our most serious calamities. The great majority of miners fully appreciate the seriousness of this matter, and I would urge them to use their influence with their less enlightened fellow-workers to prevent, as far as possible, the recurrence of avoidable accidents which have resulted in deplorable loss of life.

The vitiation of mine air from other sources than that of gases resulting from blasting, namely, respiration, lights, decaying timber, etc, must also be considered. Excessive temperature and humidity are not directly injurious to the health of the miner, but affect his capacity for work.

Wet and dry bulb readings should be taken at least once a month from all development points and occasionally from the various stopes also. All readings must be systematically recorded.

I have so far been considering the factors affecting the health of the miner underground, namely, dust and vitiated atmosphere, and I have stated the necessary provisions for dealing with dust. I now come to the question of ventilating these mines with the object of diluting and displacing the air and maintaining a fresh air current throughout all the working points. As a standard for our fresh air we may assume the adoption of the standard recommended by the Mining Regulations Commission, which will, in my opinion, be quite satisfactory as far as the Rand is concerned :—

(1) " In every metalliferous mine sufficient air shall be provided and such other arrangements made that in any sample of air taken under normal working conditions from any part of the mine not less than one hour after blasting."

(*a*) " the proportion of carbon dioxide shall not, as regards any mine within the Witwatersrand area, exceed twenty volumes per ten thousand of air .... (0.2%)

(*b*) " the proportion of carbon monoxide shall not exceed one volume per ten thousand of air.................. ..(0.01%)

(*c*) " no practically determinable amount of the oxides of nitrogen (NO and NO_2) shall be present."

Our task is : (1) The provision of a sufficient quantity of fresh air. (2) The efficient distribution of same.

I consider that the most convenient way of illustrating the principles to be followed will be to describe the system already installed at the East Rand Proprietary Mines Of course the details will not always be applicable to other mines, but the general principle remains the same.

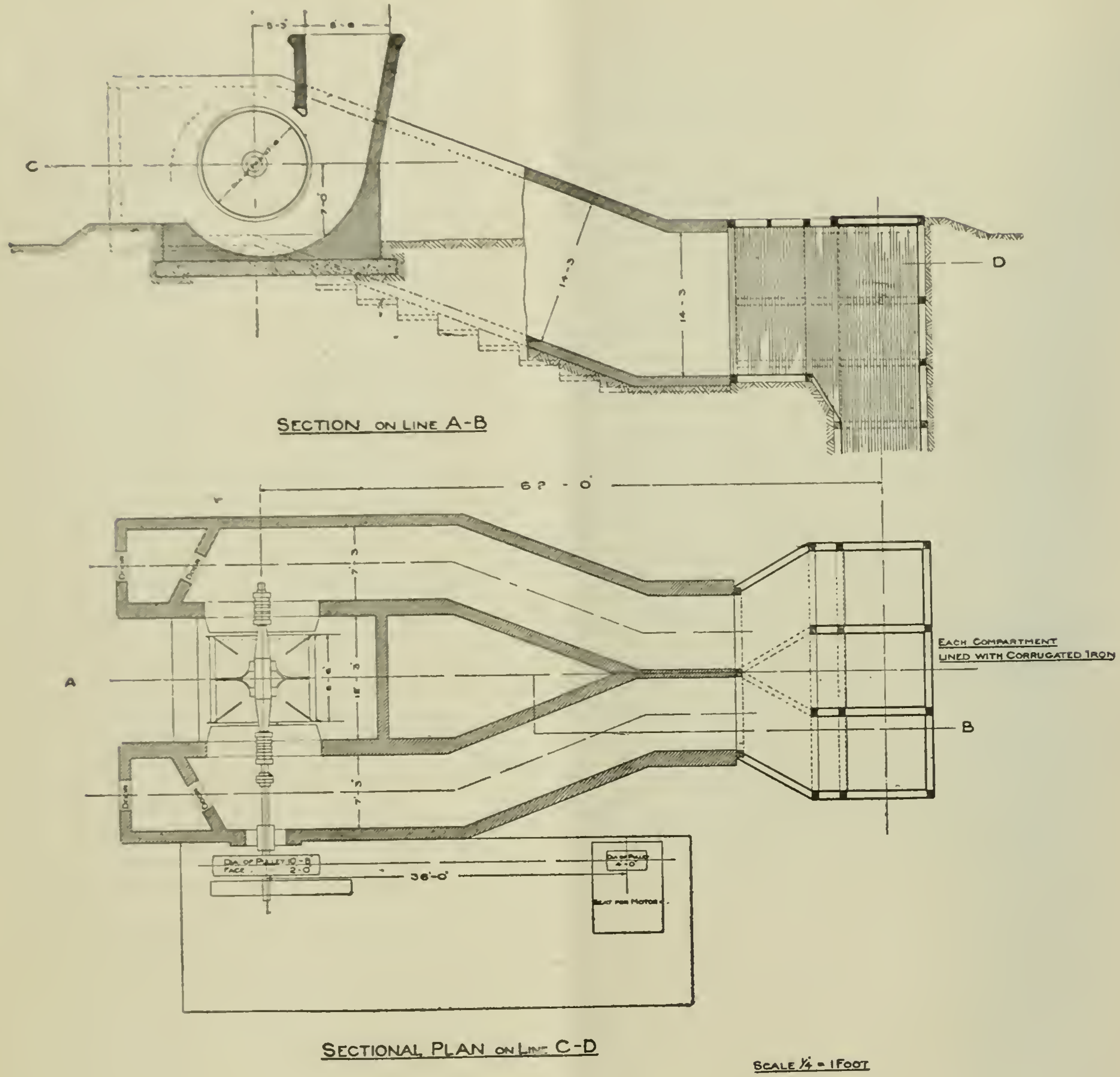
The position of affairs in 1906, when the question of installing artificial ventilation was taken up, was that the Farrar, Comet and Cason Shafts were the natural downcasts of the property, while the No. 1, Driefontein, and the Main Shaft, Angelo, were upcasts. The old St. Angelo Shaft, West of the Main Shaft of the Angelo, and the present Cason upcast were disused prospecting shafts. Thorough systematic distribution did not exist.

We decided to make all the main shafts downcast and use the two prospecting shafts which were connected to the old stopes as the upcasts, stripping and equipping them accordingly.

The arrangement for the fans at the top of these upcast shafts are shown in Figs. 2 to 7.

Description of Fans.

Shaft.	Fan.	Dia.		Width.		Output.	Water gauge.
		ft.	in.	ft.	in.	Cub. ft per minute.	ins
Angelo Upcast	"Sirocco" Double Inlet.	10	6	8	6½	300,000	3
Cason Upcast	"Barclay's" Double Inlet Drum Pattern.	16	0	8	0	350,000	4



SECTION ON LINE A-B

SECTIONAL PLAN ON LINE C-D

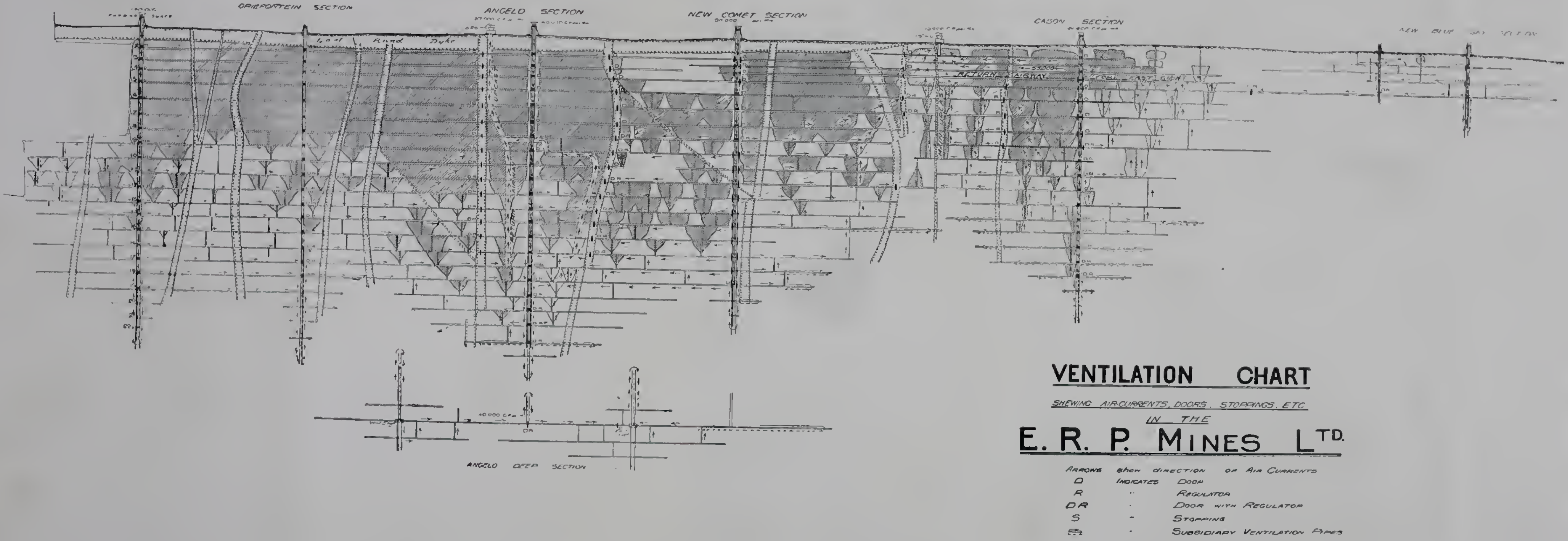
FIG. 2.—General Arrangement at Top of Upcast, Angelo Section, showing Upcast Shaft, Drifts and Fan Setting.

I shall now describe the system of distribution of the air, starting from the collar of the downcast shafts. The Farrar is a five-compartment shaft with an effective ventilating area of 145 sq. ft. Following the course of the air current in the ventilation chart, the crosscuts from the shaft to the levels are securely bratticed off. In the upper levels where no work is proceeding the doors in the brattice are securely fastened, to prevent short-circuiting of the fresh air through the disused portion of the mine. From the 6th to the 17th levels, inclusive, self-closing doors are fitted in the brattice, and in addition regulators for the admission of the required quantity of fresh air for each level. From the 18th to the 22nd (bottom) level the shaft is divided into a downcast and an upcast by means of a close brattice between the hoisting ways and the ladder way. The object of this is to provide a return way for the air from the bottom of the shaft up to that level which is connected through the main upcast. Figs. 8 and 9 show this brattice.

It will now be clear how the air is distributed from the downcast to the various levels as required and how the currents proceed eastward towards the upcast. On the west side of the main upcast there is a transverse dyke running throughout the mine, of which advantage has been taken to form a natural brattice. The levels running through this dyke are permanently closed where

Supplement to The Journal of The Chemical, Metallurgical and Mining Society of South Africa, August, 1910.

Illustrating "THE VENTILATION AND HEALTH CONDITIONS OF A WITWATERSRAND MINE, WITH SPECIAL REFERENCE TO THE VENTILATION SYSTEM OF THE EAST RAND PROPRIETARY MINES," by Mr. S. PENLERICK (Member).



VENTILATION CHART

SHEWING AIR CURRENTS, DOORS, STOPPINGS, ETC

IN THE

E. R. P. MINES LTD.

Arrows shew direction of Air Currents
D Indicates Door
R .. Regulator
DR - Door with Regulator
S - Stopping
- Subsidiary Ventilation Pipes

Fig. 2.—G

I shall now
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levels where
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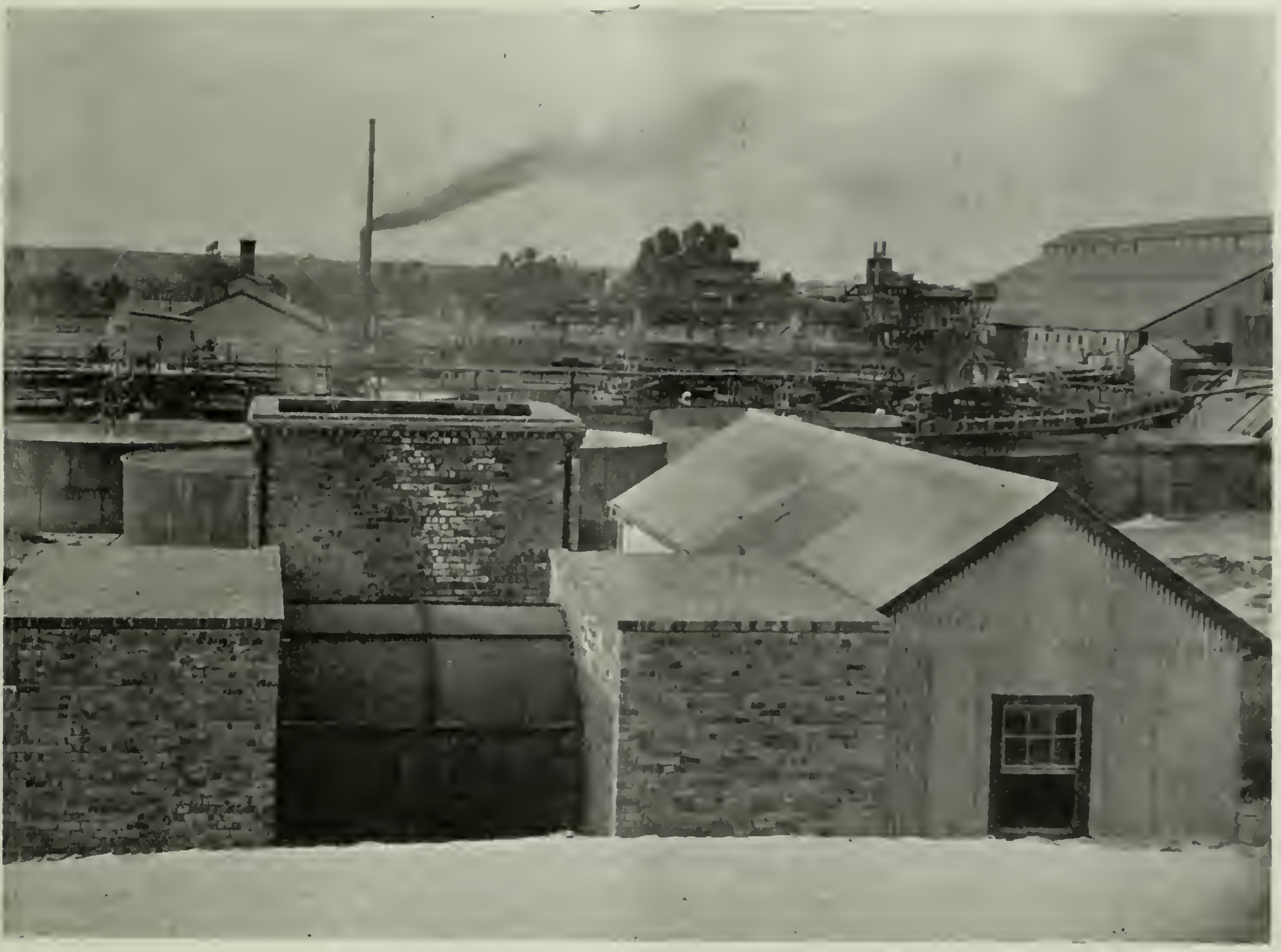


FIG. 3.—Fan and Motor House, Angelo Section.

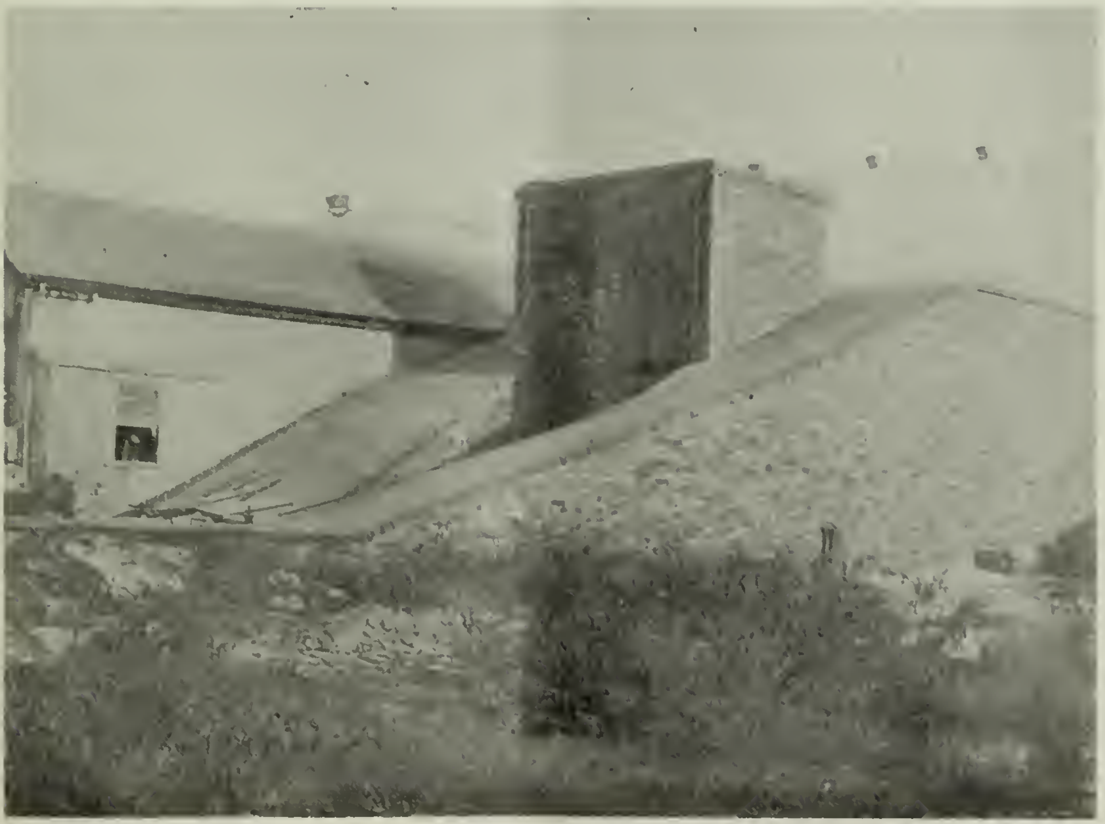
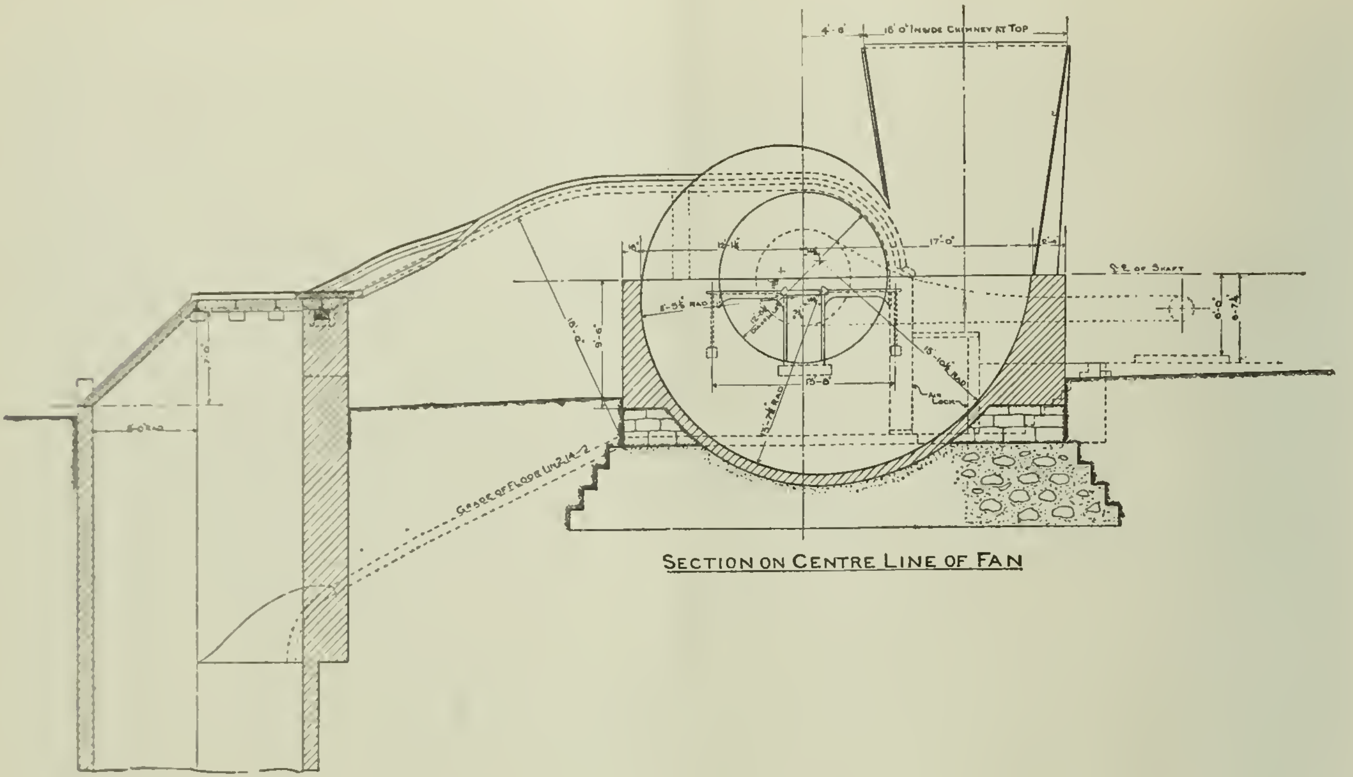
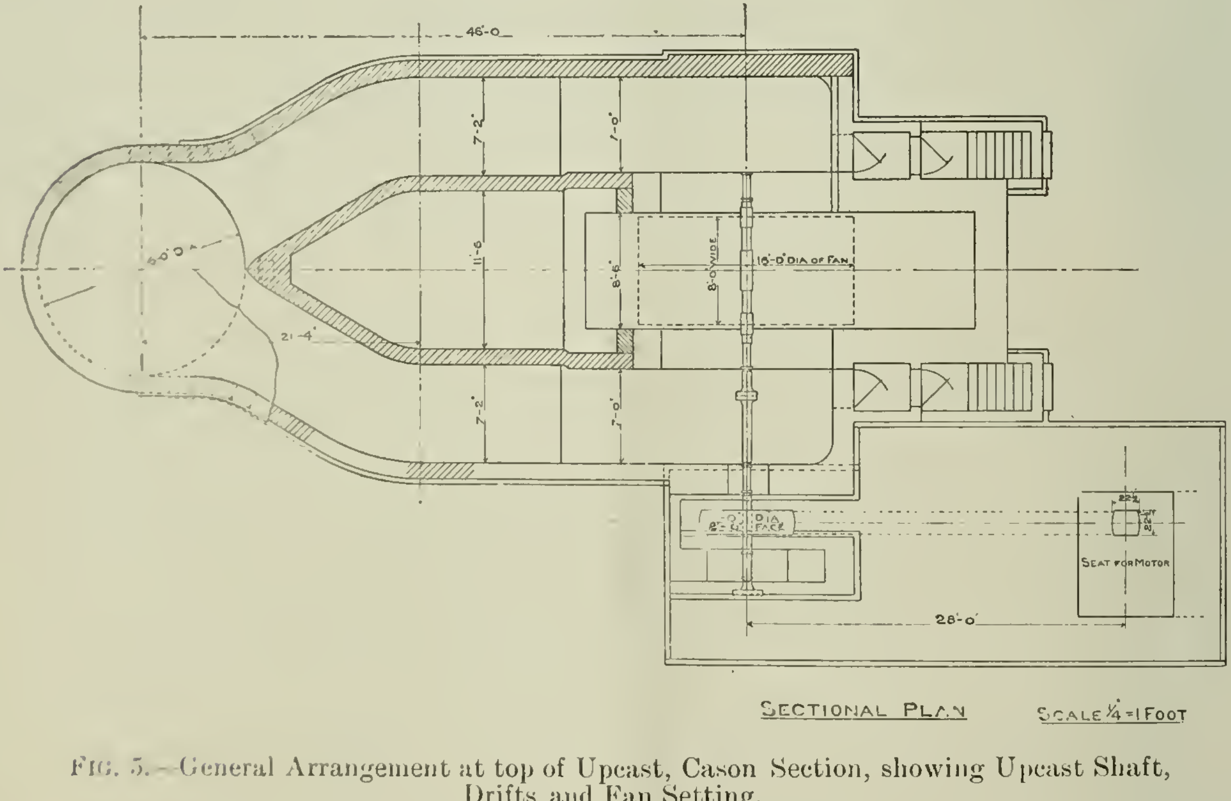


FIG. 4.—Fan Drifts and Delivery. Angelo Section



SECTION ON CENTRE LINE OF FAN



SECTIONAL PLAN SCALE ¼ = 1 FOOT

FIG. 5.—General Arrangement at top of Upcast, Cason Section, showing Upcast Shaft, Drifts and Fan Setting.

there is no tramming, and are closed with double swinging doors where tramming has to be carried on. Fig. 10 shows one of these swinging doors. Fig. 11 shows the details.

The course of the air-currents from the collar of the other downcast shafts may be similarly traced.

Fig. 12 shows the method of bratticing the station crosscuts and, at the same time, the door and regulator will be seen.

Auxiliary Ventilation—The 21st level of the Angelo Shaft is the transfer level for this section. The arrangement is shown in Fig. 13. It will be seen that we have three 9 in. and two 15 in. diameter pipes delivering into a 21 in. diameter pipe, with a 21 in. Sirocco fan installed below the 19th level to exhaust the air from this section, the intake air being derived from the main shaft. The delivery from this fan is into the main upcast.

Before installing this small fan here we used a ring jet, Fig. 14, through which compressed air

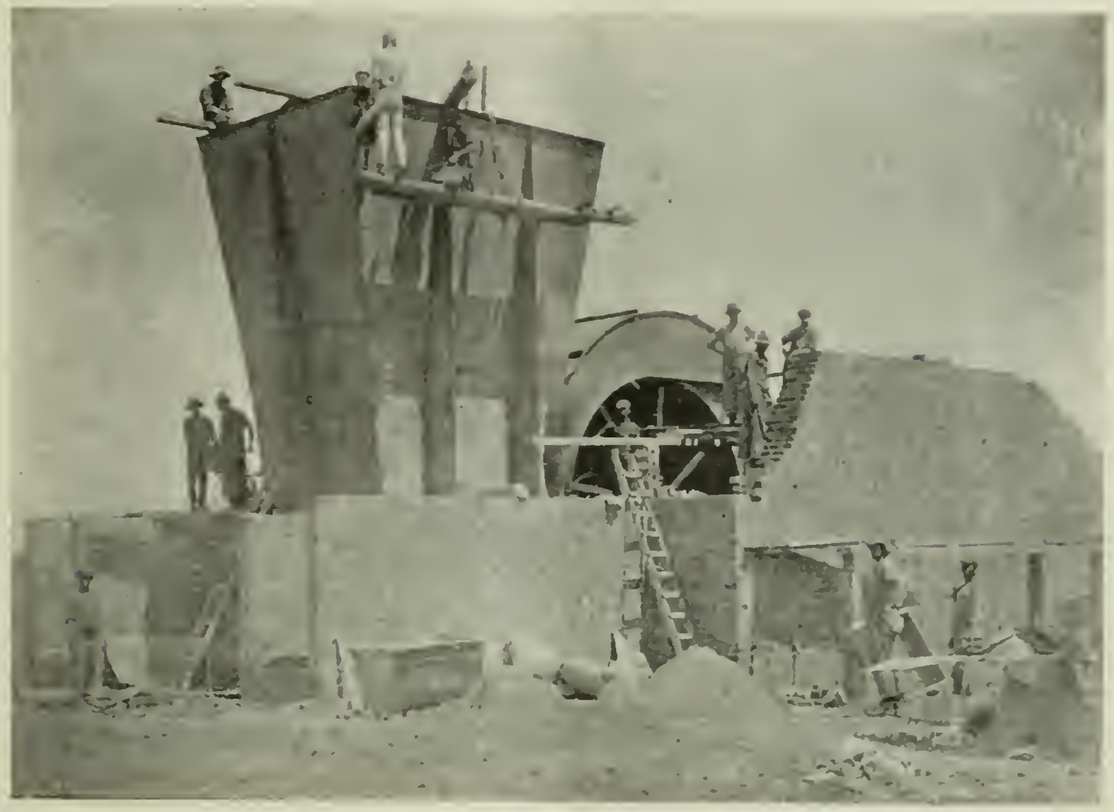


FIG. 6.—Fan in course of erection. Cason Section.

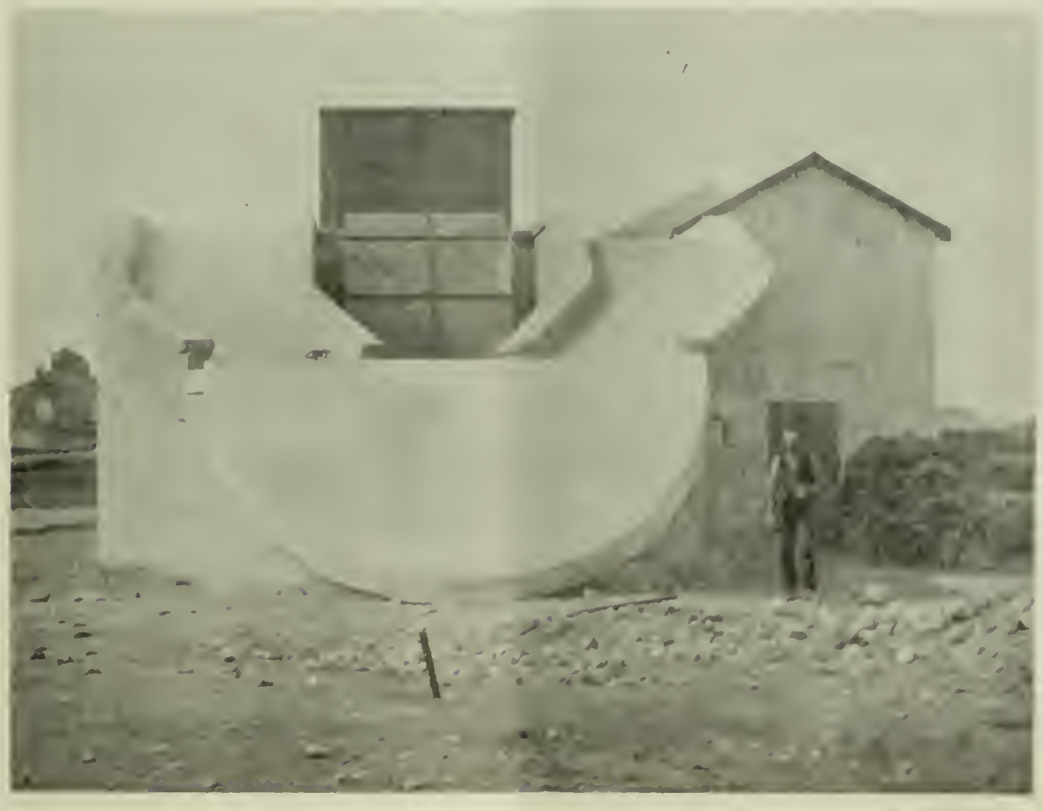
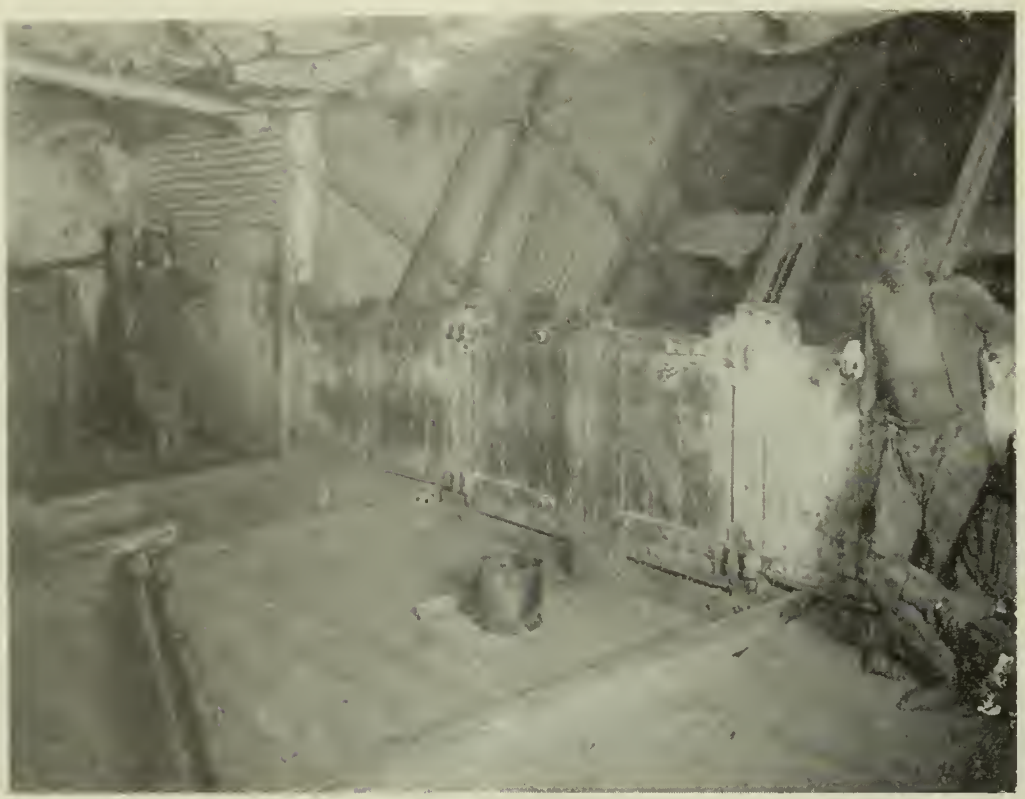


FIG. 7.—View of top of Upcast and Fan Drifts and Delivery. Cason Section.



—Brattice between Ladder-way and Hoisting Compartments, forming Upcast and Downcast in Shafts. Driefontein Section.

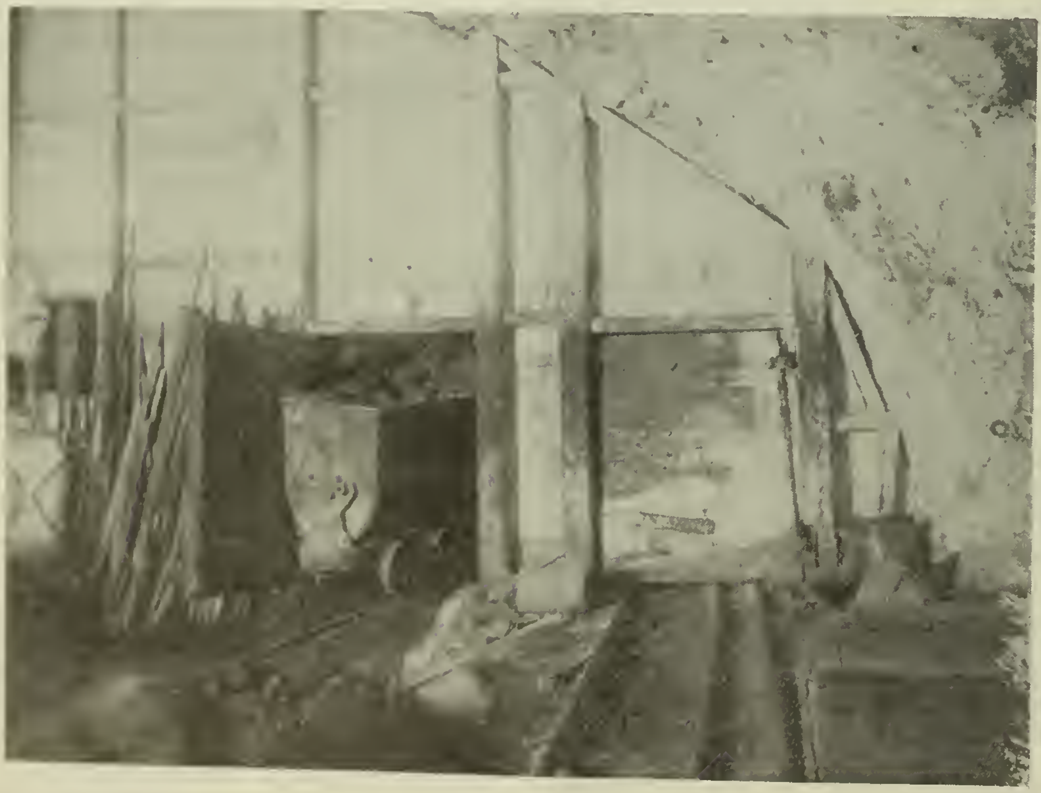


Fig. 9.—Brattice between Ladder-way and Hoisting Compartments, with Swing Doors for Tramming, Angelo Section.

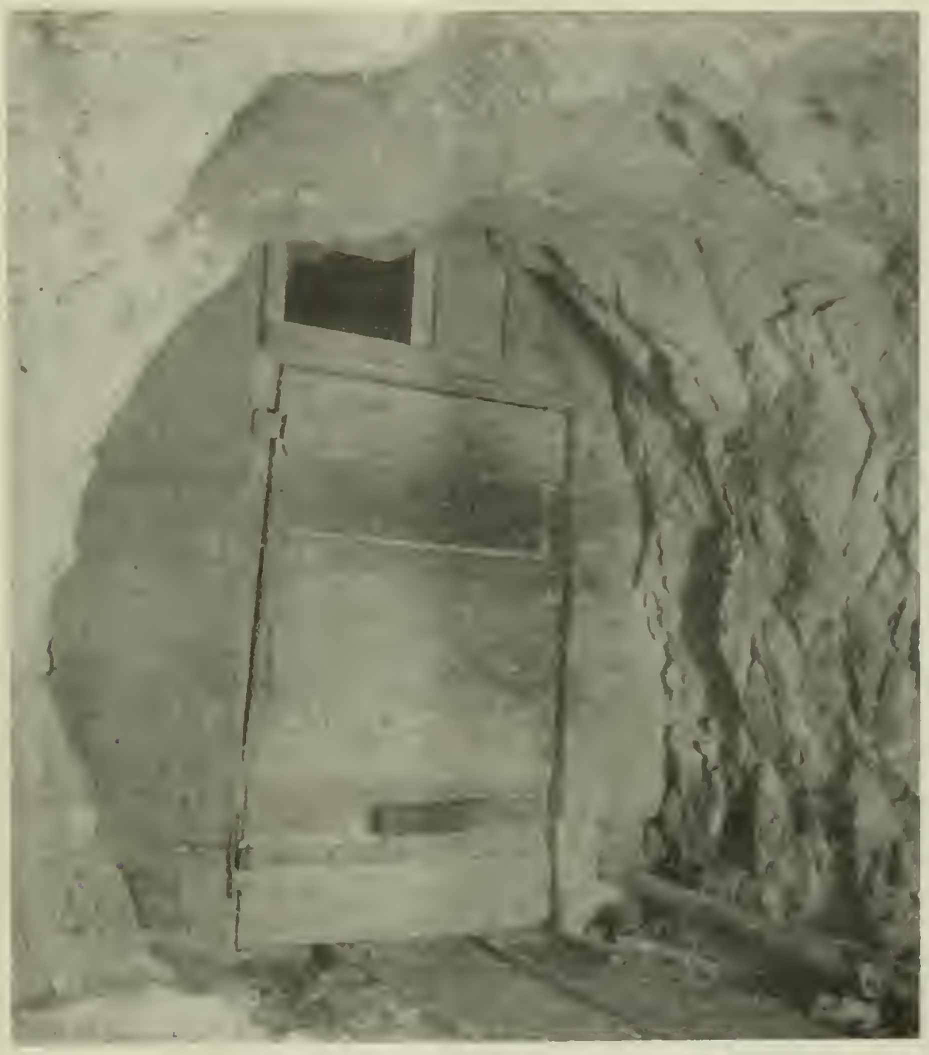


FIG. 10.—Swinging Door and Regulator.

was directed up the pipe inducing a draught and exhausting the air from below. This form of jet is very convenient for temporary use and for requirements below 1,000 ft. per min. at a low water-gauge.

With regard to the ventilation of development drives, where a drive face is being carried far beyond the last connection with an adjacent level, it is necessary to provide for an intake and an outlet. This object is best obtained by using a galvanised iron pipe for the outlet. We prefer to use the pipe for the outlet, as the drive in which there is traffic is kept fresh throughout. An example of this is shown in the Angelo Deep section 27th east drive.

I may here mention that the Angelo Deep section, which is only in the development stage, will depend upon natural ventilation until connection is made with the Angelo section. At present the west shaft is downcast and the east shaft is upcast, as will be seen from the diagram.

In the case of long winzes and rises, these should be bratticed to provide a return. (This brattice at the same time provides protection to the miners in passing between the level and the face, a pent-house covering the travelling way.)

We will now consider what the effect of this scheme has been. Samples were taken from the bottom of three shafts, viz., the Farrar, No. 1 shaft (Driefontein), and the main shaft (Angelo). In all these shafts sinking was proceeding. The average results for these points were as follows :—

$$CO_2 = 0{\cdot}552\%$$
$$CO = 0{\cdot}011\%.$$

After the ventilation scheme was in operation samples were again taken from these points, and the following average results obtained :—

$$CO_2 = 0{\cdot}149\%$$
$$CO = 0{\cdot}006\%.$$

It will be noted that these figures do not represent the average for the mine. The average figures for the mine, however, were :—

1st.—before the adoption of the ventilation scheme :—

$$CO_2 = 0{\cdot}193\%$$
$$CO = 0{\cdot}012\%.$$

2nd. after the adoption of the ventilation scheme :—

$$CO_2 = 0{\cdot}127\%$$
$$CO = 0{\cdot}005\%.$$

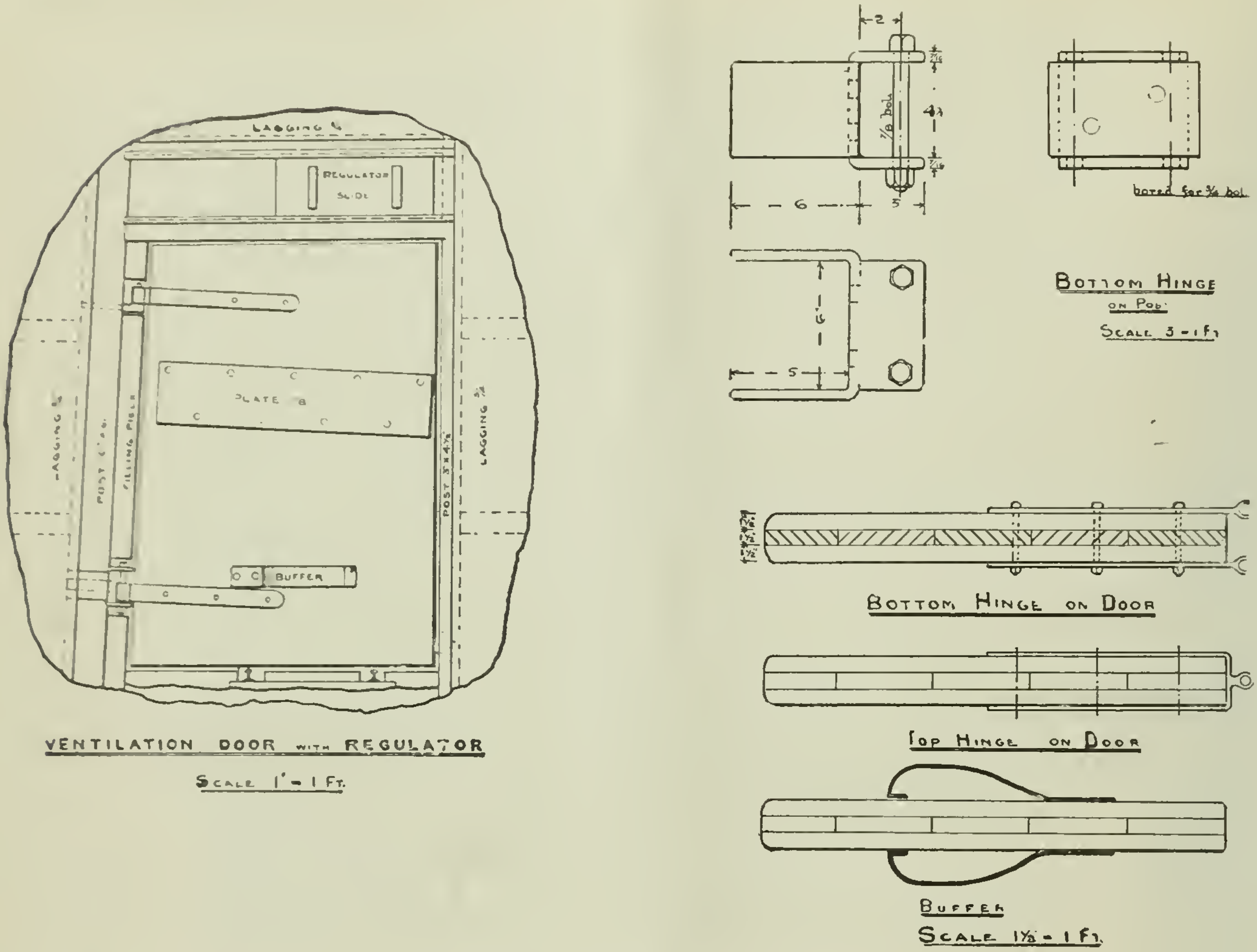


FIG. 11 —Details of Swinging Door and Regulator

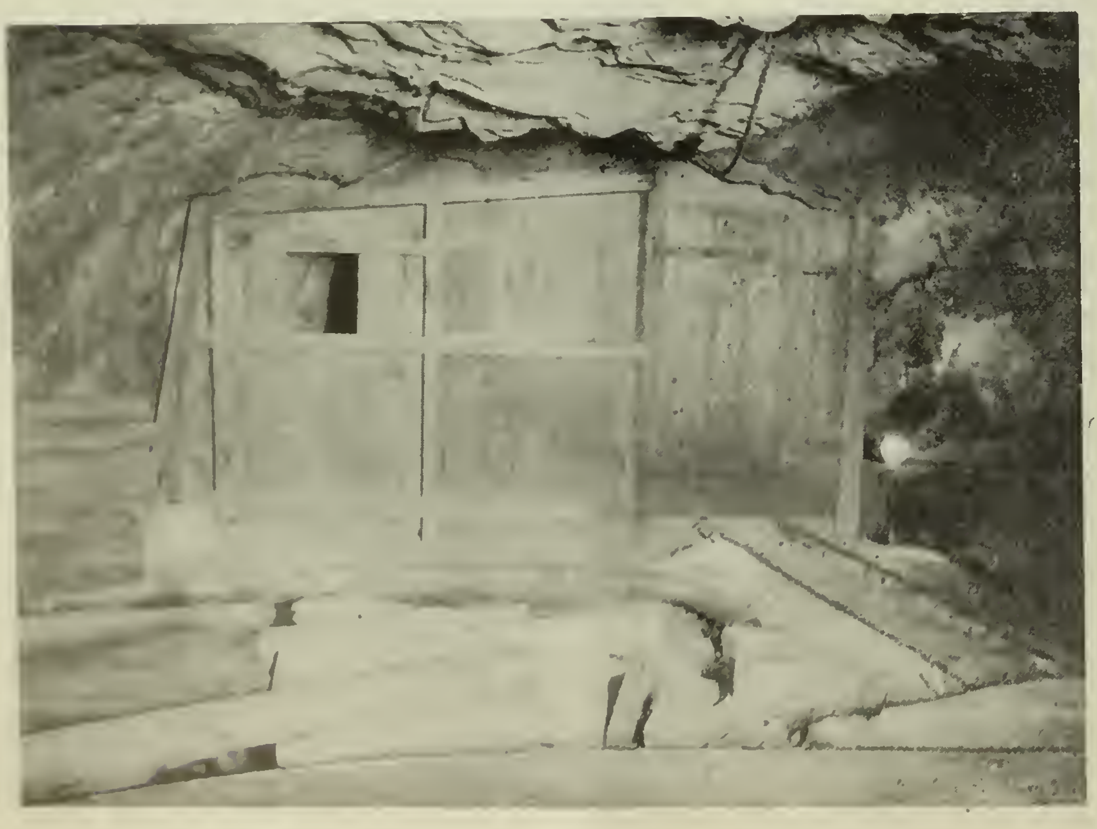


FIG. 12.—Shaft Cross-cut, Brattice, Door and Regulator.

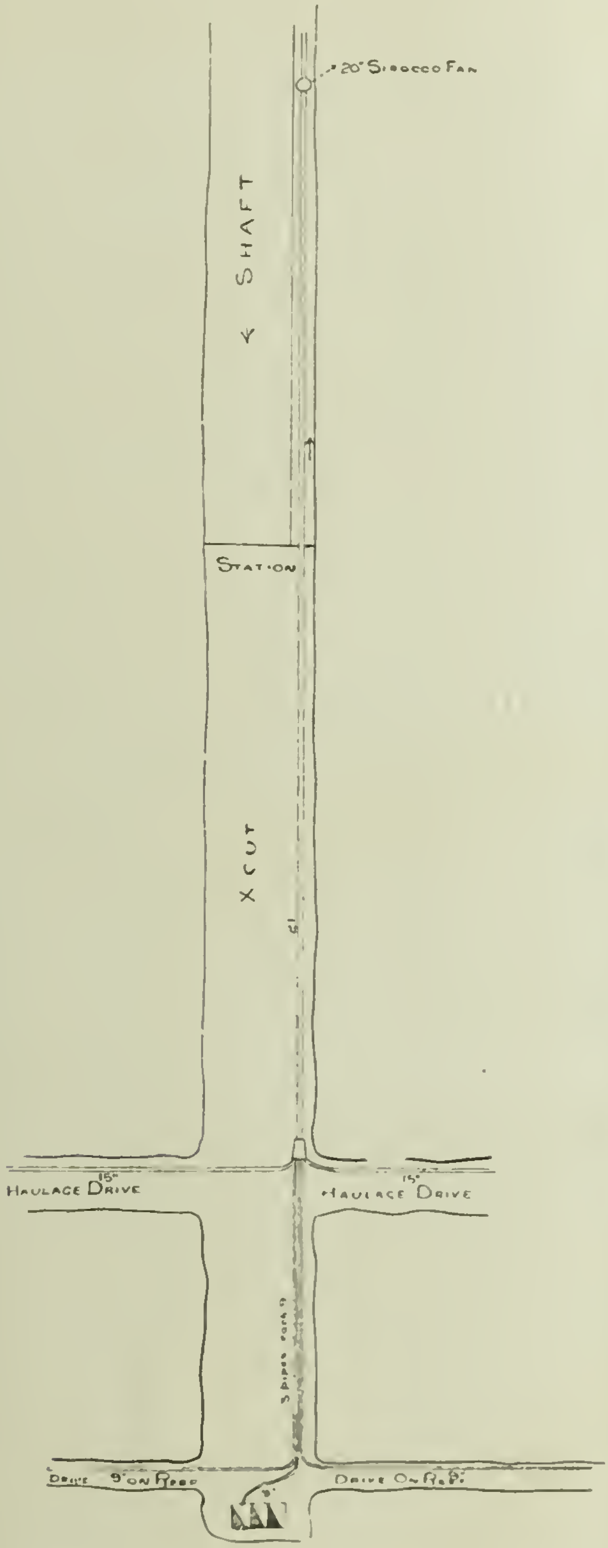


FIG. 13.—Auxiliary Ventilation, 21st Level. Angelo Section.

The samples were carefully taken and the analyses were done by Dr. Weiskopf. Recent samples taken from the air drift at the Angelo upcast directly after blasting when the vitiation of the air should be at its worst, gave the following results; $CO_2 = 0·182$ % and $CO = 0·001$ %. The temperatures have been reduced by from 2° to 3°.

The capital expenditure on the installation for the whole property, including fans and motors, erection of same, equipment of up-cast shafts,

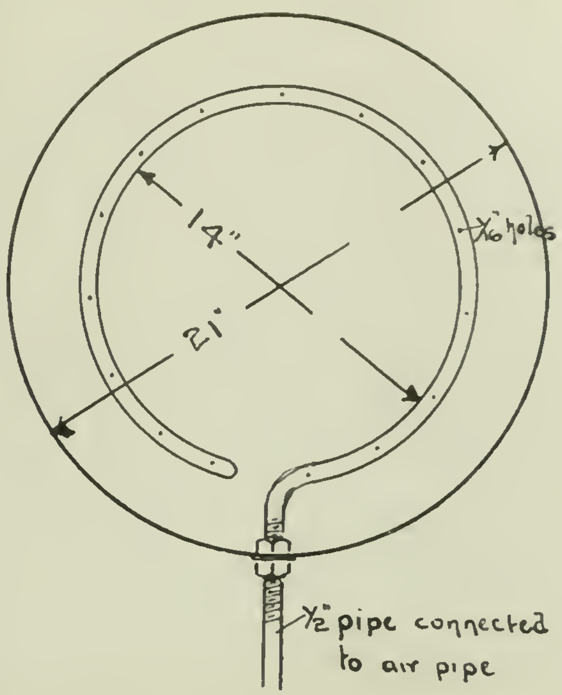


FIG. 14.—Ring Jet for Auxiliary Ventilation.

bratticing, doors, pipes, etc., was about £25,000.

The running cost, including maintenance of plant, and of the equipment in the mine, is under 1d. per ton milled.

We feel sure that those other companies which have adopted artificial ventilation, namely the Cinderella Deep and Village Deep, have had similarly satisfactory results.

We are perfectly well aware that the scheme applied to the East Rand Proprietary Mines is not in detail applicable to every mine on the Rand but the general principles will apply.

The cost is insignificant compared to the valuable results obtained, and under the circumstances some system of artificial ventilation should be adopted in practically every mine on the Rand.

Another important factor in securing the most satisfactory conditions of the mine air is the practice of confining all drilling and blasting operations to the day shift. We are aware that many companies to-day are not in a position, for financial and other reasons, to adopt this practice, but where the adoption of it is possible, we cannot too strongly recommend it.

Mr. H. Stuart Martin (*Member*): I have been asked to propose a vote of thanks to the author for his valuable paper on ventilation. I can at least say that I am delighted to propose this resolution, and I am sure you will all heartily join with me in thanking the author for the very able way in which he has gone into the matter. It is particularly an interesting paper, and more particularly an important one, and one that concerns all of us and all those working underground. It is a subject which from the very first day I entered into the mining world I have had to pay considerable attention to. In my case, we were forced to go in for mechanical ventilation not only to keep the dust down but also on account of the very dangerous and inflammable gases. In our case we have the dangerous gases from explosives and we have the dangerous dust which is the cause of miners' phthisis. Coal dust does not cause phthisis, but under certain conditions does harm to the body. However, it is highly dangerous in the dry steam coal collieries, owing to its inflammable nature, and is kept down by systematic watering. Good ventilation does not prevent dust, but rather excites it and carries it to other parts of the mine. Still the gases must be cleared. The gases can only be cleared by good ventilation, in conjunction with the proper use of water, and I feel confident that the dangers from deleterious gases and dust can be in a great measure mastered if not almost entirely done away with. I am particularly pleased, and I am sure all of you are, that the author has brought this matter up, and I hope at another meeting we shall have a large number of members here and that we shall enter into a detailed discussion. To-night I am not prepared to say anything on the paper, but I hope that at another meeting, when discussion will ensue, I shall have something more to say. I have very great pleasure in proposing a vote of thanks to the author and I am sure all of you will heartily join me.

Dr. L. G. Irvine (*Member*): I feel that it is a compliment to myself and to Dr. Macaulay to be asked to second this vote of thanks. I do not know what his feeling is, but probably with me he feels rather like Simeon of old, when he said, "Now lettest thou thy servant depart in peace." The author has given us to-night an excellent object lesson of what ought to be done from end to end of the Reef, and I think that he and the group he works for are to be greatly congratulated for the thorough way in which they have taken up the whole problem of systematic ventilation, and demonstrated its practicability, its efficiency, and not least of all its *cheapness*. I think that they have laid this bogey of ventilation once for all. I am sure you will all agree with me that the author's paper is one of the greatest practical use. There is not a superfluous sentence in it, and not a sentence without its practical point. One has not time to go into the paper in detail, but its whole tone is exactly as it should be.

There are one or two points in it, however, to which one may refer. The author appeals for co-operation between the Government, the management of the mines, and the mine employes, in carrying out the regulations regarding dust-laying and other matters, and in effecting the changes which are necessary underground. This is absolutely right and absolutely necessary. We have had far too much in the past of mutual bickering—the men saying they have no appliances, and the employers that the men will not use them when they have them. We want a new spirit underground. Employers and employed must heartily co-operate to find out what are the best devices, and to use them when they have them. And we want better discipline. I do not think the regulations will ever be properly enforced till mine officials as well as the miners are made liable to penalty for breaches of the regulations. And it will be of no use for the management of one mine to carry out a strict enforcement of discipline unless all of the mines do so.

The author makes a statement about working hours. He says that the actual period spent at work now-a-days cannot in any way be considered excessive. I take occasion to doubt if that is always so. I think there is sometimes a good deal of delay in haulage, especially in the haulage of natives, and this wants remedying. I do not agree that the number of hours worked is in all cases what one would like to see. Personally, I have a strong sympathy with the 8 hours proposal.

With regard to the living conditions of miners the author says that the incidence of phthisis has been generally believed to be entirely due to the conditions under which the miners work. I do not think that this has ever been quite the expert opinion. We have always agreed that living conditions above ground count for a good deal, and for these the personal responsibility of the miner is of course considerable. But living conditions above ground can also be improved. We want to see "back to back" quarters abolished —we want to see for underground workers one man in one room. I know of shafts also where there are no change houses for white men.

The Mining Regulations' Commission has made certain suggestions regarding tuberculosis. They suggest that men affected by tuberculosis should be legally prevented from going underground. With regard to natives I quite agree With regard to white men I do not. I take up this position. I think the Government, before granting a blasting certificate for the first time to a miner who has applied for it, has a right to have him examined by a Government Medical Officer. If he is affected with tuberculosis I do not think he should be granted a blasting certificate. But once a man has got a blasting certificate it would savour I think of medical tyranny to have him regularly inspected. I think such a system would lead to evasion and concealment, and would tend to defeat its own ends. What we want to do is first to make underground conditions such that the risk of contracting tuberculosis is reduced to a minimum. And second we want to make an avenue of escape from mining work possible for the man who has contracted tuberculosis, by providing sanatorium treatment for early cases, to give a man time to look round for alternative employment, to which he should be helped in every way possible. And for those permanently disabled we want to have the sanatorium, and some sort of pension fund. I believe in a sanatorium, but I want to see it empty all the time. The true "cure" of miners' phthisis is prevention. I believe that these methods would secure the end aimed at, namely, to get the tuberculous workman out of the mine, better than a drastic compulsory measure such as the Commissioners suggest.

The author has also called attention to the important question of gassing accidents. It is high time that we got rid of the idea that gassing accidents are "acts of God." They are almost invariably acts of human negligence, either on the part of the employers in neglecting to provide proper means of local ventilation of close places, or on the part of the men in not using them.

I quite agree with the author that the standard of ventilation (·2% CO_2) laid down by the Commission is satisfactory. The value of such a standard is its flexibility. It is applicable to all parts of a mine, and, if the air supply of the "close" places of a mine is below ·2% CO_2, the general average of the mine air will be much better than that. The author has given us a splendid object lesson as to what has been done in the way of systematic ventilation on the East Rand Proprietary Mines. Detailed application from mine to mine will vary, but the principles may be universally applied. He very rightly makes a strong point about local ventilation. To my mind this is very important, it is one of the most important elements in the problem, both from the point of view of the prevention of phthisis, and from the hardly less important standpoint of the prevention of gassing accidents. It does not matter how much air is going into a mine, if it does not get into the ends of the drives, or the bottom of the winzes, where gases collect. Local ventilation of winzes, rises and drives should become part of the routine work of mining. We want to see mine officials systematically applying the simple devices necessary as part of their ordinary work without waiting for the Government Inspectors to come along and tell them to do it. Every mine may not require general mechanical ventilation, although where doubt exists it should be installed, but everyone of the mines requires means of local ventilation. All winzes, drives and rises must be ventilated sufficiently and have dust-laying devices, to remove the gases and lay the dust. Rises are a specially difficult feature—the best device for dealing with them is to make it the policy to do without them.

One of the most satisfactory points in the paper is that it demonstrates the comparatively insignificant cost of the whole affair. Some mines in this respect of course will have an advantage over others. We have had too much on the Rand of a somewhat slavish adherence to conservative methods of mining. People have said "other metalliferous mines have done without mechanical ventilation so we can do without it too." But the mines of the Rand have to be judged on their merits, and each mine on its merits. If Dolcoath with natural ventilation supplies 300 cub. ft. of air per man per minute, that has no bearing on the fact that the air supply on somes mines on the Rand may sink at times to 15 cub. ft., or even hang at times in a stagnating balance with no air going into the mine at all.

But of one thing I am convinced, and that is that the leaders of the industry are now determined to do their utmost, on the evidence laid before them in the Report of the Mining Regulations Commission, to remove faulty conditions underground. They realise the importance of the problem, not only from the humanitarian point of view, but from the standpoint that it is false economics to permit a state of affairs to continue, which leads to inefficient work, inefficient work men, and an unsatisfactory labour supply. I know you will all join with Mr. Stuart Martin and myself in proposing a cordial vote of thanks to the author for his paper to-night.

Mr. W. S. V. Price (*Member*): As one of the very few practical, or rather working miners, who are members of the Society, I have very great pleasure in supporting this vote of thanks. This question of "ventilation" is so very largely bound up with Mr. Johnson's paper on efficiency, that I think we really cannot do justice to the discussion of them this evening as the time is getting on. To show how these matters are bound up. I was working on a mine not long ago where there was a rise about 500 ft. or 600 ft. from the station, the drive continuing for another 600 ft. or 700 ft. further in. The air naturally was very foul in the drive, and a brattice had been put up in the rise. As there were a lot of old ventilating pipes in the mine, I suggested to the mine captain that we should use some of these, with a bell mouth in the connecting drive, and the other end up the rise, as it would freshen up the rise, and make the outer portion much fresher, in fact it would be almost as good as if the drive was just starting from the position of the rise. "Why, man," he said, "what for ? there is brattticing up there!"

I hope at a later date to make up a few further remarks on this paper. I again wish to support the vote of thanks to the author for his clear and lucid paper, and especially for this admirable scheme of ventilation. I should like to ask the author if the mine has not been repaid for the outlay, by increased efficiency on the part of the men and boys?

Mr. S. Penlerick (*Member*): I believe Mr. Price knows our mine as well as I do myself. Not only from a health point of view, but from an economical point of view, the £25,000 spent has been repaid us many times over.

Dr. Aymard (*Visitor*): It is about seven years since I was here before this Society. I would like to add my testimony to this most instructive paper on ventilation. I know a little about ventilation, but I have learned a great deal more. I think I made a remark the other day that ventilation might be overdone but certainly not in the direction the author indicates. As medical men, whenever we start a new thing we almost invariably overdo it, and the result is that we go a long way round and gradually come back to it again. I fully agree with the interesting remarks Dr. Irvine has made. He and Dr Macaulay have been the great pioneers here. I have come along as a practical worker, following their teachings. I have made great efforts to destroy the dust at the point of origin, and I think I may say that I have succeeded. It has been pointed out by the Mining Regulations Commission that men with tuberculosis should not be allowed underground. I do not agree with Dr. Porter in a great many things, but I think he was justified, from a public health point of view, in saying that such men should not be allowed underground. But as a medical man myself I think it could not be carried out, and it is not necessary. It would be a great hardship on the men, and it would be unnecessary. It is not as if they go down and sit all together in one drive. Immediately they go down the mine they are distributed. There is one other point to which I am opposed. The further you go down into the earth, the drier the atmosphere must be. We have no right to ask a man to work in a steaming or even saturated atmosphere. The forcing of men to work under the spray or atomiser cannot be defended medically.

Mr. R. Gascoyne (*Member*): The system described by the author is an up-to-date one, and quite on a par with the most advanced system you will find in Europe to-day, and I certainly think the other mines on the Rand should take it as a pattern. There is one remark where he deals with the water supply in which the statement occurs: "As there can be no possible excuse for non-compliance with the regulation, non-compliance should really be a criminal offence. I would urge the rigorous enforcement of the law, and, if necessary, an increase of the penalty for offences." Now, if it is necessary to make the law observed in regard to fresh water surely it is equally necessary to make it observed in regard to fresh air. Fresh air is as equally necessary to the miner as fresh water. It used to be enacted in the mining regulations that 70 cub. ft. per person is the amount of air required, and the Commission on Mining Regulations now says that this cannot be obtained in the mines, I say if you cannot obtain that amount it is no use making new regulations because they will require the adoption of fan ventilation just the same as the 70 cub. ft. of air per minute per person underground.

The author points out that efficient distribution is more important than a large quantity of air, a remark with which I agree. Another important matter is that the author takes his air to the lowest point. That is a very important point in a deep mine. It will be found that in a deep hot mine it seems almost impossible to get the air down to the lowest point as it eludes your grasp and escapes by a short cut to the return wherever possible.

With regard to ventilating a drive by pipes, that a pipe should be made a return rather than an intake, will depend upon circumstances. If there is a long length of drive and it is desired to keep that drive clean and fresh, that may be the best plan; but if there is a short pipe, especially in the absence of compressed air, it is best to make it an intake so that the air can be thrown on the working face. If the pipe is a return it will be found that the air escapes to it without touching the face of the drive. On the East Rand the ventilation has reduced the temperature by 2° or 3°, but as a rule if mechanical ventilation be introduced in a deep mine, the temperature of the atmosphere should be reduced from 15° to 20° as compared with the rock temperature. I must congratulate the author on the small cost, because it compares very favourably with European collieries where it also runs at from ½d. to 1d. per ton. Another thing I must congratulate him upon is that the upcast shaft is used entirely for the return air to escape to the surface. It is a very excellent plan, and ought to be carried out everywhere, because if the upcast is used also as a winding shaft it will be found that the return air damages the rope and timbers, and in a short time will either ruin the shaft or it will give a great deal of trouble, not to mention the risks caused by the damage to the ropes. Again, the upcast shaft should always be lined with steel, because steel will stand the return air much better than timber. With regard to the hope expressed that other mines that have started mechanical ventilation will meet with the same results, I have no doubt that if the same system as that the author has adopted is carried out on other mines, they will meet with equally satisfactory results.

Mr. A. T. Judge (*Member*): I am very grateful to the author for the paper he has read. I do not want to criticise his paper for I feel that I do not know enough about ventilation. I wish to answer the remarks made by Dr. Aymard as to killing the dust at its source. I think Dr. Aymard is altogether mistaken about the principal source of dust. The amount of dust from rock drills is trifling compared with that from other sources. I maintain that it does not cause one tenth of the phthisis existing. The main source of dust is due to the amount of explosive used. We can go into drives 1,000 ft. in from ventilation 12 hours after blasting, and find the air hazy with dust. I think what they have done on the East Rand Proprietary Mines is the only cure for miners' phthisis. At the present moment miners may use appliances that palliate the evil, but without ventilation there will be no cure. We want full colliery ventilation.

Dr. D. Macaulay (*Member*): I wish to support my colleague, Dr. Irvine, in seconding the vote of thanks, and for this reason, that I consider the second concluding paragraph in the author's paper is the Magna Charta of the miner in this country. He says the principles of ventilation may not be applicable in every detail to other mines, but the general principles will apply to every mine on the Witwatersrand, and he also says the cost is insignificant. This matter has been under the consideration of one of the most important mining groups on these fields since 1906, who have authorised one of their people to make this statement to night. I wish to say this, that inasmuch as the Society has taken an honourable position in bringing this matter into prominence, it is to be hoped that you will see to it that it does not fall again into insignificance, and that you will work at it without cessation until the author's principles are carried out from one end of the reef to the other. It is cheering for me to come here to-night and find this Society has still so much life. I hope it will continue to live and flourish until we can say that the gold mines on the Witwatersrand are not only the most flourishing but also the healthiest.

The President: This paper is of world-wide interest as it describes the first instance of a metalliferous mine being fully ventilated. I should announce that owing to our having had to postpone the meeting we are not able to have with us Mr. C. B. Saner, Mr. R. W. Schumacher, and Mr. J. G. Hamilton, the President of the Chamber of Mines, who are unfortunately prevented by a prior engagement from being present to-night.

Mr. Tom Johnson (*Member of Council*): The author has told us that although the cost of the installation of the ventilating plant was £25,000, and the running cost 1d. per ton milled, that it has already paid for itself. This is what is to be expected, and I think that the others who have installed ventilating fans might give us their experience, and if the results are like those we have just heard it would be encouragement for those who have not already done so to put in ventilating plants.

Dr. C. Porter (*M.O.H., Johannesburg*): I would like to offer my congratulations and thanks to Mr. Penlerick not only for his excellent paper but for the enlightened lead he has given in regard to mine ventilation on the Rand. I desire also to thank him for the personal courtesy he extended to the members of the Regulations Commission when they had the pleasure and privilege of going over his mine, and seeing the good work he has done there; and I think I may safely say that when considering the reasonableness of the proposals in regard to ventilation, which are incorporated in the Mining Regulations Report, the Commissioners were very considerably influenced and guided by what they had the opportunity and advantage of learning from the author.

Mr. Judge, of the Nourse Deep, has just said that in his opinion by far the largest amount of dust generated in mines resulted from blasting. He also said, if I took him correctly, that the amount generated from the actual operation of the rock-drill was comparatively small. I speak on this point with submission, because I am not a technical man; but, on the other hand, I possess the use of my senses, and I have had very unusual opportunity during the last couple of years of observing these things. So far, however, as I am entitled to an opinion, I venture to agree entirely with what Mr. Judge says; and for that reason, while I fully appreciate and certainly do not wish to discourage in any way, the efforts now being made by a medical man to prevent dust by means of what he calls a "dust destructor," but which I think would be more correctly termed a "dust allayer," I do not see how that apparatus is in any way going to affect the particular source of the evil of which Mr. Judge has spoken. I should like to point out too—and this the Mining Regulations Report (p. 42) shows—that a similar wet-bag apparatus was devised and tried by Dr. Haldane and others in Dolcoath Mine as long ago as 1904, and is thus described in their "Report on the Health of Cornish Miners," p. 26, par. 3 :—

> "We made a number of experiments as to the practicability of surrounding the mouth of the hole by a wetted canvas bag, through an opening in which the drill passes, the bag itself being fixed to a metal ring pressed by means of an adjustable rod against the rock round the hole, and luted round with clay. We found that this arrangement was very effective in stopping dust. It did not, however, appear to be practicable, as it was often impossible to fix the ring securely, particularly when the surface was very irregular, and the rock crumbly."

Whilst preserving an open mind, I am not satisfied, from what I recently observed of the working of the local edition of this contrivance at the Wolhuter Mine, that it offers any great advantage, in ordinary circumstances, over an intelligently-handled jet, and it is certainly at present much more difficult to adjust. But if this practical difficulty can be overcome, the apparatus would undoubtedly be very valuable in rises, as it would prevent much water and mud from falling back on to the rock-driller. Returning to sources and causes of dust, I also venture to believe, for I personally observed it, that in dry working places a very large amount of dust is raised by the exhaust-air from the drills. Merely to wet the dust and let it fall on the ground is not going to prevent that dust being raised again when it is dry, and the necessity for keeping the floor and sides of the working place wet is, therefore, very clear. While then one welcomes any efforts to mitigate the dust evil, I think it is right to indicate sources of danger which occur to one as not having been adequately recognised.

Next, there is the allegation made to-night by Dr. Irvine that it is not by any means certain that tuberculosis is actually contracted underground, and that therefore the proposed exclusion from underground work of white persons suffering from tuberculosis savours of medical tyranny. Now I wish to make it clear—I say it with great respect and I am sure you will not take my remarks in any other sense—that the Commission is not going to answer individual criticism. The Commission is fully aware from the very fact that its members are human, and for other reasons, that there may be, in the very nature of things, many shortcomings in its Report. It is impracticable to cover everything in a report of that kind. Coming back to Dr. Irvine's reference to the doubtfulness of mine-infection and to "medical tyranny," I therefore merely ask you to compare the remarkable statement he has made to-night with his and Dr. Macaulay's joint evidence in May, 1908. In Vol. II. (Evid.), p. 246, pars. 2 and 3 of the Mining Regulations Report, that evidence reads as follows :—

> "There is no doubt that every consumptive person is, through his sputum, a source of danger to his neighbours, and that the disease is largely spread through this agency. There is no evidence as to what proportion of white miners or natives infected with tuberculosis become so infected in the actual course of their underground work. Probably the majority are infected in the ordinary course above ground. At the same time there is a definite risk of infection being actually conveyed underground, especially when we bear in mind the considerable extent to which tuberculous phthisis exists amongst native mine workers. *It is probable that more than one in every two hundred of the latter is infected.* And this risk would of course be increased, quite apart from any question of dust, if the ventilation of the working places should be from any cause inadequate."
>
> "From the simple hygienic standpoint it would be a wise precaution, in the interest of those affected as well as that of those working with them, to prohibit all persons demonstrably affected by tuberculosis from working underground. Certainly *this provision should be made compulsory for all native workers*, although in them the generally painfully rapid course of tuberculous phthisis commonly attains the same end in a more compelling fashion. With regard to white workers, opinion is probably not yet ripe for such a step. No doubt it must be regarded as a hardship to compulsorily debar a man, however crippled, from following his occupation, but the measure would only anticipate by a few months, or at most by a year or so, the final irrevocable compulsion of the disease itself. For the affected man to continue working underground is merely suicidal, and the risk of his conveying infection to his neighbours is certainly a definite one. That such a step would greatly prolong the lives of those affected by advanced tuberculosis we do not assert, although in early cases of tuberculosis, with a slight degree of silicosis, it might certainly do so."

I have quoted fully in order to supply the context, but I invite special attention to the three concluding sentences in each of the two paragraphs cited.

Reverting to the Mining Regulations Commission's Report, I have only to say that what we have written, we have written and we stand by: and up to the present I have not heard nor read anything to induce me to alter any opinion which my colleagues and I have formed during our inquiry. But there is one thing I would like to ask—because criticism is very helpful to everyone—and it is that before people criticise they should first of all read the Reference with which the Government honoured us, and secondly that they should read the Report; because some people have criticised without doing either. I have been asked, for instance, why we did not report on the question of compensation. Well, simply because we were only told to report in regard to underground sanitation and health conditions on mines. We were not asked to report on any matters connected with compensation. Then I would ask that the Report should not be merely read, but that where it is quoted, it should be carefully read and correctly quoted, and not distorted, unintentionally I am willing to believe, as has been done by some of the most pretentious correspondents whose letters have appeared in the Press. There is a journalist in this town who I want to say most emphatically is a thoroughly honourable and upright man, a valued personal acquaintance of my own—I mention this in connection with the necessity for accuracy in use of quotation. This gentleman said in an editorial article that the Commission in general and myself in particular were greatly to blame because we waited for three years before referring to miners' phthisis and pointing out to the Government the condition of things that existed. Well, my friend had evidently not read the Report at all, although he is the editor of the newspaper with perhaps the largest circulation—if I am not libelling our other papers—for in the section of our Report which deals with miners' phthisis, is printed in full a letter which we wrote in November, 1908, to the Government calling their attention to the matter. That is two years ago. This mistake was afterwards discovered by this journalist, and he said in a subsequent article that our letter of November, 1908, was a most full and earnest recommendation; but he nevertheless proceeded to recommend—I am not a politician in any way—that the miners in Fordsburg should not vote for Dr. Krause (the Chairman of the Commission) because the Government did nothing when they got this letter from us. Now, that is absolutely untrue, and absolutely the reverse of all fair-play. This Report was sent to the Government on the 6th of November, 1908, and on the 24th December, 1908, as a Christmas box for the miners, the Government put into force the regulations embodied in the recommendations we made in this interim letter or report. And so this statement by this honourable journalist was not only absolutely untrue, but, after finding it untrue, he for some reason—for what reason I cannot say—perpetrated another gross incorrectness, and attacked the Chairman of the Commission on the ground that the Government had not done what you will find by looking at the *Gazette* and looking at our Report, it did and did promptly. I would say again that the Commission does not intend to defend anything said in that Report. We are satisfied as to our findings, which are based impartially on the evidence before us, and we do not care two pins whether they please you or displease you or anyone else.

Perhaps I may now be allowed to refer to the suggested connection between Johannesburg road-dust and silicosis, though I do not intend to express any final opinion at present on the assertion that has been made, that people who live in Johannesburg, but never go underground, are dying like flies from silicosis. I merely ask you to consider the following facts in connection with this curious allegation. I am informed by several of the leading medical men in Johannesburg, including Dr. Mackenzie, who has unusual opportunity at the Hospital of knowing the results of *post-mortem* examinations regarding deaths from respiratory diseases—that there is no evidence whatever of deaths from silicosis except in persons who have been working underground. The second point is, as shown by your President this evening, that except in the vicinity of the mines the dust is rounded and relatively harmless as compared to sharp dust, and contains much organic matter; and if the older members of the Society will carry their minds back to the slides shown by Dr. Pakes in 1903, they will recall the same fact. Thirdly, I have worked out the death rates from pneumonia per thousand (without correction for age-distribution) in Johannesburg during the three years 1906-1909, and compared them with those in England and Wales—including the country and towns—during the same period. Now, in Johannesburg from 1906 to 1909, the death-rate from "pneumonia at all ages" was 1·31 per thousand. In England and Wales it was 1·24. The rate from "lobar pneumonia at all ages" was 0·88 in Johannesburg and 0 71 in England. From "broncho-pneumonia at all ages" it was 0·43 in Johannesburg and 0·54 in England and Wales. There is no very alarming difference there; certainly not the difference one would expect if everyone was at risk of death from silicosis in Johannesburg. Now, in order to get at how pneumonia specially affects *adults*, I took "lobar pneumonia *over 15 years of age*." The death-rate therefore was 0 71, while in England and Wales it was 1·00. For broncho-pneumonia it was 0·09, and in England and Wales 0·26. I am aware that one cannot always come to a correct final conclusion in a matter of this kind on figures alone; and there is no doubt that the inhalation of dust does markedly predispose to respiratory diseases. But I know of no satisfactory evidence that non-mining people generally, who are walking about the streets of Johannesburg, are at risk of dying of silicosis; and so far as I can find from inquiry there is not a shred of justification for such a statement. But I do not possess any recent details of age distribution of the population of Johannesburg, and have therefore not made any correction in this respect in the rates quoted. We know, however, that in Johannesburg there are a great many more young people per thousand than in England; and, as pneumonia is a disease of old people particularly, it is probable that our true pneumonia death-rates are correspondingly higher than the figures quoted would indicate. But, even then, they would not justify the statement in question, so far as I have been able to ascertain. At the same time I think such suggestions are often very interesting, because they indicate directions of useful inquiry; and one naturally values them much more when they are not merely expressions of opinion, pious or otherwise, but are backed up with something like tangible evidence. I wish most cordially to thank and congratulate Mr. Penlerick in regard to his paper.

SAND-FILLING ON THE WITWATERSRAND.

(Read at June Meeting, 1910.)

By Edgar Pam, A.R.S.M. (Member).

DISCUSSION.

Mr. O. P. Powell (*Member*): Mr. Pam's paper deals with one of the most important mining subjects that has ever been discussed by this Society, and is of great value as a record of pioneer work here of sand filling on Silesian lines. When sand filling was first seriously considered on these fields a certain amount of apprehension was felt as to the lowering of current sand residues underground, an apprehension that was not altogether dispelled by the earlier investigations and tests of possible cyanicides. Consequently, as stated by Mr. Pam, old sand residues free from cyanide have hitherto been used at the Village Main Reef and elsewhere

At the Simmer & Jack Proprietary Mines, Ltd., a sand filling plant has now been in operation on novel lines for close on two months, and from the beginning has been treating current sand residues direct from the cyanide vats, with satisfactory destruction of the cyanide solution moistening them. The system employed comprises the pumping of sand residues as a pulp from the vats to classifiers which deliver a thick underflow into a bore-hole, and return in circuit the overflow water to the pump for bringing up more sand. (See Fig. I.)

The following is a more detailed description. The sand residue as discharged from the vats and containing weak cyanide solution is dumped from trucks into a box beside the track clear of the vats. Into this box water is constantly sprayed, and the pulp thus formed gravitates to a pump and has a ratio by weight of about 3½ of water to 1 of sand. This pulp is then pumped through a 6 in. pipe for a distance of about 900 feet to classifiers at a 6 in. borehole 200 feet deep. The mixing box is 10 ft long, 5 ft. wide and 4 ft. in depth, and discharges into a launder 20 ft. long with a uniform grade of 8%. This graded launder ensures the pump being supplied in a steady manner with a uniform mixture which is essential to efficient pumping. The truck discharge and wooden mixing box will ultimately be replaced by a system of launders running below the discharge doors of the vats. These launders will convey the sand pulp to the pump, and as is the case with the present arrangement a uniform mixture will be provided, as the pulp will not flow until the requisite quantity of water has been added. A 4 in. centrifugal sludge pump is at present in use. The delivery column rises constantly from the pump to the classifiers at a 60 ft. higher level, and will serve for a larger pump should we decide on replacing the present one. By having a delivery pipe of larger diameter than the delivery of the pump the velocity of the pulp is reduced and the wear of the pipe reduced by partial settlement of sand on the lower side. The underflow from the classifiers at the borehole containing about 30% moisture is delivered by launders with a grade of 30% to the mouth of the borehole down which it falls. The two diaphragm cones (8 ft. diameter, 10 ft. deep) which we started with handle between them 200 tons of sand per day of ten hours. Two additional cones have now been erected, and with a pump of larger size the plant will deal with some 400 or more tons of sand per day of ten hours. (See photograph.) As the classifier underflow entering the borehole is very thick the make-up water necessary to be added to the small volume of water in circuit is comparatively small. The sand residues contain about 12% of moisture as cyanide solution containing up to ·02% total cyanide. It is proposed to instal a vacuum pump*

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* See this *Journal*, Vol. ix., Jan., 1909, p. 240.

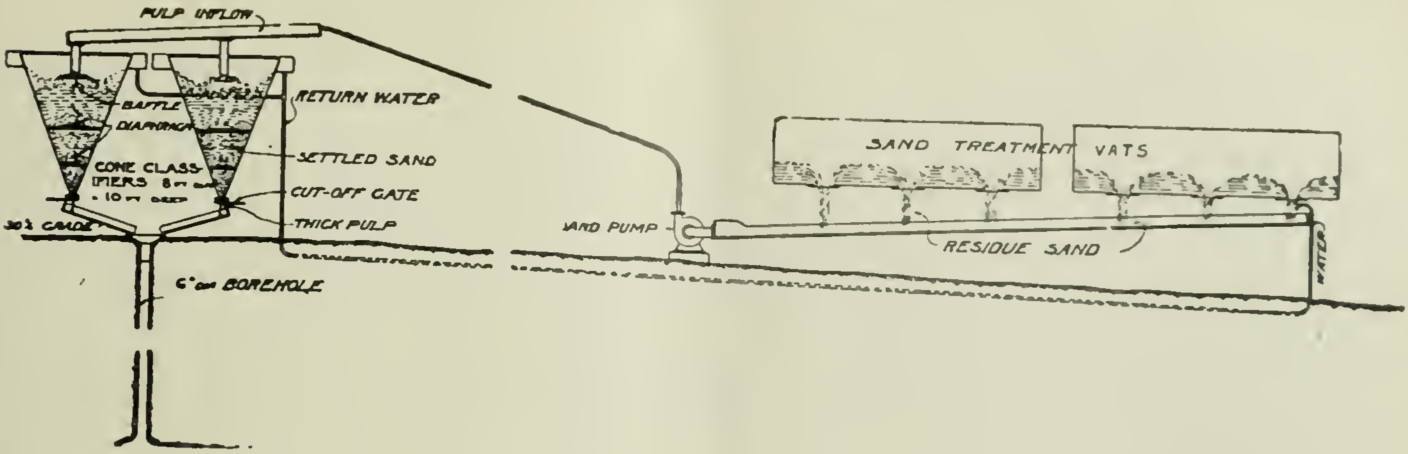
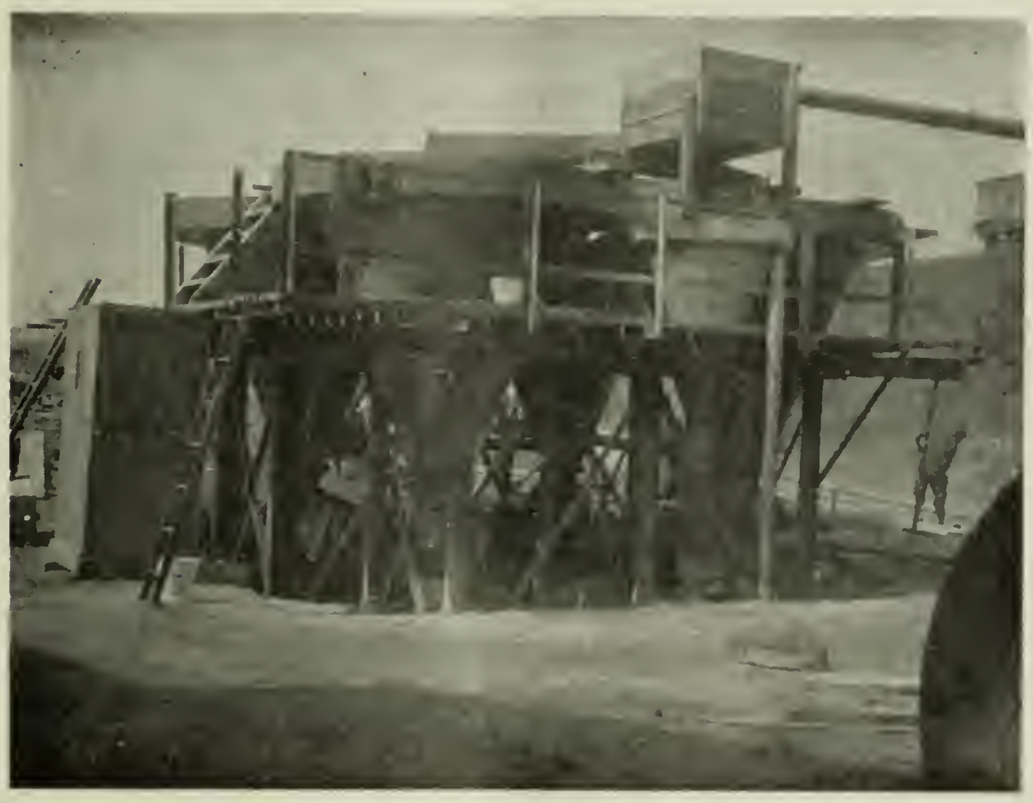


Fig. 1.—Sand Filling Plant. Elevation showing General Arrangement.



connected to the leaching pipes from the vats, which will still further reduce the moisture in the residue, and consequently the cyanide to be destroyed, as well as remove all the gold-bearing solution possible.

The various cyanicides suggested by Mr. H. A. White and others have been tested in our laboratory by Mr. McA. Johnston, and until recently we have used permanganate of potash* on a working scale as the most suitable, but we are now using the cheaper chloride of lime† satisfactorily. By adding ·20 lb. to 0·25 lb. of the former reagent in the form of a 5% solution to every ton of sand dumped in the mixing box adequate destruction of cyanide is effected. Frequent tests are made of the moisture in the pulp entering the bore-hole from the cone underflow, and in no instance has this contained more than ·0025% of total cyanide, whereas the limit allowed by the Westralian Government is ·010% total cyanide. The circulating water is slightly alkaline owing to the alkalinity of the moisture in the fresh residue.

From the point where the cone underflow leaves the bore-hole it is conducted by wooden launders to the stope to be filled. A small stream of water underground is diverted into the launder, in order to form a pulp which will flow on the grade available as this is in places considerably under 30%.

Tests have been made of the air in practically every portion of the stope that was being filled and nowhere could a trace of hydrocyanic acid be found. The water draining from the deposited sand was also tested with similar results, which would indicate a further neutralisation of cyanide taking place after the pulp enters the bore-hole. The sand settles compactly underground and drains readily. Tests made some time ago by Mr. J. A. Vaughan at the Government Mechanical Laboratory showed only 4·5% compression of water-deposited sand supported at the sides under a pressure of 300 tons per sq. ft. It is obvious that the cleaner the sand available for filling the less will be the trouble underground with slime*.

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* See this *Journal*, H. A. White, p. 443, June, 1910.
† See this *Journal*, H. A. White, p. 15, July, 1910.

Although we have been working on a relatively small scale, the actual operating surface costs are remarkably low—a little over 4d. per ton of sand lowered. Power costs a 1d. and neutralising with permanganate about the same amount per ton.

The method described affords various advantages as follows:—

(*a*) The pulp entering the bore-hole contains about 30% of moisture as against the 40% to 50% moisture usually employed, so that much less water than usual, which is liable to contain traces of cyanide, requires to be pumped from the mine. Assuming that 1,000 tons of sand are lowered as pulp into a mine to drain there until only 12% of moisture remains, then the water requiring to be pumped out of the mine amounts to only 292 tons if the pulp lowered contained 30% moisture, but it will amount to 531 tons if the pulp contained 40% moisture, and to 864 tons if the pulp contained 50% moisture.

(*b*) A much longer contact and more thorough admixture is available for action of the cyanicide before lowering than in the usual system of sluicing direct into the mine.

(*c*) Up to 90% of the water in the sand pulp entering the classifier overflows, and thus at once returns for further use any excess of cyanicide dissolved in it.

(*d*) Owing to the small amount of water in circulation any soluble gold present in the residues becomes concentrated to a certain extent in the circulating water, which may be periodically transferred with its slime in suspension to the slime plant for recovery of gold contents, or otherwise treated.

(*e*) The velocity in launders or pipes of the concentrated pulp underflow from classifiers is lower than with a higher ratio of water, and consequently the wear by abrasion less.

(*f*) The elimination of slime in the water overflowing the classifiers permits of a sandy underflow more free from slime, and the sand deposited in the stopes drains more rapidly and completely.

(*g*) The capital expenditure and operating cost of the installation is low, the former being mainly for launders, a pump, two pipe lines and classifiers, whilst for the latter the expense of pumping is the main item, and with a large daily tonnage the total working cost, including destruction of cyanide should be less than for sending residues to the dump.

The conditions governing the surface transport of sand for filling will vary on different mines and will influence the choice of a point of lowering, particularly where a bore-hole is to be employed. When this point is on a lower level than that of the sand vats the pulp can gravitate to the classifiers and only the overflow water will need to be pumped (Fig. II.). Where there is no natural fall it may be more advantageous to drive a tunnel or cut a trench than to pump the sand pulp.

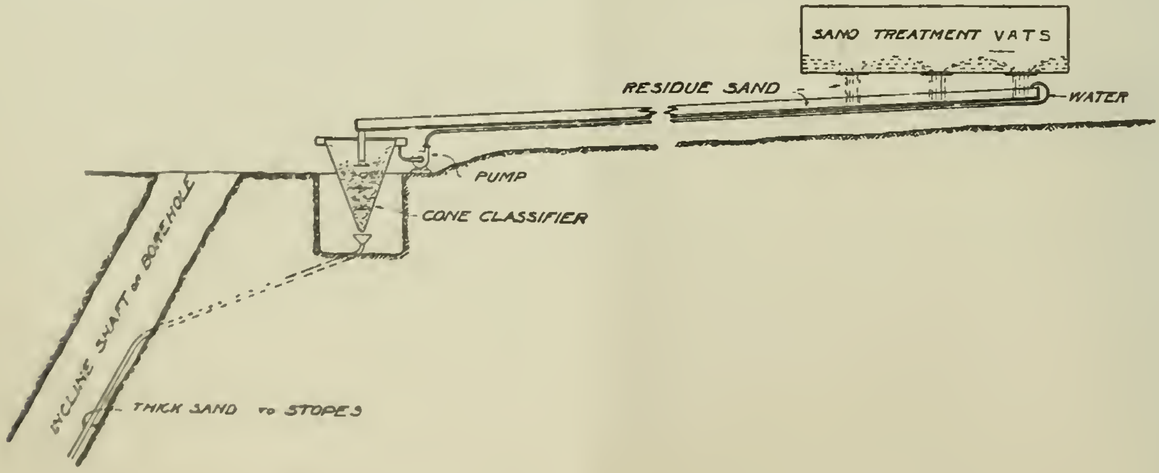

FIG. 2.—Sand Filling. Sand direct from Vats to Classifier at Borehole by Launder.

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* *S. A. Mining Journal*, p. 610, July 23, 1910.

Among the many advantages of sand filling, to which Mr. Pam has referred in his paper, that of improved underground ventilation is one of the utmost importance. In outcrop mines the question of ventilation does not usually present any serious difficulties. In the deep levels the worked-out area, constantly increasing in size and providing space for the dissipation of air, will be filled in, and by keeping such filling as close to the working faces as is practicable, every cubic foot of air will be usefully employed,

Mr. B. C. Travers Solly *(Member)*: I was working on the Great Boulder Proprietary Mines, Western Australia, when the filling of stopes with residues was started, sometime in 1897 or 1898. and knowing that the Government had, owing to accidents occurring, stopped the system, appointed a Royal Commission, and on its findings passed regulations under which the sand filling was restarted, I wrote in June last to Mr. Richard Hamilton, who was then and still is General Manager of the Company, asking him for particulars of the conditions under which the filling is now carried on, and I have just received his reply.

Inter alia, he says :—"The mines on this field have, as you know, dumped their tailings into the stopes in a fairly dry condition. Those using the residues direct from the presses gave them a good drying with compressed air before dumping them into the stopes. This applies both to the raw and the roasted residues. Our mining regulations specify that the strength of cyanide of potassium in the residues must not exceed ·01%, and the mines find they can comply with this regulation without trouble. The ore on this field contains generally somewhere about 13% of carbonate of lime in the raw state. In roasting, this is converted into calcic sulphate and calcic oxide, while there is also apparently some unaltered carbonate left. With the roasted ore you would not expect so much decomposition of cyanide as with the unroasted ore residues. The raw treatment plants go in for very close concentration, and in some instances they dump residues direct from the presses, and so long as the stopes into which the residues are tipped have free circulation of air there are no attendant cyanide troubles. Where passes have been blocked so that a current of air was stopped there have been instances of men being temporarily overcome by cyanide fumes, but I do not know of any during late years. When these instances occurred there was more ore percolated than at present and salt water was used for washing, and I have no doubt that the residues contained more than ·01% of cyanide of potassium. No mine in W. A. has ever tried to fill stopes by running residues into them by means of water."

It seems to me therefore that the Regulations proposed by the Transvaal Commission, whose report has lately been published, have been taken directly from those in force in W. A., without any thought of the vastly different conditions with which we have to deal.

The recommendations of the W. A. Commission, which may be found in "Notes and Extracts" in the November, 1905, number of the Journal of this Society, read :—"That tailings should not be used for filling (1) in a wet state, (2) when they contain more than ·01% of their weight of cyanide of potassium, (3) in any place where there is not a current of air passing freely."

No. 1 is, however, translated into practice in W. A. as "in a fairly dry condition." The draft Regulations proposed here state that tailings so used must be previously dried.

No. 2 is mutilated in passing from the recommendations of one Commission to those of the other, and we are left to surmise in the latter whether the ·01% of cyanide of potassium is to be calculated, as in the W. A. Regulation, on the weight of the residues or on the moisture contained in them. If the latter is intended the regulation would be reasonable. If the former, the essentially differing conditions of the ores of the two fields make it a highly dangerous allowance. In W. A. the condition No. 2 is, as Mr. Hamilton states, easily complied with, and seeing that, with an allowance of 15% for residual moisture, ·01% on the residue works out at something over ·06% calculated on the contained solution, this is not to be wondered at.

While this limit is safe in Kalgoorlie-Boulder, where the ore contains some 13% of carbonate of lime, it would be highly dangerous on the Rand, with the acid forming nature of our ore.

The main points which strike me as being so widely different in the conditions on the Kalgoorlie-Boulder belt and on the Rand, apart from the chemical conditions of the ore, and which make their method of "dry" sand filling an economic impossibility here are as follows:—In nearly every instance in Kalgoorlie-Boulder the mines are outcrop, the veins are wide (I have worked in stopes 40 feet from wall to wall) and they are very steep. The residue is trammed from vats or presses and dumped directly into cribbed shoots, which lead to the underground workings where the filling is required. The method of mining is that of back stoping, and the sand would run to its angle of repose directly under the working face. Here, however, we have, so far as the future is concerned, to deal mainly with deep level mines and comparatively narrow and flat ore bodies, where in filling with "dry" sand it would be necessary to have an extra compartment in the shaft, tramming on an up grade to the stope to be filled, and, in most cases, mechanical conveyance of the sand down the stope to the portion being filled.

It will be noticed that Mr. Hamilton states :—"No mine in W. A. has ever tried to fill stopes by running residues into them by means of water." There was no inducement to do so. There was every reason against it. At the time when sand filling was started water of any kind was scarce, having to be pumped from Hannan's Lake miles away, and fresh water cost up to 15/- per 100 gallons retail. I believe at the present time, with the extensive water supply system, it costs 2/6 or 3/- per 1,000 galls.

Taking these points into consideration I am not alone in thinking, that this Society would only be doing its duty to the industry by entering, without loss of time, a strong protest against the promulgation of these proposed Regulations.

Mr. H. F. Lofts, M'Toko, Rhodesia (*Member*) : The extremely interesting articles on "Sand Filling on the Witwatersrand," by Mr. E. Pam, followed by "The Destruction of Cyanide," by Dr. Moir and Mr. Jas. Gray, and the copious notes by Mr. H. A. White, must appeal to all of us, as touching almost the most important subject ever dealt with by this Society. Therefore I, in common no doubt with many others feel I should like to add my quota to the voluminous discussion which I am sure is bound to follow three such striking papers.

First, I understand the residues contain iron salts, pyrite, and cyanide. These in turn produce (if old) sulphocyanide and HCN, and the object of these tests is mainly to prevent the formation of HCN when the sands are safely lodged in the stopes, as it is very injurious to health.

Of these papers, Mr. Pam's shows the practical aspect, Messrs. Moir and Gray deal with the chemical details, and Mr. White with the experimental side and costs per ton.

With reference to Mr. Pam's article, I would suggest the use of launders instead of pipes, because they would pan out considerably cheaper than $\frac{7}{16}$ piping, and lengths could be added and alterations made with less expense, also a launder is less liable to choke than a pipe. In order to make a tight joint in the planks erected in stopes, a system of grooving and tongueing with soft timber similar to that used in the bottom of cyanide vats, if bolted would be perfectly water-tight, and there would then be no possibility of the barricades breaking as suggested by Mr. Laschinger, and no necessity for the tedious method of filling up the crevices with grass. One of the advantages not mentioned by the author is that it would induce cyanide men to work with as weak a solution as they possibly can, and battery managers to make use of lime, thereby reducing costs in one department and increasing efficiency in the other.

Dr. Moir states that ·002% KCN yielded the best results, and I fail to see why solutions of this strength should not be in general use, personally I have used ·001 on a practical scale and obtained 100% precipitation, and found the solution as rich in gold as others of much higher strength. Working with this strength, it was important to keep the extractor well filled with zinc, to use acetate of lead and occasionally add some KCN at the head of each box to prevent the formation of cyanide of zinc. There can be no necessity for the use of a strong solution, the coarse gold being caught in the battery and by the concentrator. As even the water wash carries off 0·25 dwt. Au it is obvious that the only necessity for increasing the strength of a solution is for precipitating purposes. As it is it appears necessary first to oxidise and then to produce an alkalinity, I would like to suggest that as sal ammoniac is in such general use as an oxidiser a few more experiments with this substance might not be amiss. From Dr. Moir's remarks, it is evidently of very little use to consult the text book, but a question of experimenting in every likely direction.

A final water wash after treatment with very weak solutions, should remove most of the cyanide, and it is possible then that with the use of lime in the battery very little more would be required. It resolves itself into a question of how best to deal with the total cyanide, the acid itself is harmless except in conjunction with cyanide, as far as the question of health is concerned, and the volume of water in use to carry the sands down the mine would probably neutralise any free acid which is formed, so that the water when returned by the mine pump would be ready for the reduction works and boilers.

The question of the large space taken up by dumps, especially from those mines centrally situated is one of an importance which cannot be under estimated, as before long it must involve the purchase of stands as depositing sites at high figures. It is to be hoped that before long it will be possible to deal with all cyanide sands without handling, by a first and final treatment in the collecting vat and a sluicing away of the residues through a pipe with a gate valve, direct to the mine. The thorough ventilation of the mine and dispersal of the CO_2 which must follow, will naturally enable all workers, whether white or coloured, to do the maximum of work in a given time, which is impossible when the air is vitiated.

Mr. White has stated that permanganate is the best and cheapest reagent to employ, but, of course, it must also be borne in mind in this connection that simplicity is always to be desired. Any complicated tests would involve delay and much trouble. One wholesome result, which is not to be despised, is that it should tend to encourage the retreatment of tailings all over the world. On some mines which are just on the border of the profit-making stage, the gold obtained in this way would help them materially towards the dividend paying stage. It has now been proved that the gold left in the residues is not (now fine grinding is resorted to) in a free state but combined with the pyrites: therefore a thorough concentration would eliminate practically all the gold-bearing material and would be the end point as far as the reduction works are concerned. If you take the average mine dump as assaying ·5 dwt. this would result in the case of a 100 stamp mill dealing with 12 tons per stamp in a recovery of practically 30 oz. per day, equal about £40,000 per annum, quite a round sum. To emphasise the speed with which a dump could be transferred, the Robinson Company put through 200 tons per hour, equal to the output of a 400 stamp mill. This sand filling, I am informed, has been in use in Australia for some years, but has not become quite a success owing to their failure to remove all the HCN. It is not necessary that sand-filling should be limited to the Rand, an up-country mine is quite as much justified in making use of the latest ideas involving increased profits and efficiency as the Rand mines. So often because a property is a small one and situated up country the owners think it is only advisable to work in a rough and ready way. If this is an advantage on the Rand, where so many mines are low grade, how much more important and profitable it would be where only high grade propositions are working and where all expenses (and in this comparison the difference is not only a trifling one) are correspondingly higher and heavier, including timbering, transport of timber, wages and cost of handling residues. This latter point is one which though already mentioned, should be emphasised. The increasing scarcity of mine timber renders it more expensive every year, therefore the outcrop companies (this would not apply to nearly the same extent to the deep levels) should find a considerable diminution in their expenses on this account alone.

We notice in town the roads are being watered with oil and asphalted, trees are planted to filter the air, and in fact no expense is spared to minimise the dust evil, but all this is a step in the wrong direction. The movement should have commenced at the mines, whence the poisonous dust originates. Laws are passed compelling factories to consume their own smoke: it is surely (on the score of health alone) just as important to pass an act compelling all mines to deal with their residue dumps for their own benefit and that of the whole of the South African public. The only way in which attempts to utilise the dumps have been made is in building; there are no other industrial uses to which they can be put. This may be compared to the proverbial emptying of the sea with a tea cup. I understand many dumps have been retreated in the past few years, but this meant simply removing the sands from one spot and placing them in another. The system now in use goes a step further and completes the process. For this reason I am sure Messrs. White, Pam, Moir and Gray earn the lasting thanks not only of the community but the public generally, in originating and improving upon a system which is likely to have such far-reaching results.

Mr. W. J. N. Dunnachie (*Associate*): In connection with Mr. Pam's most interesting paper on sand filling, there is a suggestion that the sand pack, owing to a want of cohesion, may be limited to the life of the timber enclosing it, and when that life is spent the sand may "run."

I throw out the suggestion that *self-supporting* waste packs be built; with a foundation cut in the footwall, if thought necessary. These waste packs should be built on the inside—that is the sand side—of the 1½ in. deals and uprights which the author is at present putting in. If all the protection the Village Deep and Ferreira Deep levels have to keep the sand from coming through the boxholes, is a few "uprights well hitched" with some 1½ in. deals nailed on, I feel certain those 1½ in. deals are in for a tough time.

The President: I have just had an official letter from the Government Analyst of Western Australia in which he re-affirms the statement that the ·01% limit of cyanide refers to dry sands and not to any liquid contained in them, and that nothing except dry filling is done there, and that thorough ventilation of the place being filled is prescribed whenever any person is present there. I agree with Mr. Solly that a ·01% limit is dangerous under local conditions, and that the limit should be stricter.

NOTES ON PRECIPITATION.

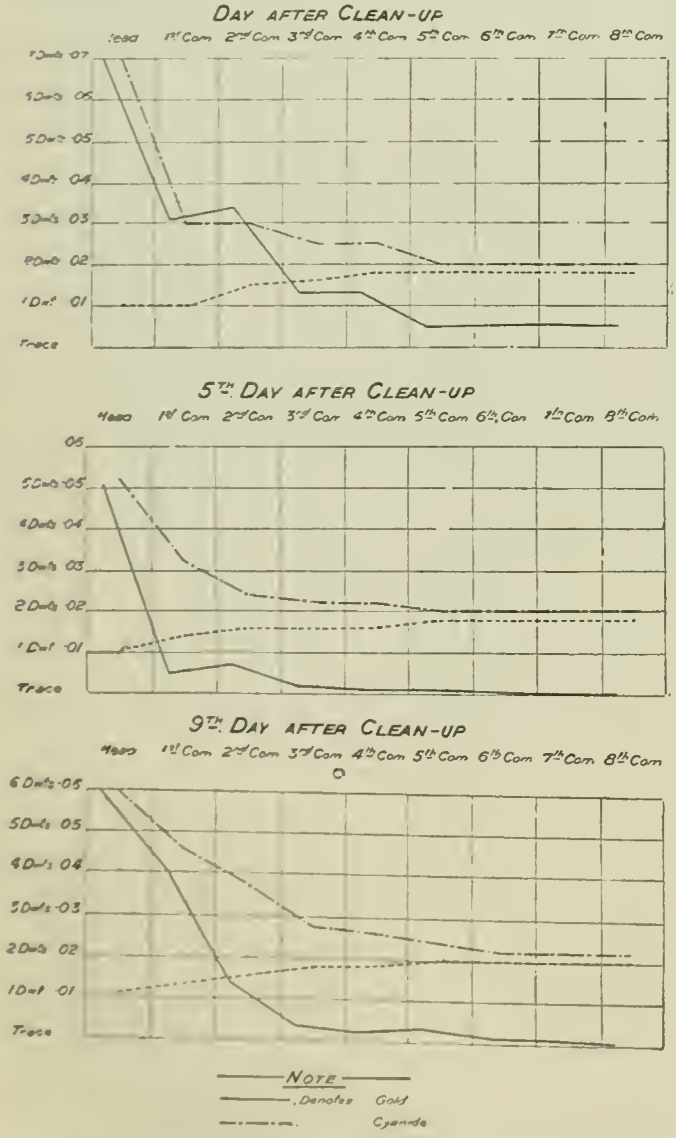
(Read at July Meeting, 1910.)

By F. D. Phillips (Member).

DISCUSSION.

Mr. H. Brazier (*Associate*): The points raised in this paper in regard to the last two or three compartments of an extractor box were gone into very thoroughly on the Crown Mines some time back, with results very similar to those obtained on the Knights Deep; and it was during these investigations that a certain fact came to light which is of considerable interest and which seems hitherto to have escaped notice, or at any rate, publication.

The solution deposits its gold, loses free cyanide, and gains in alkalinity, and by representing these three occurrences graphically we find that all three reactions coincide. I append three diagrams taken from a series, all of which demonstrate the same fact, and it is from the information conveyed by the diagrams that I venture to query the result of the author's calculations in regard to expense. There being no further reaction after the precipitation, wherein does the extra expense enter beyond the labour of dressing? Also, allowing that there is no further decomposition it will surely be admitted that the zinc is likely to be of more use in the boxes than in the store, even if only in the light of a safety measure, or as a filter for trapping particles of slime that may and often do come along.



Note error in two lower diagrams: The "gold" line should run out parallel with "cyanide" and "alkalinity."

Regarding the effect of temperature on precipitation, the slime solution here is divided between two sets of boxes—1,000 tons a day is passed through one set of boxes direct from clarifier; another 1,000 passing *via* a waste heat installation and clarifier, and entering 10° F. higher—the precipitation in both cases being practically perfect, the "tails" invariably assaying "traces," capacity and rate of flow in both cases being identical.

Will the author enhance the value of his paper by giving actual temperature of solution at head of boxes? One is led to think that the comparative absence of white precipitate at Knights Deep is attributed to increased temperatures, whereas our experience is the reverse—the formation of white precipitate in the warm boxes being far more pronounced than in the cold, and where any appreciable quantity of this white precipitate is formed the author will, I fancy, agree that more zinc is required in the boxes than would be the case in clean solution such as at the Knights Deep.

GRADING ANALYSES AND THEIR APPLICATION.

(Read at May Meeting, 1910.)

By H. Stadler (Associate).

DISCUSSION.

Mr. G. A. Robertson (*Member*): It is well to read the author's paper in conjunction with his other contributions partially dealing with the same subject. Taking the efficiency of stamps as zero the author finds that "from a purely mechanical point of view tube mills are very poor machines at least in comparison with stamps.* To arrive at these conclusions the author evolves formulæ for the determination of the useful work done by crushing or grinding machines and arriving at the mechanical efficiency of such machines, "29, 30."* The chief consideration there is the reduction of the cube by volume and the energy (H.P.) required for this reduction, the same formulæ being applicable to all types of grinding machines irrespective of the size of the feed cubes or even where such cubes can be reduced by a few grades only, such as for instance the work done by tube mills.

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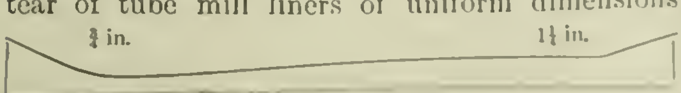
* See *Journal S.A. Association of Engineers*, Dec., 1908.

"Stamps though no ideal fine grinding machines are certainly better than their reputation."* Since the author penned these words he now finds from practical work that the "efficiency per H.P. of tube mills (17·42) is only about ¼ that of the stamps (70·10)." The "practical usefulness" of the tube mill, as the author said in reply to the discussion on his paper,† had never been brought into question nor does it appear that even now the good work done by tube mills has ever been questioned except from a "purely mechanical point of view." The writer does not know the tensile strength of a cube of pyrite in relation to a cube of quartz, but the author makes no distinction in arriving at the mechanical work done by stamps and tube mills. In arriving at the efficiency of the tube mill the author takes the load at 290 tons. I gather that this is the average load during the life of a Hornstone tube mill liner which can be reduced in volume though of course it is profitable to put through a much larger load from a metallurgical point of view; but when we go much over a 300 ton load, though we may not reduce any additional tonnage in volume we utilize the tube mill as a classifier, the tendency being for the coarser and specifically heavier particles to be reduced in preference to the more silicious portion of the ore.

The tube mill in itself is a classifier; the greater part of the coarser and specifically heavier particles of ore are reduced on the liner of the tube. By the rolling action of the cascade of pebbles on a grinding surface (the liner) these particles are reduced perhaps by a few grades only, and the smaller these particles become as they travel towards the discharge end of the tube so have they the tendency to go into circulation amongst the pebbles, depending of course upon their specific gravity, when they undergo but little further reduction. This is partially due to the speed of the mill being the same both for coarse and fine grades, whereas in reality the finer the grades become the better the settlement required and this can only be effected by decreasing the speed in feet per minute at which the liner is travelling. The accompanying sketch represents the wear of a common iron "rib" of the Osborne patent tube mill liner the measurements being ¾ in. at inlet and 1¼ in. at the discharge reckoning 2 feet from the extremities. This would appear to be about representative of ordinary wear and tear of tube mill liners of uniform dimensions and hardness. The wear there is almost entirely due to the abrasion of the coarse sand and its effect on the liners as the sand gets finer. It may be argued that this uneven wear is due to the rough edged and larger pebbles fed at the inlet end of the tube, but as the same wear was in evidence on Silex liners in the earliest days of tube milling on the Rand this argument gives no explanation, more especially when in those days round imported flint pebbles were only used. The discharge end of the tube has obviously less work to do than the inlet end, and at the same time we give it the same dimensions as regards the diameter and pebble load. It was found more economical in Australia to cut down the tube mills to 12 × 13 ft.? thereby increasing their efficiency per H.P. The Rand has arrived at a standard size of tube mill, 22 ft. × 5 ft. 6 in., a recent exception to the rule, however, being the tube mills of the Roodepoort United, whose dimensions are 16 ft. 5 in. × 5 ft. 9 in. As I have already indicated, the finer the sand becomes the greater its tendency to be stirred up and take part in the turmoil and motion of the pebbles, and I am of opinion that the Hardinge conical mill is much more efficient than the cylinder tube mill due to better settlement being allowed for the finer grains, less interstitial space and less pebble load where less work has to be done. Improved concentration within the tube mill would also be obtained, due to the fact that the pulp has to travel up an inclined channel from the time that it reaches the liner of the tube at its greatest internal diameter.



The tube mill is much more fool proof than its reputation, and when we consider that with a good iron liner exposing as large a grinding surface as possible, a 6 months run can be obtained without a single renewal of a wearing surface, it is obvious that this machine requires but little attention. The mechanical efficiency of the tube mill is evidently low, that is to say if no credit is to be given it for the hard work we get it to do. The author has no doubt some good reason for taking the whole as one, and by his method of calculation based on "Kick's Law" referring to impact "the amount of energy absorbed is proportional to the volume of the body to be crushed."

The idea of a well run ball mill is to have balls of many sizes, ranging from 6 in. dia. downwards thereby having a minimum of space between the balls. Assuming that 1 cub. ft. of ore is 168 lb. as against 451 lb. for iron, it is obvious that a good deal of crushing is done amongst the balls on account of their being so close. An examination within a tube mill will show that the largest of the pebbles are at the inlet end of the tube, and it is safe to say that there is even sufficient

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*See *Journal S.A.A.E.*, Dec,. 1908.

†See *Journal S.A.A.E.*, Jan., 1910, also this *Journal* July, p. 20.

space between the pebbles to afford shelter for a number of snakes !

In the writer's opinion there are several alternatives which would tend to increase the efficiency per H.P. of the cylinder tube mill, most of which are already in use in other industries where grinding machines form an essential part of the equipment.

(1) Taper off the diameter after the first 14 ft. the liners running up to the perimeter of the discharge screen.

(2) Discard trunnions and set the tube mill on rollers, the bands and tyres being on the perimeter of the tube. Gear in the centre.

(3) Dry grinding a classified ore from the jaw crushers with peripheral discharge.

(4) Partition off a section of the tube mill and make it in conformity with the "Molitor Compound Mill" iron or steel balls in the first compartment reducing the coarse particles of ore which in ordinary practice takes many more tons of pebbles.

Under 1 and 4 the inventor's royalties would no doubt have to be paid. No. 2 has already been tried as far as regards the bands and tyres on the perimeter and set on roller bearings, but was not a success. Considerable improvements, however, have been made in roller bearings, etc., of late, and the writer has every reason to believe that with the gear in the centre less H.P. would be absorbed. Another advantage in the successful operation of that type of tube mill is that we have full command over inlet and discharge areas, whereas at present we are entirely limited to the internal diameters of both trunnions. No. 3 has already been tried in wet grinding but without success, partly due to the fact that the peripheral discharge allowed an immediate release of the specificially heavier particles which reached the discharge end of the tube. Without water dry concentration will take place inasmuch that the silicious portion will find its way to the discharge before the pyrite. To get full advantage of dry concentration here the design of the liner requires to be taken into account.

By a combination of Nos. 1 and 4 the writer is of opinion that good work could be done. A good portion of our scrap iron or steel might also be thus turned to profitable account.

The question of hydraulic classification and regrinding in stamp batteries has been advocated from time to time in this journal and other sources, and the idea has even been tried upon the Rand but without success. A recent note, however, which appeared in the *South African Mining Journal* of August 13th evidently bordering upon this question, stated that "the New Kleinfontein mill is being prepared for crushing operations, in which the work of the tube mill, now considered an essential almost of the metallurgical scheme of the Rand, will be done much more efficiently, conveniently and cheaply by stamps alone." If that is so we must change our minds in accordance with the times, and it is to be hoped that the author will be in a position to give us the details of the experimental data there conducted, which I understand was on behalf of the Mines' Trials Committee.

The meeting then closed.

INSTRUCTIONS FOR HANDLING EXPLOSIVES.

Drawn up by a Sub-Committee of the Council of the Society, and approved by the Transvaal Chamber of Mines for Exhibition on all Mining Properties.

The following instructions must be strictly observed, as any deviation from them may cause a serious accident :—

Detonators.—Detonators must always be kept dry ; moisture or dampness, even when invisible, reduces their effectiveness, and may even cause misfires. Before inserting the fuse, the sawdust must be thoroughly shaken out of the detonator tube. This can be done by gently tapping the open end of the detonator upon a piece of wood. On no account must any instrument, whether of metal or wood, be inserted into the detonator tube to clean out the sawdust. Should it be found that the sawdust, owing to dampness, cannot be thoroughly shaken out, the detonator must be placed aside as unfit for use.

Preparing a Charge.—Safety fuse of the required length must be cut square with a sharp knife, and then inserted into the detonator tube until it reaches the fulminate. The top end of the detonator tube must then be squeezed round the fuse with special fuse pliers. Care must be taken not to squeeze the lower end of the detonator tube containing the fulminate, otherwise an explosion is likely to occur. The wrapper at one end of a cartridge must next be opened and a small hole made in the explosive by a piece of wood or the copper end of the fuse nippers. Into this hole the detonator (with fuse attached) must be inserted to two-thirds of its length, and the cartridge paper must then be firmly tied with string. On no account must the detonator be buried in the explosive (unless it be an electric detonator).

Safety fuse must always be handled with the greatest care. The powder core of even the best fuse can easily be damaged by twisting or kinking the fuse, and this in turn may lead to misfires.

Charging the Hole.—Cartridges must be inserted one at a time into the hole, and each one squeezed gently but firmly home. The primer (viz, the cartridge with a detonator and fuse attached) must always be placed on the top of the charge, care being taken to use as little pressure as possible while inserting it. In wet workings the junction of the fuse and detonator must be made water-tight, either by means of Chatterton's Compound, insulating tape, or tough grease.

The tamping bar must be of wood, but the end may be sheathed with either copper or brass. Under no circumstances must either iron or steel bars or scrapers be used.

Tamping.—The material used for tamping should be preferably of a clayey nature, and should be made up in the form of cartridges of a slightly smaller diameter than the hole. In order to avoid any interference with the detonator, the first tamping cartridge must be put into the hole with the greatest care. The remaining tamping cartridges may then be firmly but carefully tamped. When tamping, care must be taken to avoid withdrawing the primer, etc., from the charge.

Lighting the Fuse.—The fuse must be prepared for lighting by splitting the end about an inch down with a sharp knife. As soon as an efficient fuse lighter is available the cheesa stick made of blasting gelatine should be prohibited, as poisonous gases are formed by this burning explosive.

Misfires.—In the event of a misfire, only sufficient tamping must be withdrawn by means of a wooden swab stick to enable the misfired shot to be fired. The procedure to be followed must be strictly in accordance with Section 9 of the Mining Regulations.

Plugging of Sockets.—After a blast, all the rock thrown down, and also all the remaining holes or sockets, must be carefully examined for unexploded cartridges or detonators.

General.—All nitro-glycerine explosives are, when the temperature is below 45° F., liable to congeal or become hard from the effect of cold. On no account must they be used when in this condition, and not until they have been carefully softened or thawed in a warming pan specially constructed for the purpose.

All open lights must be removed to a safe distance before any case containing explosives is opened, and also during the preparation of the charges. No light of any kind must ever be placed on the cover of a box containing explosives, and smoking must be strictly prohibited in the immediate vicinity. No iron or steel tools must ever be used for the opening of explosive cases, or for charging holes with explosives. Nitro-glycerine explosives must never be exposed to the direct rays of the sun.

Obituary.

Mr. J. S. Fisher.

It is with much regret that the death is recorded of Mr. J. S. Fisher (Member), who died after an illness of over three months, on the 26th July. The deceased was born in the Isle of Man 42 years ago, and was educated at Victoria College, Douglas, and then served an apprenticeship in mechanical engineering at Barrow. Coming to the Transvaal in 1892, he commenced work on the mines in a subordinate position, and towards the end of the war (during which he served with Bethune's Mounted Infantry; medal and three clasps) he was appointed manager of the New Rietfontein Mine. In 1902 he joined the Mines Department as a Deputy Inspector of Mines, and in July, 1904, was promoted to the post of Inspector of Mines for the Johannesburg District, and had recently been transferred to the charge of the Germiston District. Mr. Fisher married in 1904, and leaves a widow and two children to whom the Society extends its condolence in their bereavement. Mr. Fisher was elected a member of the Society in September, 1902, and although not taking any active part in its discussions, on the several occasions in which his advice and assistance was sought by the Society's officials he was always most considerate, courteous and ready to help in every possible way.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Photochemistry of Sulphuric Acid.—"The authors have studied the formation of sulphur trioxide from sulphur dioxide and oxygen under the influence of the radiation from a quartz mercury lamp. Experiments made with a quartz reaction vessel mounted within a mercury lamp showed that the formation of sulphur trioxide proceeds fairly rapidly even at the ordinary temperature and more rapidly at higher temperatures. The equilibrium is quite distinct from that attained in daylight. With the mixture, $2SO_2 : O_2$, equilibrium is attained with a production of 65% of sulphur trioxide, and this equilibrium can also be attained starting from sulphur trioxide. At 160° C., with the gases confined in the reaction chamber, equilibrium was attained in one hour. In daylight at temperatures below 450° C. the equilibrium condition corresponds to practically 100% of sulphur trioxide. Again the light-equilibrium (attained on exposure to the radiation from the mercury lamp) is not displaced by rise of temperature up to 800° C., whereas the temperature- or dark-equilibrium (in daylight) is displaced considerably by rise of temperature above 450° C. With a mixture in the proportions, $SO_2 : O_2 = 1 : 13$, for example, at 800° C., the yield of sulphur trioxide is 80% on exposure to the rays from the mercury lamp, whereas in daylight, the yield is 44% of sulphur trioxide. The temperature-coefficient of the velocity of the photochemical reaction was found to be 1·2. The authors consider that a technical photochemical process for the production of sulphuric acid is quite feasible, since on using air in place of oxygen, no oxides of nitrogen are formed. Some experiments were made with a special mercury lamp composed essentially of a quartz tube, 115 cm. long and 1·8 cm. diam. This was enclosed in a tube of opaque, English quartz, 100 cm. long and 5 cm. diam., the annular space between the two serving as reaction-chamber. At 450 C. and with the gaseous mixture passing at the rate of 100-150 c.c. per minute, yields of 67·4, 70 8, 92, and 90% respectively of sulphur trioxide were obtained, with 0·78. 0.66, 8 7 and 9·3 mols. of oxygen per mol. of sulphur dioxide."—A. COEHN and H. BECKER. *Z. physik Chem.*, 1910, 70, 88-115. *Journal Society of Chemical Industry*, Mar. 15. 1910, pp. 269-270. (A. W.)

CARBON TETRACHLORIDE VAPOUR; ACTION ON MINERALS.—"The vapour of carbon tetrachloride acts on mineral metallic oxides, transforming them into chlorides, as readily as on artificial products, provided they are sufficiently finely divided. Silica is not attacked, anhydrous silicates are attacked in inverse proportion to their silica content, while hydrated silicates are totally transformed. The separation of the volatile chlorides, and the removal of the insoluble silica from the residue, makes this process of considerable value in mineral analysis, and in the detection of oxide in non-oxide minerals (*e.g.* molybdenite)"—P. CAMBOULIVES. —*Comptes rend*, 1910, 150, 221-223. *Journal Society of Chemical Industry*, Mar. 15, 1910, p. 275. (A. W.)

AUSTRIAN STATE SALE OF RADIUM.—"The sale of radium has been entrusted to the 'Bergwerksproduktenverschliess-Direktion' in Vienna, under the control of the Austian Minister for Public Works. The radium is sold in the form of radium-barium chloride of three different grades, the price for each milligram of radium chloride, including the containing cell, being 400 Kronen (£16 13s. 4d.). It is packed in cylindrical cells of 21 mm. diam. and 9 mm. long, formed of nickel-plated brass. On the bottom of the cell a layer of lead is cast, in which is a square depression for the reception of the radium-barium chloride. The cell is closed by a mica plate held in position by the screwed-on upper part of the casing. On the bottom of the casing is an official stamp (an eagle) and the series number Radium cells sealed with lead, and stamped on the soldered part, are also supplied. The cells are packed in cotton-wool and sheet-lead in a small box, together with a certificate bearing the number of the cell, and the weight and radium-content of the preparation. The boxes are sealed with strips bearing the numbers of the cells, and are dispatched by post as registered packets at the cost and risk of the purchaser."—*Z. Allgem Oesterr. Apoth.-Ver.*, 1910, 48, 71. *Journal Society of Chemical Industry*, xxix., 5, Mar. 15, 1910, p. 276. (J. A. W.)

CARBON SUBNITRIDE.—"By removing two molecules of water from the diamide of acetylene dicarboxylic acid, $CONH_2 \cdot C : C \cdot CONH_2$ a subnitride of carbon (dicyanacetylene or carbon cyanide, $N : C \cdot C : C \cdot C : N$) is obtained as a crystalline body, of m. pt. 21° and b. pt. 76° C., and having an odour resembling that of cyanogen. It is readily combustible and takes fire spontaneously at 130° C., burning with a purple flame. Its molecular refraction and dispersion are abnormally high, probably on account of the three triple linkages. The high vapour pressure and the phenomenon of spontaneous combustion render analysis and vapour density determinations difficult, but both have been made to agree with the formula, C_4N_2."—C. MOUREU and J. C. BONGRAND.—*Comptes rend*, 1910, 150, 225-227. *Journal Society of Chemical Industry*, xxix. 5, Mar. 15, 1910, p. 276. (J. A. W.)

NEW VOLUMETRIC METHODS OF DETERMINING ZINC AND LEAD.—"The author has investigated the application of potassium cyanide in the volumetric determination of lead and zinc. The estimation of zinc is similar to that of silver by Liebig's method, and depends on the formation of the soluble double cyanide, $K_2Zn(CN)_4$, which is decomposed immediately by excess of zinc ions, producing insoluble zinc cyanide: $K_2Zn(CN)_4 + Zn^{\cdot\cdot} = 2Zn(CN)_2 + 2K$: The end point is rendered much more definite by the presence of a small quantity of any ammonium salt (except the acetate). 10 or 20 c.c. of *N*/2 potassium cyanide are titrated with a 0·2—0·8% solution of the zinc salt (free from acid) till a permanent cloudiness remains after shaking. 1 c.c. of *N*/2 potassium cyanide = 0·008171 grm. of zinc. Lead cyanide is not soluble in excess of alkali cyanide, but it is readily soluble in acids. On adding excess of potassium cyanide to a lead solution, and filtering, the excess of potassium cyanide can be titrated with hydrochloric acid and methyl orange. 20 or 25 c.c. of *N*/2 alkali cyanide are treated with a known volume of lead salt solution (free from acid) in a 100 c.c. flask. After making up to the mark, allowing to stand for ten minutes, and filtering, the cyanide is titrated in 50 or 75 c.c. of the filtrate by means of *N*/4 or *N*/2 acid. The cyanide solutions are standardised by pure salts of lead or zinc, though the purest samples may be standardised simply by hydrochloric acid with methyl orange."—E. RUPP.—*Chem. Zeit.*, 1910, 34, 121. *Journal Society of Chemical Industry*, xxix. 5, March 15, 1910, p. 300. (J. A. W.)

POTASSIUM CYANIDE AS AN INSECTICIDE.—"An aqueous solution of potassium cyanide acts as a powerful insecticide when injected into the soil, by reacting on the calcium bicarbonate present and slowly liberating hydrogen cyanide. It possesses certain advantages over carbon disulphide, as it is slower and more thorough in its action, insects are not aware of its presence from its odour and therefore do not escape, and it has no effect on growing plants, even in strong doses, nor does it stop fermentative changes in the soil as carbon disulphide does."—T. MAMELLE.—*Comptes Rendus*, 1910, 150, 50. *Journal Franklin Institute*, April, 1910. (J. A. W.)

PROCEEDINGS OF THE INTERNATIONAL COMMISSION ON ATOMIC WEIGHTS.—"*Chlorine.*—By passing nitrosylchloride, NOCl, over silver to remove chlorine, then over heated copper to remove oxygen and finally over metallic calcium to absorb nitrogen, a full analysis of this compound was made. From the ratio or oxygen to chlorine it resulted that the At. wt. of chlorine is 35·468, and *nitrogen* is 14·006.

Carbon.—Baumé and Perrot have found from the density of methane that carbon = 12·004. From the density of toluol vapour, as determined by Ramsay and Steele, Leduc finds carbon = 12·003.

Iodine and Silver.—Baxter and Tilley have determined the ratio between iodine pentoxide and silver.

They conclude that silver lies between 107·847 and 107·850. The corresponding value for iodine is 126·891.

Phosphorus.—From the density of phosphoretted hydrogen, PH_3, Ter Gazarian finds the value of phosphorus = 30·906.

Arsenic.—Baxter and Coffin compared the ratios of silver arsenate with silver chloride and silver bromide respectively, by two methods. For the value 107·880 for silver, the final mean of arsenic was 74·957.

Chromium.—Baxter, Müller and Hinnes found by the analysis of silver chromate a mean value for chromium of 52·008, when silver is 108·88 Similar analyses of silver dichromate gave Baxter and Jesse for chromium the value 52·018. This gives chromium a mean value of 52·01.

Tellurium.—Lenher led the double potassium tellurium bromide over potassium chloride, and then heated it in a current of chlorine and lastly in hydrochloric acid. The mean value from 16 experiments gave Te = 127·55.

Mercury.—Easlay found the unusually high values of 200·48 and 200·62 for mercury; but as he is still continuing his investigation, it is not desirable to accept his figures until his work is completed.

Palladium.—Gutbier, Haas and Gebhardt analysed pallados-ammonium bromide and found as the most probable mean value Pd = 106·689.

Krypton and Xenon.—Moore isolated both gases in considerable quantities from the residues of liquid air. From the densities of the two gases he found Kr = 83·012 and Xe = 130·70."—F. W. CLARKE, and others.—*Zeit. für ang. Chem.*, xxiii., 1, 21. *Journal of the Franklin Institute*, May, 1910, p. 417. (J. A. W.)

SILICON CARBIDE, ARTIFICIAL GEMS.—"The theoretical reaction on which the manufacture of carborundum is usually based is $SiO_2 + 3C = SiC + 2CO$, and the following proportions by weight (affording a slight excess of carbon over that theoretically required) are used in practice: 35·1 per cent. ground coke, 54·4 sand, 7 sawdust, 3·5 common salt. The silicon carbide thus produced occurs in the form of opaque crystals of various colours, the black predominating. Mr. Frank J. Tone, superintendent of the Carborundum Company, has discovered another reaction $3\,Si + 2CO = 2SiC + SiO_2$. This is a gaseous reaction, gaseous silicon combining with carbon monoxide at a temperature below the temperature of dissociation of silicon carbide. The silicon carbide thus produced builds up in large crystals, which start upon crystals, initially formed by the direct union of silica and carbon of the charge, and by prolonging the operation of the furnace these crystals can be made to attain a large size. For this purpose the charge mixture should be constituted so as to afford spaces for the mingling and combining of the gases. This space is afforded by using sawdust in the charge mixture, preferably in a larger percentage than heretofore employed. Mr. Tone uses a carborundum furnace with a core of carbon, preferably in the form of carbon granules, free from contamination of metallic oxides (for instance, a carbon core taken from previous runs of a furnace). This he surrounds with a mixture of pure silica, petroleum coke (or purified ground coke), and sawdust. These are mixed in a proportion so that the silica will be somewhat in excess of the amount theoretically required by the formula

$$SiO_2 + 3C = SiC + 2CO.$$

These proportions by weight are attained by the following mixture: ground carbon, 30 per cent.; silica, 57 per cent.; sawdust, 9 per cent.; common salt, 4 per cent. The furnace is then started and the operation is carried on for such a length of time that the zone of carborundum crystals adjacent to the core is decomposed to a considerable thickness, preferably to an extent of 2 in. or 3 in. The decomposition of these crystals results in the liberation of silicon vapor, which combines with the carbon monoxide gases in the furnace and builds up large crystals of carborundum just outside the zone of graphite which is formed by the decomposition. The crystals thus produced are clear and transparent, being free from coloration, and can be made of a large size. It is important that the carbon of the charge should not be in excess, since otherwise the crystals produced are apt to be black. These crystals have a very high index of refraction (exceeding that of crown glass and the diamond) and possess double refraction and are stated to be of great value as gems, since they can be cut with regular facets and polished. By adding to the furnace charge a small amount (not more than 0·5 per cent.) of a coloring matter, like oxide of chromium or oxide of manganese, the crystals can be tinted without destroying their transparency." (949,386, Feb. 15, 1910; assigned to Carborundum Company.)—*Metallurgical and Chemical Engineering*, March, 1910. (J. A. W.)

CALCIUM SILICIDE FOR STEEL REFINING.—"The production of calcium silicide, which is now finding more and more extended use in steel works, is the object of a patent of Mr. Georges Strauss (assigned to Société Anonyme La Compagnie Génerale d'Electro-Chimie de Bozel, of Paris, France). By heating a mixture, in definite proportions, of lime, silica and carbon in an electric furnace, calcium silicide is obtained according to the equation

$$CaO + 2SiO_2 + 5C = CaSi_2 + 5CO.$$

But the direct formation of silicates is a troublesome element in this process. To avoid this, Strauss uses calcium carbide (instead of lime) and metallic silicon (instead of silica) as starting materials. He gives the following equations for the reactions:

$$CaC_2 + 2SiO_2 + 2C = CaSi_2 + 4CO.$$
$$2Si + CaO + C = CaSi_2 + CO.$$

Experiments made with calcium silicide have shown that this metallic silicide is capable of replacing with advantage the aluminium usually employed in steel works for the purpose of preventing the formation of blow holes in steel ingots. The advantages ensuing from such application are, in addition to the economy secured by the very low price of calcium silicide, very great rapidity of reaction and the suppression of the undesirable 'shortness' which aluminium communicates to ingots of steel." (948,190, Feb. 1, 1910.) *Metallurgical and Chemical Engineering*, Vol. viii., No. 3, March, 1910, p. 150. (J. A. W.)

PREPARATION OF HYDROCYANIC ACID. — "By passing a mixture of air and steam over coal heated, in a generator, to a red heat one obtains a gas consisting of nitrogen, hydrogen, carbon dioxide and carbon monoxide. If one uses for 1 Kg of coal 3 cbm air and 0·6 kg steam one obtains a gas most suitable for this process. This gas consists of about 51·8% N, 15·3% H, 27·6% CO, 3·9% CO_2 and 1·4% CH_4. This mixture is then passed through a pipe into an electric oven filled with coke. By passing the electric current between two carbon electrodes the coke is heated to such a temperature that the C, N and H combine to form hydro-cyanic acid. The yield at a temperature of about 3500°C is 35%. The gases are then quickly drawn off, cooled and passed into a suitable absorption apparatus, in which the HCN is retained, while the free gases are returned up the generator." — O. Dieffenbach and W. Moldenhauer.—*Chemiker Zeitung*, No. 55, 1910. (T. D.)

The use of Phosporus Solutions in Gas Analysis.—"The absorption of oxygen with phosphorus has certain drawbacks. In pure oxygen the phosphorus does not oxidise; the presence of traces of hydrocarbons, the vapours of organic substances and certain inorganic gases also hinder the oxidation. However, if a solution of phosporus is used the absorption is complete and easy to regulate. A 1%-1½% solution of phosphorus in purified castor-oil was used and prepared in the following way: 3 grs. well dried phosphorus were brought into a 250 cc. flask containing 230 cc. castor oil and heated to a temperature of 200°C in an oil bath. The hot flask was then removed, tightly stoppered, dried outside and thoroughly shaken until complete solution took place. The analysis of dilute oxygen is carried out in the ordinary way with a Hempel pipette. If the oxygen is pure or mixed with hydrocarbons the absorption takes place by heating the solution with an electric current. Points to be noticed :—

1. The wire for heating must not come in contact with the gas.
2. To prevent electrolysis an excess of water in the pipette must be avoided.
3. The phosphorus must not be oxidised to ignition point as, in this event, the oil burns and forms CO_2; this objection can be overcome by not heating too strongly and by using a not too concentrated solution. A solution of 0·8 grs. in 100 grs. castor oil will be found useful."—

M. Centnerszwer. — *Chemiker Zeitung*, No. 56, 1910. (T. D.)

Solubility of Gold in Nitric Acid.—"The author has conclusively proved that nitric acid used for the separation of gold dissolves some of the gold. He carried out very carefully an experiment in which he treated 20 grams of fine gold for 2 hours in hot nitric acid of 1·42 specific gravity. After filtration the solution contained 660 milligrams gold per litre." — Fred. P. Dewey (transactions of American Chemical Society, Boston 27-3112, 1909.)—*Zeitschrift fur Angewante Chemie*, May 27, 1910. (T. D.)

METALLURGY.

Steel Industry in the Transvaal.—"The establishment of an iron and steel industry in the Transvaal would be welcomed by mining engineers because it would tend to reduce the cost of mining, not only by a possible diminution in the price of the steel used underground, but by improvement in quality. At present the quantity of drill-steel annually imported into the Transvaal is from 5,000 to 5,500 tons, which is sold at a price varying from £30 to £35 per ton. This is shipped principally from England, but considerable competition from American, German, Swedish, and Belgian manufacturers is felt. As a result, prices are lowered and inferior qualities of steel are put on the market.

The best steel for drilling was formerly made in the crucible, but gradually this has been replaced by Swedish bessemer, English bessemer, and even open-hearth steel. Swedish bessemer is steel made in the converter from Swedish raw materials, which are remarkable for their low sulphur and phosphorus contents. A similar material is made in England from equally pure materials, but owing to the enhanced value of any steel to which the mystic word 'Swedish' is attached, this epithet is retained in the case of many steels that are not derived in any sense from the Scandinavian peninsula. The use of Swedish raw material in making high-class steel is generally supposed to give certain definite good qualities and 'body.' What 'body' is has not been accurately determined, and the cause is equally obscure. The Sheffield steel-maker of the old school attaches the utmost importance to this mysterious property, which he attributes to his Swedish ingredients, but it is probable that he under-estimates the value of his long experience in the melting and subsequent treatment, often the result of several generations of careful observation, and that it is to this, and this alone, that the property of 'body' is due, provided suitable raw material be used.

Steel, equal to the best Sheffield bars, could be made in the Transvaal by the electric furnace. This produces an intense heat, permitting of the most basic slags; as a result, common scrap, such as is found in large quantities in South Africa, can be melted and refined to a steel of crucible quality. The establishment of this industry would not only provide a steady market for scrap-iron and steel, but would also enable the mining and railway companies to obtain first-class steel castings at short notice. The success of this industry appears to depend on the use of an electric furnace, as this is the only means by which common scrap can be economically refined to produce high-class steel. In the case of any other process about 30% of pig-iron must be imported or made, otherwise complete refining cannot be effected. In Europe and America there are several electric furnaces in use. It is interesting to note that a company controlling the process most generally adopted by American and English firms has already sent engineers to South Africa with a view to the establishment of this industry, while the Transvaal Government has sent representatives to visit the principal works in Europe, where electric steel-making is established.

The mining companies will profit in two ways: by obtaining better steel and by having a ready market for their scrap steel, the price of which during recent years has only been from 5s. to 7s. 6d. per ton, while the quantity exported during the last two or three years has amounted to 40,000 tons.

The possibility of starting blast-furnaces for the manufacture of pig-iron is being investigated carefully, and also the question of export. It will be impossible to export pig-iron to Europe under present conditions of labour and supplies. The export trade would have to compete with such producers as the Dominion Iron & Steel Co. or the North of England blast-furnaces, where excellent ore and coal are available on the sea-coast, together with efficient labour. The deposits of iron ore in the Transvaal are undeveloped, and though the outcrops are large, the composition of the ore in depth is unknown, coal of suitable coking qualities is not widely distributed, while the labour and transport conditions alone present serious difficulties for the successful establishment of this industry.

There remains then only the possibility of pig-iron manufacture for local use. Rail-mills cannot be worked economically on small outputs. The market for foundry-iron is comparatively small and largely fed by scrap cast-iron, which sells at about 80s. per ton. The only considerable items on the import list of iron and steel are large quantities of corrugated sheet-steel. The possibility of erecting a small

Swedish blast-furnace to supply a small mill to make sheets and rails is under consideration. These might be made economicaly, if a thorough investigation proves all the conditions favourable, but at present the only industry that can be deemed profitable is the refining of scrap-iron and steel in the electric furnace.

South Africa is essentially a market for high-grade steel. Of the iron and steel imports, which have a value of about a million sterling, there are 5,500 tons of mining bars and an equal tonnage of shoes and dies, of a value of over £250,000. Small railway-castings, axles, and crusher-jaws could all be made profitably, provided a suitable quantity of iron and steel scrap be available at a reasonable price. This industry should prove beneficial to the mining community, the power-supply companies, and the promoters; incidentally improving the quality of steel used on the Rand and preventing the wasteful export of scrap, which is part of the undeveloped wealth of the country and should not be allowed to go elsewhere.

Extreme care must be exercised in the choice of the locality in which the first blast-furnace is erected. Coking coal exists in the Middelburg district and also iron ore, both titaniferous and good hematite. It remains to consider the effect of the proposed improvements intended to make Delagoa Bay the principal port of South Africa and the possible influx of Indian labour into the adjacent Portuguese territory. Africa is in a stage of transition. United South Africa has internal and international questions of the utmost importance before her, and not the least important is the future of Delagoa Bay and the surrounding territory. Steel-works designed to make a small tonnage of high-class material require an expenditure of less than £100,000, and Johannesbug alone provides a market, but blast-furnaces and rail-mills involve some hundreds of thousands of pounds, as well as questions of wide and international importance. After the smaller industry is firmly established, it will be quite soon enough to consider possible expansion.

The subject is of the greatest practical importance to the industrial stability of the Transvaal. Iron is the bread of commerce and steel is the staff of industry."—Donald F. Campbell.—*Mining Magazine*, Jan., 1910, p. 55. (H. A. W.)

Gold Refining.—"In German patent 207,555, Sept. 22, 1908, of the *Norddeutsche Affinerie A. G.*, a modification of the original Wohlwill process of gold refining is given, which appears to broaden greatly its scope and applicability. If gold, rich in silver, is refined in a solution of $AuCl_3$, containing HCl or other chlorides, a layer of AgCl forms on the anodes which must be removed from time to time. In practice this has been found necessary whenever the silver content in the anode is more that 6%. If the layer of AgCl is not removed from the anode, chlorine is set free and this will take place at an earlier stage of the process, the higher the current density used and the higher the silver content of the anode. To overcome this a pulsating current instead of a constant direct current is used, which is best obtained by connecting a direct current dynamo in series with an alternating current dynamo. With raw gold containing 10% silver an anodic current density of not more than 750 amps. per sq. meter must be used in the original process, but with an alternating current about 1·1 times as strong as the direct current, 1,250 amps. per sq. meter can be used without any necessity for scraping off the silver chloride. Another advantage is that in the original process gold passed into the slime to the extent of about 10% and had to be recovered by chemical means, whereas with the pulsating current only a little gold passes in at the beginning of the process and afterwards none at all. The slime, therefore, consists essentially of AgCl and its spongy structure causes it to drop off by itself."—*Metallurgical and Chemical Engineering*, Vol. viii., No. 2, p. 82, Feb., 1910. (J. A. W.)

Metallurgical Trials.—"From Johannesburg we learn that systematic milling tests of great interest are being made by Mr. F. L. Bosqui, in behalf of H. Eckstein & Co. Twenty stamps, a tube mill, the requisite classifiers, and amalgamation plates have been set aside at the Jumpers Deep mill for this purpose. A small zinc-dust precipitation press, with a capacity of 60 tons of solution per day, is on its way to supplement this trial plant. It is hoped within six months to ascertain (1) the economic limit to fine grinding in tube mills, (2) how much of the minus 150 sand now treated by percolation can be treated advantageously—while in mixture with the slime—by agitation, followed by vacuum filtration, (3) whether Brown agitators and vacuum filtration are preferable to decantation, (4) how the work of comminuting the ore should be apportioned between stamps and tube mills, and (5) whether zinc-dust is better than zinc-shaving as a precipitant. We may add that a 500-ton Butters plant is being installed at the Crown Reef, and should be ready in June. Thus several points affecting the development of local metallurgical practice will be decided at an early date."—*The Mining Magazine*, Feb., 1910, p. 87. (W. R. D.)

Measurement of Pulp and Tailing, III.—"*Tonnage Crushed in Mills.*—This is often estimated merely by count of cars dumped per week or month, and gives at best only a rough approximation to the truth. If a fair proportion of the cars are weighed daily, if moisture samples are taken when the material is not quite uniform in this respect, and if a correction is made periodically for differences in the volume of ore in the storage-bins, the results are reliable within about 2%. A slightly higher degree of accuracy is claimed for the continuous-weighing devices, which are operated by the pressure of a belt-conveyor, and the accuracy of which can be checked by occasionally comparing the record obtained with a cartload of ore weighed by reliable scales. When using these devices the corrections for moisture and fluctuations in mill-storage must not be neglected. When 'mill time' is used in reckoning tonnage, the time lost in repairs, etc., must be carefully recorded, and the results are reliable only when based on past experience of the average duty performed with a particular ore.

If the density of the dry material is nearly constant, and frequent measurements are desirable, it is advisable to draw up a table covering the readings usually observed: at the head of the vertical columns is given either the gravity or the number of pounds of pulp per cubic foot, the seconds per cubic foot at the left of the horizontal rows: the tons of solid per day are entered at the corresponding intersections, as in Tables B, C, and D.

Even if the gravity methods were less accurate as regards a single observation, they would give more accurate results in many cases than the first method described, provided two or three were taken at intervals during the day and averaged; the time occupied in drying usually precludes repeated deter-

TABLE A.—Tons of Solid Slime in Agitator.
Marysville, Montana.
Density of Dry Slime = 2·60.

Average level of pulp is read with respect to datum mark on staves, specific gravity of pulp is determined by weighing 1,000 c.c.; total tons dry slime in agitator read directly from table, interpolating.

Upper datum-mark = 420 fluid tons. Lower datum-mark = 310 fluid tons.
One inch = 3 fluid tons.

Specific Gravity of Pulp.	Water Ratio.	Solid Factor for 100 tons.	TONNAGE OF DRY SLIME. When capacity of Vat to same level in fluid-tons 420	410	400	390	380	370	360	350	340	330	320	310	300
1·268	1·9	43·8	184	179	175	171	166	162	157	153	149	144	140	136	131
1·257	2·0	42.0	176	172	168	163	159	155	151	147	143	138	134	130	126
1·247	2·1	40·3	169	165	161	157	153	149	145	141	137	133	129	125	120
1·238	2·2	38·7	163	139	155	151	147	143	139	136	131	127	123	120	116
1·229	2·3	37·3	157	154	150	146	142	138	134	130	127	123	119	115	112
1·220	2·4	36·0	151	147	144	140	137	133	129	126	122	119	115	112	108
1·213	2·5	34·7	145	142	139	135	132	128	125	121	118	114	111	107	104
1·206	2·6	33·5	140	137	134	130	127	124	130	117	114	110	107	103	100
1·200	2·7	32·4	136	132	129	126	123	119	116	113	110	106	103	100	97
1·193	2·8	31·4	132	128	125	122	119	116	113	110	106	103	100	97	94
1·187	2·9	30·5	128	125	122	119	116	112	109	106	103	100	97		
1·181	3·0	29·6	124	121	118	115	112	109	106	103	100	97			

TABLE B.—Tonnage table for Five Stamps.
Density of Ore = 3·0.

Pulp is run into cylinder holding 15 U.S. gallons (2 cub. ft.) and weighed on platform scale; time for 2 cub. ft. is taken with stop-watch. Value in last column divided by seconds = tons per stamp per day.

Net weight of 2 cub. ft. pulp lb. avp.	Specific Gravity of Pulp.	Water Ratio.	Tons per Stamp per 24 hours, if 2 cub. ft. caught in one second.
131·50	1·052	12·40	84·3
0·675	1·053	12 22	86·0
0·75	1·054	12·00	87·5
0·875	1·055	11·79	89·1
132·00	1·056	11·57	90·8
0·125	1·057	11·36	92·4
0·25	1·058	11·155	94·1
0·375	1·059	10·96	95·7
132·50	1·060	10·775	97·2
0 675	1 061	10·59	98·8
0·75	1·062	10·42	100·4
0·875	1·063	10·25	102·0

TABLE VII.—Sluicing Data.
Based on Wet Slime-cake or Residues containing 33·33% water (M = ·50 C = 66·66).

	Density of Dry Solid.	Specific Gravity of Final Pulp = p = Hydrometer Reading.*** 1·01	1·03	1·05	1·07	1·10	11·15	1·20	1·25	1·30	1·35	1·40	1·45	1·50
Values of Q = tons water added per ton of dry solid in wet cake or residue sluiced = R – M	2·4	57·4	18·5	10·75	7·41	4·92	2·98	2·0	1·417	1·026	0·750	0·542	0·380	0·250
	2·6	60·7	19·6	11·4	7·90	5·27	3·22	2·19	1·575	1·166	0·873	0·654	0·483	·346
	2·8	63·5	20·6	12·0	8·33	5·57	3·42	2·36	1·715	1·286	0·98	0·75	0·571	·429
	3·0	65·8	21·4	12·5	8·68	5·83	3·61	2·50	1 833	1·389	1·072	0·833	0·648	·500
Tons water added per ton of wet material sluiced $T = \frac{Q\ C}{100}$	2·4	38 3	12·3	7·17	4·94	3·28	1·99	1·33	0·945	0·684	0·500	0·361	0·253	·167
	2·6	40·5	13·0	7·60	5·27	3·51	2·15	1·46	1·05	0 778	0·582	0·435	0 322	·231
	2 8	42·3	13·7	8·0	5·55	3·71	2·28	1·57	1·14	0·858	0·658	0·50	0·381	·286
	3·0	43·9	14·3	8 33	5·79	3·89	2·41	1·67	1 22	0·926	0·715	0 555	0·432	·333
Tons water added per cubic foot of wet material sluiced**	2·4	1·96	·630	·351	·252	·168	·102	·068	·0485	·0350	·0256	·0185	·0130	·0086
	2·6	2·146	·689	·403	·279	·186	·114	·077	·0557	·0412	·0304	·0231	·0171	·0122
	2·8	2·315	·750	·438	·305	·203	·125	·086	·0624	·0469	·0357	·0273	·0208	·0156
	3·0	2·470	·805	·470	·325	·219	·136	·094	·0686	·0522	·0402	·0312	·0243	·0167
Efficiency of sluicing = E = tons dry solid moved per 100 tons of added water R – M100	2·4	1·74	5·40	9·3	13·5	20·3	33·6	50·0	70·5	98·3	133·	184·	263·	400·
	2·6	1·65	5·10	8·77	12·7	19·0	31·0	45·7	63·5	85·7	114·	153·	207·	289·
	2·8	1·575	4·78	8·33	11·6	18·0	29·2	42·4	58·3	77·7	102·	133·	175·	233·
	3·0	1·52	4·67	8·0	10·7	17·1	27 7	40··	54·5	72·0	93·	120·	154·	200·

** Assuming material compacted so that interstices are full of water.

*** p = pounds per cubic foot × 0·016.

TABLE VIII.

Specific Gravity and Weight per cubic foot of Wet Cake or Residue. Moisture ranging from 25 to 40%.

Data for constructing tables similar to Table V. for material differing in moisture content.

Material compact so that interstices are full of water.

Moisture in Cake or Residue Sluiced %	Value of **C** % Solid.	Value of **M** Water Ratio.	Density of Dry Solid **d**	Specific Gravity of Wet Material.	Weight of Wet Cake per cubic foot lbs.	Weight of Dry Solid per cub. ft. of Wet Cake lbs.
40·0	60·0	0·6667	2·4	1·538	96·15	57·69
			2·6	1·585	99·08	59·45
			2·8	1·628	101·74	61 05
			3·0	1·667	404·17	62·50
35·0	65·0	0·5384	2·4	1·610	100·67	65·43
			2·6	1·667	104·17	67·71
			2·8	1·718	107·36	70·08
			3·0	1·765	110·39	71·70
33·33	66·67	0·50	2·4	1·636	102·27	68·18
			2·6	1·696	106·0	70·66
			2·8	1·750	109·38	72·92
			3·0	1·80	112·50	75·0
30·0	70·0	0·4286	2·4	1·690	105·63	73·94
			2·6	1·756	109·69	76.83
			2·8	1·820	113·75	79·62
			3·0	1·875	117·19	82·03
25·0	75·0	0·3333	2·4	1·778	111·11	83·33
			2·6	1·857	116·06	87·05
			2·8	1·931	120·7	90·53
			3·0	2·00	125·0	93·75

minations by Method No. 1. It must be remembered that any flow-determination is usually instantaneous, or at least represents a period not exceeding a minute or two, which may be far from the average flow of the entire day when the various adjustments of a mill or concentrator are considered. Several rapid and fairly approximate determinations, at more or or less regular intervals during a day and averaged, are preferable to a single observation made with all possible precautions and carried out with scientific accuracy, if the sample taken in the latter case is subject to momentary variations.

Mill Tables.—In the daily control of mill-work, particularly that of an experimental character, it is desirable to make the labour of computation as light as possible, so that results for rate of crushing or the contents of a vat can be read off rapidly after making the required observations by weighing pulp, etc For this reason it is convenient to have a suitable table prepared to meet any special case, and an hour or two spent in making such a table will be saved many times over in subsequent work when systematic experimentation is carried on."—W. J. SHARWOOD. —*Mining Magazine*, Jan., 1910, p. 45. (H. A. W.)

SORTING: AN ECONOMIC PROBLEM ON THE RAND.—"Sorting on the Rand is an operation generally understood to mean the removal from the gold-bearing ore of all barren rock, usually called waste rock or, in short, 'waste.' Wager Bradford, in his excellent paper read before this Association, dealt with the question of sorting very exhaustively, but he put particular stress on sorting underground, not, as generally understood in late years, that is by erecting sorting plants underground, but as meaning careful mining, and, if the waste has unfortunately been mined, its elimination in the stope before any further expense is incurred in handling. As an example he gives the result of milling waste rock at the Rose Deep mine, where the average value was 1·62 dwts. as against ·609 dwts., the returned value of waste rock for nine months previously.

It appears to me that in those days (1903) with high working costs all round, and with the limit of payability somewhere in the neighbourhood of 8 dwts., they did not sufficiently discriminate between barren rock and low grade ore. This is, to my mind, the fundamental error made by Wager Bradford. Obviously, waste is rock containing no gold whatsoever, while rock going 1·5 dwts. is gold-bearing ore and must be regarded as such, and its unpayability is only a matter of degree and not of fact What was unpayable in 1903 is highly payable to-day when mines make good profits on ore going 4 dwts. and under. Assuming waste rock assaying 1·5 dwts., and also that by closer sorting half of the waste discarded could be retained, and would probably give a screen sample of 2·5 dwts., say 10s. per ton. All this waste is brought to the surface, and if it is not sent to the mill it probably costs 3d. per ton to send to the waste dump. If it were milled, the cost of sorting per ton would not exceed 1s. 6d.: the cost of milling and cyaniding can now be brought down to 3s., so that with an extraction of 90%, 9s. can be obtained for an expenditure of 4s. 6d., or a very handsome profit.

With waste rock assaying originally only 1 dwt., and more intensive sorting as recommended, the saving would be 1s. 6d. after allowing 1s. 6d. for sorting and 3s. for milling and cyaniding, the value after sorting being 1·5 dwts.

The question of sorting therefore resolves itself into:—

(1) Is it practically possible to sort out barren rock only; and

(2) If so, can it be done at a profit.

In order to decide the first point it must be known whether, given efficient labour and mechanical appliances, it is possible to distinguish barren rock from gold-bearing ore. I am not competent to decide this, and shall leave it to mining engineers, metallurgists and geologists.

Once the first point is satisfactorily decided, the second point becomes merely a mathematical investigation, which, taking into account all known factors, determines the most profitable life of the mine with the working costs and capital expenditure on the basis of the altered conditions, as was so ably done by Ross E. Browne and R. N. Kotzé before this Association.

It can be fairly said that sorting can be divided under two main headings covering operations entirely distinct, and each requiring special considerations, as follows:

1. Sorting as part of the mining operations, comprising (a) careful sampling of the rock *in situ* with a view of only breaking gold-bearing ore; and (b) eliminating waste rock in the stopes before any expenditure is incurred in handling same.
2. Sorting as generally understood, which comprises the picking out of the waste after the total ore has been prepared by preliminary breaking and washing to facilitate the work and to render the labour employed efficient.

The work comprised under (1) raises many questions, and as it is outside the province of the writer, he will leave it in the able hands of the mining engineers for exhaustive treatment. However, the question of sorting, as expressed under (2), is a purely mechanical one, and with your permission the writer proposes to make a few remarks dealing with some features of this work. Sorting operations may be conducted underground and on the surface. Many references to underground sorting have been made, but they generally referred to the work which the writer regards as part of the mining operations; and it remained for W. Calder to seriously propose abandoning all sorting and preliminary breaking on the surface in order to do the work underground, presumably at a saving in working costs if not in capital outlay. Calder says that the chief objection hitherto has been the cost of power. This is new to the writer, as the power requirements are generally very small if compared with those of the whole reduction plant: and they certainly do not exceed 5% of the total, leaving out the power used for hoisting rock and for compressing air. If the cost of power in the reduction plant is approximately 1·0s. per ton milled, the cost of the power required in the sorting and breaking plant would be ·05s. per ton milled. To seriously claim that work underground can be more efficiently done than in the light of day the writer cannot understand, and I have no doubt the packs in stopes resulting from sorting underground will in time become very valuable gold mines on their own account. The disadvantages attaching to sorting underground are obvious and need reiterating only to be appreciated. They are:—

(1) Increased capital cost;
(2) Higher operating costs.

The necessity of keeping an even grade requires work to be done on many levels, and to do sorting underground under these conditions would require a sorting plant at each level, say from 6 to 10 and more. The cost of these sorting plants, taking into account the greatly increased size of the excavations and the fact that numerous small sorting belts and breakers would be required in lieu of the one large belt and breaker on the surface, would be from 60% to 100%, more than for a plant at the surface. Anyone interested can easily make the necessary estimate. Operating charges will be higher than on the surface for several reasons.

The writer claims that there is no case for sorting underground, as none of the reasons advanced will bear a close examination, and sorting on the surface, having established its claim, it is now only a question of how to do the work economically. Before dealing with this point in detail it is as well to point out that although sorting and breaking are two distinct operations, they overlap to a certain extent, and for that reason are generally considered as one. The arrangements made for doing the sorting and breaking on the surface vary considerably on the different mines on the Rand; and while on one mine a very elaborate plant can be found, the adjoining one has only the veriest fragment of a sorting and breaking plant, showing conclusively that there is no general agreement in respect of the plant required. Before the waste can be picked from the ore, the whole must pass through a certain preliminary preparation such as screening, coarse breaking, and washing. To permit of efficient work on the part of the operators, it is necessary to so prepare the ore that a mistake cannot possibly be made; that is to say the waste should be clearly seen and small pieces of ore adhering to large pieces of waste should be detected. The work of picking out waste rock may be done on a sorting table or on a sorting belt, the old-fashioned sorting floor being practically defunct and quite useless with the large quantities of ore now handled.

A complete sorting station contains therefore as a rule:—

(a) Screening plant.
(b) Coarse breaking plant.
(c) Washing plant.
(d) Sorting plant proper, and
(e) Fine breaking plant.

The sorting and breaking plant may be erected close to the reduction plant, providing a central plant for dealing with all the ore handled and the advantage of this arrangement is that the work of sorting and breaking is performed under the most favourable conditions as regards—

(a) Supervision;
(b) Concentration of work;
(c) Capital outlay:

The drawbacks are:—

(a) Unnecessary hauling of waste rock on the surface;
(b) Re-elevating of all the ore at the central sorting plant;
(c) Risk of a complete stoppage of work due to fire or other serious breakdown.

As against the central sorting and breaking station there can be a separate station at each shaft head, as is the case on many mines. The advantages are:

(a) No unnecessary hauling of waste rock;
(b) The necessary elevating of the ore is done in the headgear;
(c) Damage by fire or a serious breakdown in one of the stations will disable only a portion of the plant.

The disadvantages are that the supervision is more difficult; the capital outlay is greater; and the benefits that could be derived from concentration of the work cannot be obtained.

Dealing now with the various appliances used, there is first of all the question of screening. Screening is done in one or two stages, and in the former case the appliance generally used is the grizzley, while with double screening the operation may be performed on two grizzleys, or by a combination of grizzley and screening trommel. The first, and in many plants the only, screening is performed in the head frame by a grizzley in front of the shaftways, and set at an angle of 38 degrees from the horizontal. The spacing of the grizzley bars is arranged to give a clear opening of $1\frac{3}{4}$ in. The grizzley is too well-known to require describing, and it may be sufficient to state that there are two ways of holding the bars, the old method holding the bars by means of tie rods and distance pieces, and a method of holding each bar independently. The latter method has the advantage that each bar can be taken out and replaced when worn, with a minimum of labour; while in the former case the whole set has to be lifted by block and tackle, requiring much time and labour. The length of the grizzley depends on the minimum area required to obtain efficient screening under given conditions, but headgear grizzleys should not be less than 14 ft. Stationary grizzleys are cheap to maintain, and for that reason the use of shaking grizzleys has not become general.

Screening trommels for single stage screening can be only used if the supply of the ore is more or less continuous and not intermittent, as in the case of the ore supply coming direct from the shaftways.

The coarse breaker is not found in all plants, but it serves to relieve the fine breaker of some of its work, and also assists in sorting as after coarse breaking a piece of ore previously adhering to a large piece of waste rock is frequently broken off, thus permitting of discarding the waste rock or at least the greater part of same. Unfortunately, the coarse breaker also produces a large percentage of 'fine,' and as this consists of both waste rock and ore, which is not afterwards subjected to sorting, the usefulness of the coarse breaker is greatly reduced, and may be entirely negatived. It is obvious that in order to obtain the full benefit from a coarse breaker all the material passing through should be subjected to a close examination, with the intention of removing all waste rock. The washing of the ore may be done on the sorting belt or table, or by a separate appliance, for which a trommel is generally used. Washing on the sorting belt, although not so effective as in a trommel, is nevertheless fairly efficient, and for that reason the washing trommel is not always used. The portion of the sorting belt on which the washing is done should have a steep grade, say, up to 18°, so as to permit of the water and slime flowing back. The amount of water required varies with the condition of the ore, but should not exceed from 20 to 40 gallons per ton of ore passing over the sorting appliances. The water and slime is generally collected in two settling vats, the first of which can be comparatively small, as the sand will settle freely, but the second or slime vat should have a large area to permit of settling all slime prior to returning the water for re-use.

The actual picking out of the waste rock is done on a sorting belt or a sorting table. The sorting table usually consists of a rim of from 20 to 30 ft. diameter, and from 30 in. to 42 in. wide, on which the ore is spread out, so that the sorting hands can see and remove the waste. The velocity of the rim should not exceed 30 ft. per minute. The ore left on the table is scraped off by a plough, rigidly held against lateral movement, but suspended in guides and well balanced so that it can follow the movements of the table. The sorting belt is an enlarged conveyor belt, travelling a speed not exceeding 30 ft. per minute, and should be arranged horizontally or, if required, at angle not exceeding 10° from the horizontal. The sorting belt has the advantage over the sorting table that it serves not only for sorting, but that it can also be made to elevate the ore, a point of great use in many plants. It has also been found that the maintenance on the sorting belt is considerably less that on the sorting table. I do not propose to give details of either table or belt, as they are too well known to require description. The sorting capacity of either table or belt depends upon the space available for sorting hands, and it may be taken that each man requires 3 ft. of length of belt or rim of table, and that he can pick from 2 to 3 tons of waste per shift of 10 hours.

The carrying capacity of sorting table and belt can be taken approximately as in the table hereunder, the assumed speed of belt or table being 30 ft. per minute :—

	Carrying capacity of	
Width of sorting surfaces.	Sorting tables. Tons per hour.	Sorting belts. Tons per hour.
30 in.	60	54
36 in.	72	66
42 in.	84	78
48 in.	96	90

Either appliance can carry considerably more, but only at the expense of efficient sorting. There is no limit to the diameter of sorting tables except constructional reasons ; whilst the length of the sorting belt is determined by the tensile strength of the material of the belt. However, since all the waste on either table or belt of a given width can be picked off in a certain distance, it is useless to make the table larger or the belt longer. The waste rock obtained from the sorting appliance can be disposed of in various ways, conveyor belts, trucks, and other means being in use on the Rand.

The fine breaker is usually of the gyratory type, and it is too well known to require a description. One point worth mentioning is that breakers of any type require a very substantial foundation, and if they cannot be arranged on a concrete foundation near the ground level, a very heavy structure, which should be of steel, will be needed.

There can be no doubt that the question of sorting has, in spite of all the papers written on the subject, not yet received the attention it deserves. I am unfortunately unable to go more fully into this matter at the present time, but I would like to throw out the suggestion that the Mines Trial Committee consider the advisability of making tests for the purpose of bringing the solution somewhat nearer than it is to-day. Meanwhile I would like to point to the following facts :—

According to Ross E. Browne, the cost of sorting per ton of rock sorted out was 1·435 shillings for the average mine of the Rand Mines Group, say in 1903, and it can safely be assumed that to-day, with greater efficiency all round, the cost per ton sorted out would not exceed 1s. The cheapest cost of milling and cyaniding a ton of rock, ore, or waste, is about 3s. ; therefore, every ton of waste rock not sent to the mill results in a saving of 2s. It is common knowledge that a good deal of waste is milled every day, and it is also commonly believed that the waste rock dumps contain a good deal of ore, or in other words, the sorting as at present carried out is inefficient. But how can this be altered ? I would suggest as an experiment—

(a) That the design of the sorting and breaking plant be so altered that all rock, waste and ore, is open to close scrutiny.

(b) That only people with good eyesight and a fair amount of sense be employed in the sorting plant, instead of the usual collection of piccannins, cripples, and other nondescripts.

(c) That a close and constant supervision by qualified men be exercised.

(d) That the management of the sorting plant be under an official not concerned with the tonnage mined, and who should be paid a bonus depending on the difference between the value of the screen sample and the value of the waste dump.

In order to meet these conditions a sorting and breaking plant as sketched might be useful. In this plant all screening as at present in vogue will be discarded, and instead the ore from the shaft will fall directly into the coarse ore bin (A), from which it will pass to washing trommels (B), provided with holes not larger than $1\frac{1}{2}$ in. Only the very finest material will fall through, and the balance will be fed to two sorting belts (C) of the required width and length, and moving at a speed not exceeding 20 ft. per minute. The sorting hands will be placed, as at present, on both sides of the sorting belts, and they will pick out, those on the inside, waste rock and clean ore, the former to be dropped through the floor to waste rock conveyors (G), and the ore to be placed on the conveyor (D). The picking hands on the outside of the sorting belts will pick out waste, and such waste to which small pieces of ore adhere, the former to be dropped on the waste rock conveyors (G), and the latter on the conveyors for waste and ore (E). The clean ore is conveyed direct to the fine breakers (K): the waste rock to the waste bin (M), and the mixed ore to the coarse breakers (F). After passing the coarse breaker the ore is again passed through washing trommels (B), and from there passes again to sorting belts (H) (called second stage belts) where further waste can be removed, to be carried by the waste rock belts (I) to the waste bin.

The underlying idea is that by at once removing the ore and waste that can be readily detected, the chances of efficient sorting are greatly increased, and should be inversely proportional to the reduced quantity of material on the sorting belts.

The sorting belts have to be so long and must run at so low a speed that all the material reaching the end can safely be classed as ore to be passed through the fine breakers It is assumed that only by giving the sorting hands ample time to see the ore, and ready means for sorting same into three classes, can the work be done in an efficient and satisfactory manner. Naturally many variations in the arrangement of the plant are possible, depending on the method of working proposed. It is claimed that in a plant as described, or one equally efficient, it will be possible to remove practically all the waste without risking any loss of ore due to faulty arrangements. It should be possible to keep the value of the waste rock at or near ·1 dwt. per ton.

The cost of this type of plant would not be much greater than one of the old type, as the breaking plant remains the same, and only the sorting plant is amplified. The power demand also should not increase by more than 30 per cent., but since the power requirements of the sorting and breaking plant are only about 5 per cent. of those of the reduction plant, this point does not carry much weight.

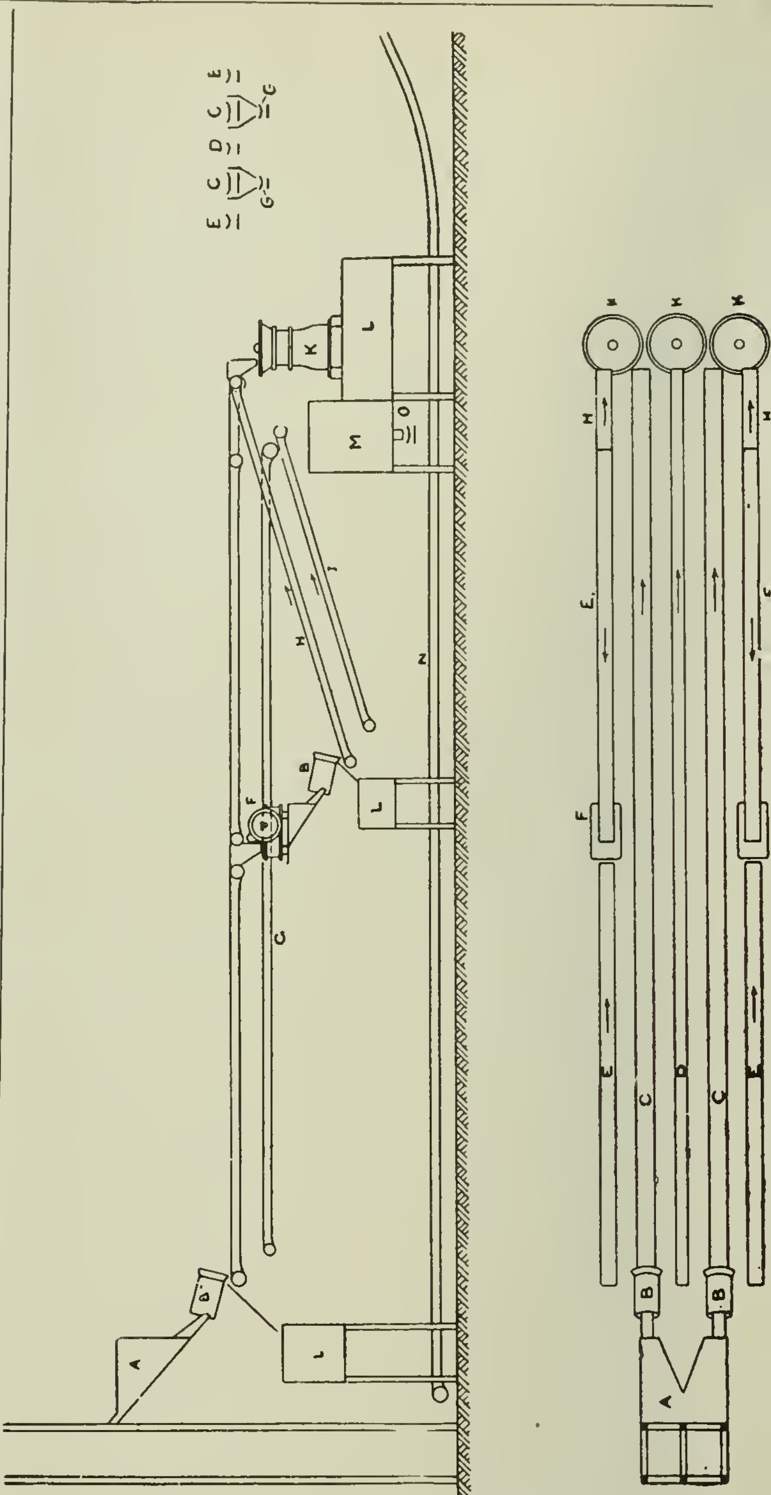
The new design of plant would naturally lend itself most readily for central sorting stations, as there the conditions would be the most favourable for efficient work. I have calculated that with a plant of this type and the principle of close sorting strictly carried out, the profits from a given mine can under certain conditions be considerably increased, while the capital outlay can at the same time be reduced. The following tabulated statement embodies these calculations :—(See 1st table, p. 95).

To the difference in favour of more intensive sorting must be added the saving in capital outlay equivalent to £68,000.

It will be interesting to see how a better extraction from higher grade ore milled affects the result, and to that end the following calculations have been made. It is assumed that a further expenditure of 5 per cent. on the reduction plant will raise the extraction by 2 per cent., whilst an additional outlay of 10 per cent. would raise the extraction by 3 per cent. or 94 per cent. and 95 per cent., which seems to be the economic limit for ore milled at a value of 23·106s. We have therefore :—(See 2nd table, p. 95).

An extraction of 94 per cent. for the 30·55 ore will give practically the same residue as 92 per cent. extraction for 23·106s. ore.

Provided, therefore, that it is possible to sort so as to have a waste dump at an average value of ·1 dwts., the very useful saving of £2,480 will result when obtaining 94 per cent. extraction. If it should be thought that the value of the waste dump cannot be kept at ·1 dwt. per ton, the profit of £2,480 can then be used for increasing the value of the dump. A calculation shows that the value of the waste



PROPOSED IMPROVED SORTING AND BREAKING PLANT.

A. Coarse-Ore Bin.
B. Washing and Screening Trommels.
C. First Sorting Belts.
D. Ore Conveyor.
E. Conveyors for Waste and Ore.
F. Coarse Breakers.
G. Waste Rock Conveyors.
H. Second Stage Sorting Belts.
I. Second Stage Waste Rock Conveyors.
K. Fine Breakers.
L. Fine Bins.
M. Waste Rock Bin.
N. Ore Conveyor to Mill.
O. Waste Conveyor to Dump.

Comparative Statement of Capital Outlay, Working Costs, and Profits of Closer Sorting and Ordinary Sorting.

METHOD OF WORKING.	ORDINARY SORTING.	CLOSER SORTING.
1 Tons hoisted per day	4,000 tons	4,000 tons
2 Percentage sorted out	15 per cent.	35 per cent.
3 Tons sorted out	600 tons	1,400 tons
4 Tons milled per day	3,400 tons	2,600 tons
5 Capital cost of sorting and breaking Plant	£36,000	£48,000
6 Capital charges at 12 per cent. per annum	£4,320	£5,760
7 Capital charges per day	£12	£16
8 Capital charges per ton milled	·070s.	·123s.
9 Capital charges per ton sorted out	·40s.	·228s.
10 Capital cost of reduction plant	£340,000	£260,000
11 Capital charges at 12 per cent. per annum	£40,800	£31,200
12 Capital charges per day	£113 6s. 8d.	£86 13s. 4d.
13 Capital charges per ton milled	·667s.	·667s.
14 Capital charges per ton sorted out	3·77s.	1·238s.
15 Cost of sorting per ton milled	·264s.*	1·908s.
16 Cost of sorting per ton sorted out	1·767s.*	3·534s.†
17 Cost of breaking per ton milled	·279s.*	·279s.
18 Total cost of sorting and breaking per ton milled (8 + 13 + 15 + 17)	1·278s.	2·977s.
19 Cost of milling and cyaniding per ton milled (assumed)	3·500s.	3·500s.
20 Total reduction costs per ton milled	4·778s.	6·477s.
21 Value of ore mined per ton (assumed)	20s.	20s.
22 Value of waste sorted out per ton	·6 dwts. = 2·4s.	·1 dwts. = ·4s.
23 Value of ore milled per ton	23·106s.	30·55s.
24 Value of gold recovered (92 per cent. extraction)	21·275s.	28·106s.
25 Cost of mining and hoisting per ton milled	8·000s.	10·461s.
26 Total working costs per ton milled	12·778s.	16·938s.
27 Net profit per ton milled (24—26)	8·497s.	11·168s.
28 Net profit per month	£43,334	£43,550
29 Difference in favour of close sorting	£216	—

*According to Ross E. Browne.
†Based on doubling the cost given by R. E. Browne.

METHOD OF WORKING.	ORDINARY SORTING.	CLOSER SORTING.
Percentage of Extraction.	94%	95%
10 Capital cost of reduction plant	£273,000	£286,000
11 Capital charges per ton milled	·700s.	·735s.
18 Total cost of sorting and breaking per ton milled	3·010s.	3·045s.
24 Value of gold recovered	28·717s.	29·023s.
26 Total working costs per ton crushed	16·971s.	17·006s.
27 Net profit per ton milled	11·746s.	12·017s.
28 Net profit per month	£45,814	£46,866
29 Difference in favour of closer sorting	£2,480	£3,532

discarded (when sorting at the rate of 35 per cent.) can be raised to approximately ·365 dwt. before the saving is wiped out, still leaving a saving on capital account of £55,000.

There is another point to be considered, and that is the greater expenditure of the mine with closer sorting, amounting to nearly £1,000 per month, practically all of which will be used to pay for extra labour, and assuming that all the profits go to shareholders abroad, this extra expenditure increases the purchasing power of the local population.

I trust that I have made out a case for further investigation, although I have no doubt that many points can be raised against the arguments advanced, particularly from the mining point of view. The difference of 2·461s. per ton milled (line 25) in mining costs points to the fact that the ore should be so mined that the percentage of waste sorted out is reduced, while at the same time keeping the value of the waste dump at a low figure. I do not presume to have solved the problem, and I merely intended to provide an opportunity for discussing the most important question of sorting before this Association."—C. O. SCHMIDT, *Journal of S. A. Association of Engineers*, Feb., 1910, p. 151. (H. A. W.)

IMPROVEMENT IN COPPER ALLOYS.—"The manner of introducing certain other constituents into brass for the purpose of increasing its tensile strength and ductility is the basis of a patent granted to S. C. Peck and W. R. Hodgkinson, of London. Small proportions of chromium, manganese, nickel, cobalt or zinc may improve the quality of the alloy. The preferred manner of introducing these elements is by the addition of the phosphate of the metal with a reducing agent such as carbon, to a small quantity of copper or one of its alloys. This gives a metallic solution of the phosphide of the metal to be added, and this metallic solution may then be used to alloy molten brass in the desired proportions. In the second operation some of the zinc in the brass is volatilized as zinc phosphide, leaving the brass alloyed with the metal added. An actual test was as follows: 2600 grms. of brass (Cu. 70%, Zn. 30%) were melted in a crucible with 200 grms. of a mixture of equal parts of chromium phosphate and powdered charcoal. The temperature was about 1000 C. The addition of charcoal and chromium phosphate was repeated twice at intervals of an hour. After four hours heating the cast was made and this metal containing chromium phosphide was used to improve the qualities of brass as follows: 1516 grms. copper and 655 grms. zinc were melted together and to the melt was added 82·5 grms. of the chromium phosphide brass. The resulting metal tested 19 tons per sq. in. tensile strength, and showed an elongation of 72% on two inches. The same brass without the chromium phosphide added showed tensile strength of 14·4 tons per sq. in. and elongation of 38·5% on two inches (943,159. Dec. 14, 1909.)"—*Metallurgical and Chemical Engineering*, viii, 4, 209, April, 1910, p. 209. (J. A. W.)

A New Process in Iron Metallurgy.—"One of the most important additions to the list of structural materials, and the most interesting appearance in iron metallurgy, of recent years, is a remarkable product of the open-hearth steel furnace, placed on the market a short time ago by the American Rolling Mill Company, of Middletown, Ohio, under the name of 'American Ingot Iron.' While it is produced by the basic open-hearth process, in chemical composition and mechanical properties the ingot iron is closely similar to wrought iron. It differs from steel in having its carbon, manganese and silicon contents reduced to small fractions of the amounts found in ordinary soft steel. In a typical sample, the carbon and sulphur are each reduced to 0.02%, manganese to 0·01%, oxygen to 0·03%, and phosphorus and silicon to slight traces. In appearance, strength (tests indicate an ultimate tensile strength of 47,000 to 49,000 lb. per sq. in.), and ductility the new metal is practically identical with wrought iron.

The prime quality of the ingot iron, however, is its resistance to corrosion.

The earliest product of the mills was an ordinary soft steel, running 0·20% carbon and 0·40% manganese. Attention was first directed to the reduction of the carbon, and after some time of working this impurity was brought down to 0·10%, giving a dead-soft metal, sheets rolled from which attained a special reputation. A demand for corrugated-steel culverts, which arose in 1904, was an incentive to still further reduction of impurities. The ferromanganese addition for clearing and recarburizing was cut down to what was thought a minimum consistent with producing workable metal, and steel running as low as 0·08 to 0·10% carbon, 0·10 to 0·20% manganese, and 0·03% phosphorus was thus obtained. So far the reduction of the manganese had been undertaken only as part of the general scheme for lowering the impurities, but the publication of Dr. Cushman's research results, which showed manganese to be the most important element in the promotion of corrosion, led to experiments on the reduction of manganese which resulted in a decision to omit entirely the ferromanganese treatment. A long period of struggle with defective metal ensued, but eventually the difficulties, chief of which were the cracking of the edges of the sheets in rolling and poor galvanising quality due to blistering, were completely overcome.

The process by which the ingot iron is made is essentially very simple. 'The same basic-lined furnace is used as in regular open-hearth practice, and practically the same charge is made, but the boiling is continued to an excessive degree, so as to oxidize the impurities more thoroughly than in ordinary steel practice. Naturally the process in its final stages goes on at a much higher temperature, on account of the higher melting point of the purer material. The usual ferromanganese recarburizer is omitted, and instead an addition of ferro-silicon or an equivalent substance is made to purify the bath of its entrained oxides. Finally, aluminium is added in the ladle, to carry out the gases dissolved in the fluid metal.'

The furnaces now in use work entirely on cold pig iron, low in silicon and sulphur, of a grade slightly above the limiting phosphorus content for Bessemer working. A fairly large proportion of scrap is charged, some of it open-hearth scrap from the mill itself, but the larger part low-carbon steel turnings. Ore is not conveniently obtained, and whenever a sufficient quantity is available mill scale is substituted for ore in the charge. To produce a slag, limestone and fluorspar are used. A comparatively large amount of the latter flux is necessary to prevent the return of the phosphorus to the metal at the high temperature maintained towards the end of the process (3,000 to 3,100° F.).

'The essentials of the melting process are the maintenance of an active basic slag, continuance of the boiling process until all foreign elements in the iron are reduced practically to zero, keeping up a sufficiently high heat for complete fusion and active boiling, and finally the treatment for removing oxygen and other gases by the addition of purifiers. In the removal of the impurities, ordinary oxidation and absorption by the basic slag plays the chief part, as usual. It is found advantageous, however, to add ore, or mill scale, for helping with the oxidation.'

'The removal of oxygen (presumably contained in the metal as oxides) is most important. When this part of the process is imperfect, the metal is red-short and cracks on rolling. In ordinary steel making, spiegeleisen or ferromanganese have the chief duty of burning out the oxygen, reducing it from the metal both through their carbon and their manganese. In the present ingot-iron process, it has been found, the removal of the oxygen can be accomplished by various substances. Pig iron in itself is a satisfactory means, but ferrosilicon is a more active agent.' The removal of other gases is accomplished by adding granular aluminium in the ladle, the quantity usually running well below 0·1% of the weight of the iron charged.

The average time from charging to tapping is 10 hours, an increase of several hours over steel furnace practice wholly traceable to the thorough elimination of impurities. The long duration of the melt necessarily increases the cost of the product materially over that of ordinary steel. Increased expenditure for fuel and, more important, increased plant and labour charges (the average output being admittedly brought down to two-thirds the usual figures) must appear in higher cost. The cost is increased also by the destructive effects of the high temperatures on furnace and ladle linings, brickwork, ingot moulds, etc., all of which require more frequent renewal than in steel practice.

With watchful, intelligent control the process is in no way erratic or variable, and highly pure metal, carrying total impurities (except oxygen) amounting to 0·05 to 0·08% can be regularly produced. The best evidence obtainable points to strength and and ductility identical with those of wrought iron. For a highly ductile metal with an ultimate strength of 47,000 to 50,000 lb. per sq. in., the yield point is unusually high, 39,000 to 42,000 lb. per sq. in. As to the relative resistance to corrosion of steel, wrought iron and ingot iron, sulphuric acid tests show ingot iron to have a resistance 40 to 60 times as great as that of steel and 20 to 40 times as great as wrought iron. The large ratios of resistance make it permissible to transfer the comparison to atmospheric rusting, after making a large reduction as a safety margin. It may be concluded that the ingot iron will resist atmospheric corrosion at least several times as well as mild steel, probably many times better, and markedly better than good grades of wrought iron.

Up to the present ingot iron has been produced only in the form of sheets. It has become popular in this form on account of its high degree of weldability and the ease with which it lends itself to the galvanising and enamelling processes."—*Engineering News*, Jan. 6; *Engineering Magazine*, Feb., 1910, p. 766. (J. A. W.)

MINING.

ACCIDENTS IN MINE SHAFTS.—"*Winding Engines.*—Attention is first directed under this heading to the probable adoption, to an increasing extent in the future, of electricity as the motive power for winding purposes in the United Kingdom, allied with the Koepe and Whiting system, which allows of great reduction in the weight of the drum, and, because of its more balanced load, gives a more effective use of the power available for winding. The apparatus at the No. 11 Pit of the Bethune Collieries is quoted as typical of this combination. In this case the drum is 26 ft. in diameter, and a steel rope 2·24 in. in diameter makes a half turn only, lying in a birch 'trod' or groove-way, the friction of steel against birch affording sufficient grip to prevent slipping. No preference from the point of view of safety is given to vertical over horizontal winding engines or *vice versa.* Attention is, however, drawn to two recent accidents due to defective forging of crank or drum shafts, and a suggestion is made that the marine practice of boring a hole through the centre of the shaft would indicate beforehand the quality of the shaft. The question of more than one winding engine in one room is also dealt with, and such practice is deprecated. With regard to speed of winding, the general rule (26) on the subject is quoted, and it is pointed out that although the 'automatic contrivance' therein mentioned is generally interpreted to mean a detaching hook, and the majority of mines are now so provided, such a contrivance only comes into action after the overwind, and therefore it is doubtful whether such contrivance satisfies the requirements of the Act in this respect. There is, too, considerable divergence in the different codes of special rules in the fixing of the point in the shaft within which the slow speed is operative It is suggested that the fixing of the point should depend on the diameter of the drum and speed of wind, but that if detaching hooks and speed controllers were in universal use no provision of this kind would be necessary. No recommendation is made with regard to the number of men raised or lowered per wind or to the actual speed of winding.

Speed Controlling Devices.—Detailed descriptions and drawings of the three arrangements best known in Great Britain are given, namely:—The Whitmore, the Visor, and the Futer's gear. General approval of these gears is noted with a qualification in these terms: 'when properly set and regulated.' Two cases are quoted where accidents might have been entirely avoided had winding engines been fitted with satisfactory speed-controlling devices.

Brakes.—In the drafting of a new rule in place of general rule 30, it will be necessary to include electricity as one of the forms of motive power. It is also suggested that the term 'adequate brake power' is in need of closer definition. Brakes are classed under three heads—foot brakes, steam brakes, and special brakes. The Committee seem to have little faith in the former, and suggest that 'however efficient a foot brake may be, as such, its holding power is not very great.' No preference is given to, or distinction made between, band or post brakes. Approval is expressed in the modern practice of providing both foot and steam brakes, but the possible occurrence of an accident is mentioned where the steam brake is rendered ineffective, and the foot brake being of insufficient power to hold the drum. The Committee therefore recommend that 'drums of winding engines should be fitted with two brakes, either one of which will hold the cage, and of these one should be connected to a device for putting it into operation should the engineman, for any reason, fail to do so.'

Winding Ropes and their Attachment to the Cage.—Under this heading are comprised winding ropes, capels or socketings, detaching hooks, and suspension chains. At the outset the Committee express an opinion that nothing is to be desired as regards the standard of excellence of winding ropes in use in this country. A table is given showing grounds for this opinion in the comparative fewness of cases of accident due to breakage of ropes. During the ten years, 1898-1907, the accidents numbered 13 fatal, causing 32 deaths, and 5 non-fatal, causing injuries to 8 persons.

In discussing ropes the Committee deal first with the safety factor, pointing out that this varies from one-sixth to one-tenth of the breaking strain of the rope as determined by the manufacturer. This factor is not, however, a recognised fraction of the total winding medium existing between drum and cage, *i.e.*, rope, capel, detaching hook, and shackles.

The Committee, in view of the difficulty of maintaining a minimum factor of safety in cases of great depth, recommend that the better plan would be to specify the margin that should be allowed as between the breaking strain of the rope, chains, etc , and the load, instancing that in a deep shaft with 20 tons load and rope having a breaking strain of 200 tons, there is a margin of 180 tons, whereas in a shallow pit, with load of 1 ton and rope breaking at 20 tons, there is a margin of 19 tons only. In the former case the factor of safety is 10, and in the latter, 20.

The following recommendations are made as to ropes:—

(*a*) When kept in stock they should be carefully stored, and on no account placed on the ground, but coved and placed on a stand or shelf, and should be regularly inspected.

(*b*) When about to be used a rope should be placed on a turn-table, in order to avoid kinking, and carefully uncoiled from the outer end.

(*c*) A rope should not be worked round a drum or over a pully of insufficient diameter, and should be prevented from striking against any hard substances.

(*d*) It should daily receive a regular dressing of suitable grease, which should be thoroughly laid on, (*e*) Steadiness of movement at starting should be aimed at.

(*f*) A rope should not be changed from a large to a smaller drum and *vice versa*.

(*g*) Extreme variations of heat and cold should be avoided as far as possible.

Capels are considered under six heads as follows:—(1) Grooved link or eyepiece with clamp or splicing; (2) rope cone with socket and collars; (3) white metal capping; (4) Derbyshire's patent capping; (5) Becker's patent wedge capping; (6) Scott and Caddy's patent capping.

The use of rivets for holding the socket in the second type is condemned as tending greatly to weaken the rope. The tendency of wires to break at the point where they are bent back is also noted.

In the third type the necessity for thoroughly cleaning the wires and avoiding too high a temperature of the molten white metal is recognised, as well as the advantage derived by keeping the wires straight after untwisting.

Periodical recapping is strongly approved of; it ensures a continuously efficient capping, it allows of an internal examination of the rope, and it tends to lengthen the life of the rope by altering the positions of weak points at the contact of the rope with the pulley and drum. The practice, noticed at several collieries, where recapping is carried out every six months and a record kept and posted up in the winding engine house, is highly recommended, and it is suggested where such a rule is lacking that one on this matter might be incorporated in the special rules. The Committee deem the use of detaching hooks very desirable in all cases, and recommend the provision of jack catches in the headgear to hold the cage when disconnected from the rope.

As regards the attachment of the cage to the rope, the use of rigid rods instead of chains, as was the case at Fogg's Colliery Bolton, where an accident occurred on October 4, 1907. causing the loss of ten lives, is considered unsatisfactory, as their rigidity does not allow sufficient play. An instance of the introduction of a spring or buffer box between the detaching hooks and the cage is quoted, the use of which has been attended with considerable saving in the lives of the winding ropes.

Finally, under this head, the Committee recommend the use of a minimum of four chains for suspension of the cage from the shackle ring or plate in shafts where persons are raised or lowered, and suggest the introduction of a rule making compulsory the periodical annealing of these chains.

Cages.—The Committee having considered only that part of the subject relating to existing and completed shafts, draw attention to the fact that their remarks throughout the Report are not applicable to shafts in course of being sunk. The desirability of cages being used either for winding men or mineral, both from the point of view of safety and convenience, is expressed. Although their use for these purposes is now almost universal, there are still a few instances to the contrary. Little or no fault is to be found with the protection of the sides of the cages at all well-equipped collieries. There is still, however, room for considerable improvement in this respect at a number of mines where the sides of the cage (which is a mere skeleton carriage) afford practically no hindrance to the falling out of persons, materials or minerals, and no means exist to prevent contact with guides, buntons, etc., in the shaft.

On the question of cage gates to protect the otherwise open ends, the Committee find 'that the balance of probabilities appears to be in favour of the use of such types of gates which admit of persons being got out of the cage in case of emergency, rather than in the disuse of gates altogether.'

Some cages inspected were provided with slanting or conical tops, the suggestion being that this shape induced a less injurious effect on the ventilation and conduced to a steadier movement of the cage. Information supplied by Mr. C. E. Rhodes from actual experience admits of no other conclusion than that such effects are not obtained, while a slanting top has further disadvantages in that it is inconvenient for loading timber on the cage, and prevents the use of the spring attachment referred to above.

With regard to securing tubs in cages, it is suggested that an automatic arrangement which will act independently of manipulation, possesses advantages both in respect to safety and economy over catches or stops whose action is governed by hand.

The question of safety catches or clutches designed to come into action in the event of the rope breaking is discussed at considerable length. The conclusion of the Committee is summed up in these words: 'In our opinion it is much better to rely on good material in the ropes, chains, and other tackle, careful periodical examination and recapping, annealing, etc., during their working life, to fix a maximum period for the life of a rope, and to allow a liberal margin between the working load and the breaking strain of the winding tackle, than to employ safeguards, which, under the conditions obtaining in this country, can only be considered of doubtful value.' The conditions specified are the rope guides, in these days so largely used, and the great loads and high speeds of winding in modern shafts. The Committee commend to the consideration of the Institution of Mining Engineers a suggestion for putting these appliances to practical test in a colliery shaft in order to stimulate invention in this direction.

It is pointed out that smoothness of motion of the cages is conducive to safety, and that the best means to secure this, especially in deep shafts, is by the attachment of a balance rope beneath the two cages.

Guides.—Cage guides are divided into two classes—rigid and flexible guides. The former includes those constructed of wood, iron, or steel, and the latter those of iron or steel wire rope. At the outset preference is given to rope guides, except where owing to the narrowness of the shaft their use is precluded.

In the case of rigid guides, the importance of good joints is emphasised. Dangers to be guarded against in the use of flexible guides are of two kinds—(1) cages meeting owing to too much swaying of the guides; (2) breaking the rods. The former difficulty is surmounted by ensuring sufficient tension on the guide rods, and by the introduction of rubber or dummy guides between the cages; the latter difficulty, by care in examination of the guides as to wear and corrosion, and by avoiding the application of excessive strain. The daily examination of the guides gives warning of undue wear, and frequent measurement of the diameter of the rod by the calliper in as many places as possible is recommended. Corrosion can be noticed in the exposed portions of the guide, and can be avoided by efficient daily lubrication. The difficulty lies in the portions hidden from view, where it passes through the fast scaffold, at which point also it is subject to damage by fraying, owing to side movement. Descriptions and drawings are given of effective methods of obviating these difficulties, and whereby the portions of the guide usually hidden from view are brought into such positions that a proper examination may be made.

No generally agreed upon rule is available as to what weight should be attached to rope guides to produce the necessary tension or rigidity. Where weights are used, it is pointed out that in order to keep them as steady as possible they are frequently immersed in the sump, protected or unprotected from the action of the water, but if they are so immersed it is difficult, if not impossible, to carry out that part of General Rule 5 relating to the daily examination of the guides.

Signalling and Indicators.—The Committee summarise their conclusions in respect to signalling in shafts as follows :—(1) A universal code of signals is advantageous in so far as relates to sending the cages away either from the top or the bottom of the shaft, and in what may be called the 'simple' signalling, common to every colliery. (2) It is desirable that the banksman should receive signals at the top of the shaft direct, and not as in some cases, hear them as delivered in the engine-house. The best means of signalling is that in which the signals are communicated simultaneously to both engineman and banksman.

The need for uniformity in the so-called 'simple signalling' is emphasised by the fact that the codes of signals now in use in different parts of the country are subject to much variation, and dangers are likely to arise through confusion of the different codes, especially as both miners and other colliery workmen are somewhat of a migratory class. The circumstances of the shaft accident at Rawdon Colliery have drawn attention to the danger arising where a banksman has to depend on his hearing for signals delivered only in the engine-house. The signalling arrangements introduced at Barrow Colliery are quoted as remarkably efficient. Here the method is electrical, and signals from three landings are received by banksman and engineman simultaneously, the engineman only being able to give down signals.

A description of the Karlik Tachograph is given, and its use is commended to the notice of mining engineers. It is considered that any apparatus which would record with exactness the time occupied by a wind and the variation in the rapidity of the same, when drawing coal or men, would be of great value. The above-mentioned instrument fulfils these conditions admirably.

Loading and Unloading of Cages.—Under this head are included :—Keps, fallers, or pillars; winding men or minerals from mid-insets; barriers; persons riding against full tubs, or with gear; additional outlet for persons. The use of keps at the surface is strongly advocated as tending to conditions of greater safety. At intermediate levels, however, they are likely to be a source of danger, for, if by any chance they were left projecting into the shaft, a descending cage going to a lower level might cause a serious accident. At the bottom of the shaft their use should not be made compulsory, as frequently when single-decked cages are used the cage rests on a platform.

Mid-insets it is stated, are not often used for loading mineral into the cage, more frequently serving the purpose of hangings-on for men. Where the latter is the case, a staging to suit the number of cage decks, in order to avoid raising or lowering the cage, is suggested, and this staging should be as far as practicable of non-inflammable material. Where mineral or men are wound, it is recommended that the drop-sheet, where such a contrivance is necessary, to span the interval between the cage and the landing, should be arranged so that in the event of the cage being signalled away before the drop-sheet is drawn back it may be forced back by the cage in its ascent. Again, in cases where men are being wound it is desirable that the cage should proceed direct to the surface after being signalled away, without being lowered down to a lower level to allow some of the occupants of the cage to get off it, or others to get on, at the lower level.

No recommendations are made with respect to the barriers, either at the surface or at insets, but an arrangement in use at the Mariemont Pits in Belgium is described. This is a subsidiary gate, attached and lifted by jointed rods from the ordinary gate which the cage lifts, at such a distance below the surface level that it comes into use when the bottom deck of the cage is lifted a little too far above the landing, so protecting the workmen against the danger of pushing a tub, and being himself drawn by the tub into the shaft.

The practice of persons riding against loads or with gear is strongly deprecated. It is recommended that it should be universally prohibited.

With regard to additional outlets, the Committee consider that greater use might be made, in some cases, of the shafts commonly known as 'back' shafts, especially in case of emergency; and that better provision could be made by providing these shafts with a cage instead of a 'hoppit' or 'kibble,' as is frequently the case. It is indicated that a kibble cannot be considered a 'proper apparatus' as required by Section 16 (1) of the Coal Mines Regulation Act."—F. H. WYNNE.—*North Staffordshire Institute of Mining and Mechanical Engineers. Iron and Coal Trades Review*, Feb. 4, 1910, p. 166. (A. R.)

WINDING AT THE CITY DEEP, LTD.—"The development of 'deep levels' on a large scale on the Rand has once more induced a discussion of the most economicl and safe methods of bringing the miners to and from their work. At the City Deep special gear has been invented for the purpose of changing from skip to cage with as little delay as possible, the transition from ore-hoisting to man-transport being thereby effected rapidly. The shaft has six hoisting compartments, of which two will be used solely for transporting the men and the remaining four can be used either for hoisting the ore or transporting the men. The cages in the two former will accommodate 60 men each and the latter 48 men each. The changing gear consists of a carriage travelling on rails, which are laid in line upon opposite sides of each hoisting-compartment. The main portion of the carriage consists of two steel side-plates 21 ft. 6 in. long, 4 ft. deep, and ½ in. thick, rigidly braced together and carried on two pairs of wheels, one of each pair of wheels resting on the rails on each side of the shaft. The side-plates are at a sufficient distance apart to permit the free passage of the skip between them when the gear is not in use. Upon one end of the carriage is borne a 3-deck cage with a capacity to hold 16 men on each floor, and upon the other end there is a platform that receives the skip when the latter is removed from the hauling rope. In making the change the skip is raised above the level of the rails and retained by 'dogs' and the platform end of carriage is brought beneath it by a worm-gear connected with the hind wheels. The skip is lowered to the platform and unhooked, and the carriage is then moved back again so as to bring the cage to the centre of the shaft. During the travel the cage is hooked to the hauling rope and filled with men, and is immediately ready for its upward journey. When all six compartments are being used for transport of men the total load per trip will be 156, and allowing 3 minutes for each trip, including loading and discharge, it will be possible to transport 3,120 men per hour. Whitford and Mills have also devised a new form of hopper for rapidly filling the skips. It is suspended from a pair of swinging arms at a point just above its centre of gravity. The descending skip tilts the hopper and receives its charge, and when ascending again lifts the hopper back into its place ready to receive another charge from the bin. This system is to be adopted at the main loading-station of the mine, where, at a depth of 3,000 ft. there will be a bin having a capacity of 800 tons "—*Mining Magazine*, March, 1910, p. 223. (A. R.)

FENCE GATES FOR WINDING SHAFT CAGES.—"After a fatal accident, observations were frequently made by the coroner, jurymen, or the inspector of mines that improvements might be made whereby the danger might be avoided or lessened, as was the case some little while ago after an accident involving loss of life through the tilting of a cage whilst running in the shaft. In that case the coroner's recommendation, endorsed by the inspector of mines, was 'that gates should be on each cage when men ascend or descend.' That incident induced the writer to give some consideration to the subject. He was of opinion that gates could be attached to all cages at small expense, thus assuring the safety of the workmen while not affecting the output. He brought before the notice of the meeting three methods he had devised of attaching gates to cages, and recommended that they should be adopted when drawing coals in conjunction with the existing cage snecks. It was not an uncommon experience to have tubs insecurely snecked, which often caused an accident in the shaft, a danger which would be entirely avoided if his recommendations were acted upon. The first type of gate he submitted was a collapsible gate for cages with one or more decks. The object of that type was to allow the gate to collapse into a minimum of space available in the case of complicated head gears and onsets. If possible the gate should be so constructed that in case the cage was signalled away without its being closed, no fouling would take place during its journey in the shaft. The gate might be constructed of flat iron bars built up in such a manner as to allow of both a horizontal and a vertical motion. The vertical bars were $1\frac{1}{4}$ in. wide by $\frac{3}{8}$ in. thick, and were rivetted 9 in. apart to the horizontal bars (which were $1\frac{3}{4}$ in. wide by $\frac{7}{16}$ in. thick) with loose rivets $\frac{5}{8}$ in. diameter, so as to allow the gate to collapse into the desired form. The vertical bars were placed on the inside of the horizontal bars in case of an undue stress being thrown on them, when the rivet heads would not be strained, as would be the case if they were fixed on the outside. The horizontal bars were spaced 24 in. apart, the lower bar being 6 in. from the bottom of the cage deck, making the total height of the gate $33\frac{1}{2}$ in. The whole gate swung horizontally upon a pivot $\frac{3}{4}$ in. in diameter, to which the horizontal bars were attached by loose rivets, $\frac{7}{8}$ in. in diameter, thus allowing of the vertical movement. The top horizontal bar had a handhole, 4 in. by $1\frac{1}{2}$ in., to allow for the lifting of the gate. When the gate was closed, the ends of the horizontal bars rested in slots, being kept in that position by catches. When opening the gate, it was lifted clear of the catches and was moved first in a horizontal and then in a vertical direction, in which latter position it was retained by the shackle until the cage was ready for moving away, when the gate was closed. The second type of gate was a rigid gate for cages with one or more decks. That was of a somewhat similar type to the collapsible gate and was recommended when space was not a serious consideration. It must, however, be closed before any movement of the cage took place, otherwise it would foul obstacles while the cage was running in the shaft. In that gate the vertical bars were rivetted with fixed joints to the horizontal bars, and the whole gate swung vertically on only one pivot or axis, which was $\frac{3}{4}$ in. in diameter. On the hinged end of the gate there was attached an additional vertical bar, $1\frac{3}{4}$ in. by $\frac{3}{8}$ in. for the purpose of stiffening the structure. There was, moreover, an additional hand-hole provided in the lower horizontal bar, to which was also attached a stop-piece, $2\frac{1}{2}$ in. by $\frac{1}{2}$ in. by $\frac{1}{2}$ in., which prevented the gate, when opened, from going too far through the slot. In all other particulars it was identical with the collapsible gate. Being in its general construction very simple, it was not likely, when once installed, to get out of order. The third type was a false-bottom or telescopic frame gate for cages with one deck only. In that type, a framework, attached to the gate by pins $\frac{7}{8}$ in. in diameter was constructed under the cage in such a manner that when the cage was lowered to the bottom of the shaft the framework came into contact with the beating beams, the result being the vertical lifting of the gate, which of course gave access to the inside of the cage. The gate slid in slots provided at each side of the cage. When the cage was lifted, the gate, by its own weight, automatically closed itself. At the bank-level, the opening and closing of the gate were also automatic. A lever was pivoted at its centre to the side of the cage, the lower end being attached to the bottom of the gate. When the cage ascended, the free end of the lever engaged with a fixed sheave at bank, the depression of the one end of the lever lifting the gate attached to the opposite end. When the cage was lowered away, the gate, as when at the bottom of the shaft, automatically closed itself. The framework below the cage was constructed of flat iron bars, $2\frac{1}{2}$ by $\frac{1}{2}$ in., fixed securely and rigidly by bars of a similar section and by bars, 2 in. by $\frac{3}{4}$ in., which ensured the proper gauge of the bars mentioned at the beginning of the sentence. The gate itself was made up of an expanded metal panel, in the centre of which was a small door, to allow of access to offsets in the shaft. This door, which was constructed of flat iron bars, $1\frac{1}{2}$ in. by $\frac{3}{8}$ in., opened inwards on hinges, and was so hung as to close of its own accord. It was kept in that position by a self-closing latch which locked into a flat iron bar, 1 in. by $\frac{1}{2}$ in. The small door might, if preferred, be constructed in halves, so as to take up less room in the cage when opened. When the gate was opened, the height from the floor of the cage to the bottom of the gate was $35\frac{1}{2}$ in."—C. A. CROFTON, *North of England Inst. Mining and Mechanical Engineers Mining Engineering*, March, 1910, p. 351. (A. R.)

MINE SUBSIDENCES.—In our issue for December last we gave some particulars of the proposed method for packing old stopes on the Rand. This new departure has drawn attention once more to the general question of the safety of the workings. Some weeks ago there was an accident at the Ferreira Deep, indicating that sufficient ground had not been left intact in the neighbourhood of the shaft. In order to remedy the defect concrete pillars are being provided. Similarly weakness has been discovered at the Robinson Deep, just to the South of the Ferreira Deep, and in order to prevent any collapse a large concrete pillar has been built. In connection with the Jupiter, at which mine the shafts are unusually deep, a pillar 500 ft. in diameter has been left round them, while at the Cinderella Deep, where there is only one shaft, a pillar 600 ft. in diameter is to be left. These supports may be considered unnecessarily large, and the policy of leaving so much ore in the mine may be debated, but the engineers would hardly have adopted such a policy without good reason. Another point in connection with the safety of the mines that has arisen lately is the question of the advisability of working under the battery foundations. The stamps are so heavy and the plant so extensive nowadays that the vibration is felt a long way down. For instance, at the Simmer and Jack, vibration of the 320 stamps can be felt in the hanging wall 300 ft. below the surface although strong pillars have been left. It is generally considered that, having regard to the heavier machinery and the larger reduction plants, it will be necessary to make close inquiry into the state of the workings at many of the older mines."—*Mining Magazine*, March, 1910, p. 228. (A. R.)

EARTH-SHAKES IN MINING DISTRICTS.—"Certain mining districts in Great Britain are visited from time to time by sharp local shocks, which in some ways resemble earthquakes, but in others are very different. To distinguish them from true earthquakes, and yet to represent the points of similarity, they have been called 'earth-shakes.' I propose in this paper to describe their characteristic features, and also to offer a theory of their origin, which, if correct, would seem to indicate that they are due to natural causes aided by the working of the mines.

During the last twenty-one years I have been able to study seventeen of these earth-shakes, though probably many more, unknown to me, have occurred. Five of them have taken place in the Little Rhondda Valley in Glamorganshire, four in Cornwall, four at Pendleton, near Manchester, and one each at Kilsyth, Barnsley, Abercarn, and Eastwood, near Nottingham.

Nature of the Earth-Shakes.—The small size of the disturbed area is one of the most remarkable features of these earth-shakes. It ranges from one square mile or less to 144 square miles, but only one (that at Pendleton in 1905) exceeds 20 square miles. Excluding this, the average of the disturbed areas is only 10 square miles. In seven cases the disturbed area is so small that the course of its boundary cannot be determined; in five, the boundary is approximately circular in form; while, in five others, it is slightly but distinctly elongated. Notwithstanding the small size of the disturbed areas, however, the intensity of the shocks was in some cases considerable. Two of them attained the degree 7 of the Rossi-Forel scale—that is, they were strong enough to throw down ornaments, vases, etc., five were of intensity 5, and the remainder of intensity 4. The contrast between the earth-shakes and British earthquakes is shown in the following table, in which the average disturbed areas in square miles are given for the three degrees of intensity involved:—

	Earth-shakes.	Earthquakes.
Intensity 7 ..	80 ...	27,850
,, 5 ...	11 ...	859
,, 4 ...	8½ ...	376

In its nature, the shock resembles that of a true earthquake, except as regards brevity and its sudden onset. In an ordinary earthquake the shock generally begins with a faint tremor, which rapidly increases in strength until a series of strong vibrations are felt, and then dies away again in a tremor. An earth-shake, as a rule, begins with one strong vibration, like that caused by blasting or the fall of a very heavy body; and this may be followed by a very brief tremor, such as would naturally follow either of these occurrences. Thus, instead of an average duration of from 4 to 8 or more seconds, the earth-shake lasts on an average for not more than 2 secs. It is worth noticing that the average duration of an earth-shake depends on the form of the disturbed area, being 1¾ secs. when the area is approximately circular and 2½ secs. when it is elongated in form.

The sound which accompanies the earth-shake also bears a close resemblance to that which attends an earthquake shock, and again is distinguished by its brevity. This is shown in the following table, in which the figures represent percentages of the observations for each class under the seven principal types of sound:—

	Passing wagons, etc.	Thunder.	Wind.	Tipping of a load of stones.	Fall of a heavy body.	Explosions.	Miscellaneous.
Earth-shakes—							
With elongated disturbed areas	20	20	1	13	15	26	5
Circular	5	22	—	5	32	35	—
*Earthquakes**—							
Strong	47	22	10	5	4	7	5
Moderate	46	27	3	5	6	9	3
Weak	29	31	6	7	8	15	4

The first three types may be regarded as of long, and the next three as of short, duration. Taking these six types only, we find that the percentages of reference to types of short duration are 57 and 72 for the two classes of earth-shakes, and 17, 20, and 31 for the three classes of earthquakes.

One other point of contrast between earth-shakes and earthquakes is worthy of notice. Though the number of records of earth-shakes in mines is much less than on the surface, it is clear, from the information we possess, that the region effected by the shake underground is not less extensive than the area disturbed by it on the surface, also that the shock is stronger and the sound louder in the mines than on the surface. Now, in ordinary earthquakes, the shock, if felt at all in mines, is much weaker than on the surface, and the average distance at which it is felt underground is only about one-third of that at the surface. Moreover, in earth-shakes, the sound is sometimes referred to as resembling that of the discharge of a shot in the workings, fragments of rock fall from the roof and dust rises from the floor.

Origin of the Earth-Shakes. Such marked contrasts between earthquakes and the earth-shakes of mining districts point to an essential difference in origin, and especially to difference in the depth and magnitude of the originating foci.

The most significant feature of the earth-shakes is the great intensity of the shock near the centre of a very small disturbed area and its rapid decrease towards the boundary. As will be seen from Fig. 1, this implies the existence of a comparatively shallow focus. The curves marked A and B in this diagram represent the intensities of the shock at different distances from the origin, A corresponding to a focus at a depth of one-quarter of a mile, and B to one at a depth of two miles. Both curves are drawn on the

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* 'Strong' earthquakes are those which disturb areas greater than 5,000 square miles; 'moderate' earthquakes, between 1,000 and 5,000 square miles; and 'weak' earthquakes, areas less than 1,000 square miles.

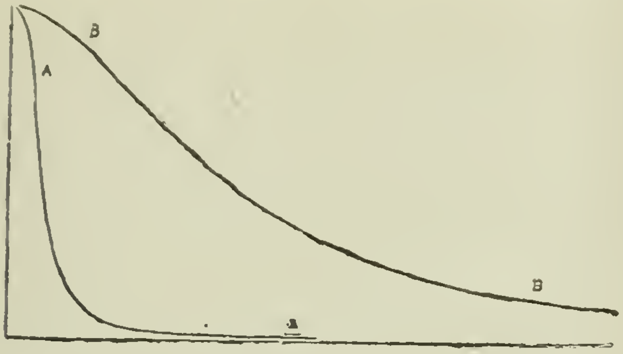

Fig. 1.

assumptions that the intensity of the shock at any point of the surface varies inversely as the square of the distance from the focus, and that the impulses are such that the intensity of the shocks at the point vertically above both foci is the same. On these assumptions, it follows that the shock that would be just perceptible at a distance of two miles from the centre for the shallow focus, would be perceptible at a distance of thirty-two miles for the deeper one—that is, the disturbed area of the latter shock would be 256 times that of the former. The observations made in mines also lead to the conclusion that the earth-shakes originate in foci which are not far below the surface, and, approximately, at the same depth as the workings in the neighbouring mines.

A second inference, of hardly less consequence, is that the foci of the earth-shakes are as a rule of small dimensions. This is shown by the small size of the disturbed areas, by the approximate circularity in form in some, and by the small difference between their length and width in others. The brevity of the shock and sound, especially in those which affected a nearly circular area, points to the same conclusion; for a focus several miles in length would give rise to a disturbance of some seconds duration—at any rate, at places in the direction of its longer axis.

These two conclusions regarding the small depth and size of the foci are clearly in favour of a local origin of the earth-shakes. In the Rhondda valleys they are generally, I believe, attributed to the rock falls in old workings; and this is, no doubt, the origin of some shocks, for such falls from the roof are known to occur and to lead to a subsidence of the ground above. The chief difficulties in the way of this explanation for the earth-shakes here referred to arise from the absence, so far as known, of any fallen masses, but chiefly from the close connexion between the earth-shakes and neighbouring faults.

As an example of this connexion, I reproduce here the map of the Pendleton earth-shake of November 25, 1905 (Fig. 2). This was one of the strongest earth-shakes felt for more than twenty years. It disturbed an area of 144 square miles, and the observations are so numerous that it is possible to draw a series of four isoseismal lines corresponding to intensities 7 to 4 of the Rossi-Forel scale. The centre of the innermost curve lies three-quarters of a mile north of the centre of Pendleton, and the longer axis of all the isoseismals are directed, approximately, N. 37 degs. W. Now, the Pendleton or Irwell Valley fault (represented by a broken line on the map), which has been traced for a distance of more than twenty miles from near Bolton to Poynton in Cheshire, passes close by the centre of the innermost isoseismal, and its mean direction in that district is N. 34 degs. W.

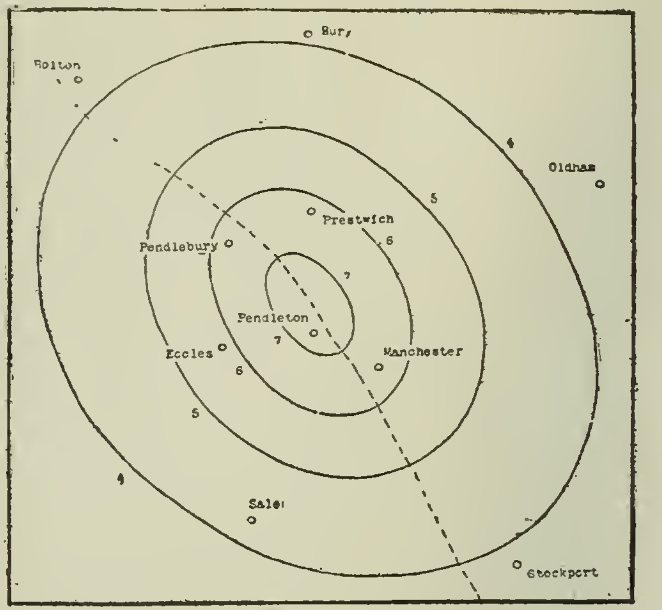

Fig. 2.

The evidence of other earth-shakes is similar, though not quite so detailed. On February 27, 1899, and April 7, 1900, earth shakes in the same district disturbed nearly circular areas, with their centres close to, and on the downthrow side of, the Pendleton fault. The Barnsley earth-shake of October 25, 1903, disturbed an elongated area with its longer axis directed N. 38 degs. W. and its centre close to, and on the downthrow side of, a fault running N. 41 degs. W. In the Little Rhondda Valley, the disturbed areas of the earth-shakes are nearly circular in form, the centre of the Llwynypia shake of June 22, 1889, being close to the north-west end of the Cymmer fault, and that of the Pentre shake of October 16, 1896, one-third of a mile on the downthrow side of the Dinas fault. From this and other evidence it may be concluded that the centres of the areas disturbed by the earth-shakes lie close to, and on the downthrow side of, known faults, and that the areas, when otherwise than circular, are elongated in directions which are very nearly parallel to those of the faults.

A fact of some importance is that some coal-seams have been worked out right up to the faults referred to. This is certainly the case at Pendleton and Barnsley, and probably also in the Rhondda Valley. Now, by the withdrawal of the coal (or, it may be, by the lowering of the water-level by pumping in other parts of the mines), the rock above is deprived to a great extent of its support, and tends to sink down and close up the worked-out seam. Nowhere can this tendency be greater than where the rock is severed by a fault. Here the sinking would take place by small fault-slips, each of which, by the friction resulting, would give rise to a rather strong shock. But, as the slip would only affect a small region of the fault, and would occur at a slight depth, the intensity of the shock would fade away rapidly from the centre; the disturbed area would be small and circular, or slightly elongated in form; the shock and sound would be of brief duration, and would closely resemble those produced by the fall of a heavy mass of rock.

If this be a correct explanation of the origin of the earth-shakes—and it seems to account for all their peculiar features—it follows that the shakes cannot be classed either with true or with spurious earthquakes. They are of natural origin in so far as they are produced by fault-slips, but of artificial origin in that the slips are precipitated by human labour and not by the gradual cooling and warping of the earth." —CHARLES DAVISON, *Mining Journal*, London, March 12, 1910, p. 310. (G. H. S.)

POSSIBILITIES OF RESCUE WORK IN COAL MINES. —"Dr. Haldane remarked at the outset that there had been a great deal of discussion recently on mine rescue apparatus and organisation of rescue stations and rescue corps. In his present contribution he intended to discuss the wider question of the possibilities of rescue work with or without rescue apparatus. He had come to the conclusion that immediately after a blast few, if any, of the men were already dead, and the great majority could be saved if they could only be protected from the action of after damp. To prove this the author recounted the circumstances of the recent explosion in the Derran Colliery, South Wales. There were three possible ways of protecting men from after damp, one was to get pure air to them, another to get them out of the after damp, and a third was to give them pure air to breathe until they are in the pure air again. If, as generally is the case, down cast and up cast shafts were close together, separation doors and air crossings would be destroyed so that air could not be got round in the ordinary way. If the workings were connected with any other pit it might be possible to clear out the after damp by opening the communicating door and closing the down cast shaft with planks. In many cases it may be found very desirable to reverse the ventilation, and the author thought that in any case means for reversing the ventilation ought always to be provided. In connection with rescue work, means of testing the air in the mine are very important, and for this purpose a mouse or a bird enclosed in a box capable of being shut air tight might be used, the box being capable of connection with the exhaust valve of a rescue apparatus in order that the animal could be revived at once if necessary. A rescue corps might also be provided with light subsidiary apparatus for fixing over the faces of disabled men in order to protect them until they could be moved, but this has not yet been properly considered. The lecturer also referred to the advisability of mine managers discussing with their staff what action should be taken in the event of an explosion."—Dr. J. S. HALDANE. *Colliery Guardian*, May 6, 1910. (T. D.)

MISCELLANEOUS.

AMBULANCE WORK ON THE GOLDFIELDS. —"Despite all precautionary methods adopted by the mining companies, the possibilities of accidents cannot fail to be great in such a large field of labour as Kalgoorlie, and nearly all persons connected with mining, sooner or later, become eye-witnesses of some form of accident. The question then arises—What can be done to alleviate the suffering of the injured one? and it is lamentable to reflect how very few mining men are capable of rendering 'first aid.' The needless suffering so frequently caused by the ignorance of those unskilled persons with whom the patient is first brought in contact is as undoubted as it is deplorable. By rough handling or even the mere want of the slightest knowledge of how to lift an injured limb, a simple fracture may be converted into a compound or even complicated one The method of arresting hemorrhage is easily learned yet many lives have been lost, the very life-blood ebbing away in the presence of sorrowing spectators, perfectly helpless because none amongst them had been taught the first rudiments of an ambulance pupil's work, viz., the application of an extemporised tourniquet.

It is scarcely necessary to multiply instances to demonstrate the incalculable value of a knowledge of 'first aid.' The object of this article is to draw attention to the work being carried on by the St. John Ambulance Association. About four years ago a centre of this Association was established on the goldfields, at which time, with the exception of a few obsolete stretchers on some of the big mines, practically no provision had been made for the proper handling of the injured. Since then casualty wards have been erected and equipped with up-to-date appliances, and on some of the mines a special ambulance corps is always at hand.

The casualty wards are fitted with electrically-heated beds, hot and cold water, medicine chests, splints, bandages, etc., and it is here that much can be done to relieve pain before the arrival of a doctor. The electrically-heated bed is a valuable acquisition to an accident ward, seeing that a man brought up from underground wet and cold can be placed between warm blankets immediately, if necessary. The beds are inexpensive and can be made on the mine.

The Association has given great attention to transport work and the mines are gradually becoming equipped with proper ambulance wagons or litters. But although a mine may have every up-to-date appliance all these are practically useless without a trained body of men to use them. To supply this want the Association has instructed a number of men from many of the mines, has proper training quarters for use of members, and from time to time lectures are given by medical men.

Although so much has been done there still remains the absolute necessity for all those in charge of a number of men, both on the surface and underground, being trained to render first aid. From statistics it is apparent that the majority of fractures that occur underground are of a simple nature and if those first on the scene of accident apply suitable splints, the possibility of a more serious fracture is prevented. The broken ends of the bone are so sharp that any attempt to remove the patient without first securing the limb in splints are likely to lead to serious injury. A good improvised method of splinting is to use parts of the side of the candle box or similar pieces of wood tied by means of handerchiefs, braces, string, etc. Extreme cleanliness should be exercised in regard to wounds of every description as by thoroughly washing and treating the wound with some antiseptic the possibility of blood-poisoning is considerably lessened. This applies not only to large, but to small wounds.

The pamphlets published by the Mines Department of W.A. and South Africa fully describe the proper methods to be adopted when men are overcome by gases underground. The suggestions made by the writers of these pamphlets have been carried out as far as possible, and no difficulty has been experienced in treating bad cases. It has been found that the electrically-heated tables in use are of great assistance in restoring patients suffering severely from shock. Too much precaution cannot be taken in cases of 'gassing' and the old idea of giving large quantities of alcoholic stimulants is indeed dangerous. The brandy bottle which was so much in evidence in the early days has given place to readily prepared doses of sal volatile.

Experiments have recently been made with two kinds of safety appliances for underground rescue work; these are the Vajen-Baden safety helmet and the Fleuss-Davis-Hall apparatus. The construction of the Vajen helmet is remarkably simple and it requires only a few moments explanation for one to thoroughly understand the working of it.

The great advantages of this helmet lie in the use of compressed air, instead of special cylinders of oxygen, as required by other similar appliances, and the entire absence of complicated mechanism. The air reservoir at the back of the helmet is first filled with air to 150 lb. pressure; the helmet is then placed over the head, which it entirely covers, and strapped tightly to the shoulders. By opening the valve on top of the reservoir, the air is forced through the supply-tube to an inside point directly above the nostrils, and in practice the supply is ample for one hour, at least, without causing the wearer any discomfort. The fresh air being constantly forced into the helmet creates an outward pressure, and the foul air escapes through the neck-gear and around the bottom, which is lined with absorbent lamb's wool. The side or ear-pieces have special sounding diaphragms which render the hearing perfectly distinct. A whistle attached in front is used for a call and is a convenient means of signalling at any desired time. The helmet furnishes full protection to the head from falling debris, as it is padded on top and reinforced with eight ribs over a double thickness of leather so that no ordinary blow can penetrate the skull. It weighs only 6 lb. There is no hose attachment which is liable to kink or break and thus impede the movements of the wearer. This helmet has been extensively used in mines in many parts of the world and has been most favourably reported upon.

With reference to the Fleuss-Davis-Hall apparatus, Mr. E. A. Mann, Government Analyst, says ;—'Its weight is considerable as compared with other similar appliances, but it is so well distributed and carried from the shoulders that one soon becomes accustomed to it. The rubber tubes may probably be found not to stand this climate well. The apparatus is complicated and has many parts which may get out of gear. The use of oxygen instead of compressed air is inconvenient. But on the other hand, it is cool and comfortable and can be used readily from the moment it is put on. The head is not confined in a helmet which makes observation readier. The provision of a gauge in front enables the operator to tell exactly what limit of time he has available for work at any moment. It has stood the test of long trial and was very favourably reported upon by the Royal Commission on Mines."

It will be seen from the foregoing that each of these rescue appliances has points in its favour, but owing to the intricate mechanism of the Fleuss, the Vajen helmet appears to be the more suitable for ordinary rescue work underground."—C. A. BOLTON. —*Monthly Journal of the Chamber of Mines of Western Australia*, Jan. 31, 1910, p. 387. (H. A. W.)

SWEDISH RADIUM.—"Work is now in full swing in the radium factory at Islinge Lidingo, Sweden. A short time ago the new large smelting plant was started and it is working very well. It is calculated to smelt one ton of ore per day, but as a matter of fact has been doing about 20 per cent. more There are thirty workmen employed in the factory. At present the most critical work being done is the production of radium concentrate, from which the pure radium will ultimately be extracted. The ore is obtained at Kolm-Bellingen where sixty miners are employed. It is expected that the annual production of radium will be four to five grammes, which is a large quantity compared with the yield of other lands. The value of radium is now 400,000 f. per gramme." *Chemical Trade Journal*, May 7, 1910. (T. D.)

THE ORIGIN OF SOUTH AFRICAN TIN DEPOSITS.—The following are the summary and conclusions of the valuable paper on "The Origin of South African Tin Deposits," read recently by Mr. R. Recknagel before the Geological Society of South Africa:—

"It has been shown that the pneumatolytic theory does not give a satisfactory explanation of the formation of tin ore deposits and that it is unsound in principle, as it rests on assumptions which are in part contrary to chemical and physical facts. The fact that cassiterite has been proved to exist as a primary accessory constituent of granite in many parts of the world where workable tin deposits occur is considered of the utmost importance and forms the starting point of the theory advanced. The source of the tin ore deposits is not the unknown "great depth," but the acid igneous rocks in which the deposits occur or which are actually or presumably in existence in close neighbourhood to the deposits. In normal cases tin is supposed to occur as a silicate within the acid rocks—in accordance with Sandberger's investigations. Contrary to Sandberger, and in accordance with the ascensionists, it is considered unlikely that a concentration of the metals from the silicates into workable deposits could be achieved in one process by lateral secretion. It is considered likely that the occurrence of cassiterite as a primary constituent of the acid rocks marks the first stage of an igneous concentration of metals. The metals which in the normal state are distributed as silicates in minute quantities throughout the igneous rock masses were, in certain cases, concentrated by magmatic differentiation into zones of the igneous rocks and solidified there as native metals, oxides, sulphides and arsenides. In the case of tin, the first product of concentration was almost invariably cassiterite, in rare cases only, stannite. Besides some metallic minerals, as oxides of iron and copper pyrites, certain minerals containing fluorine and boron, especially fluorite and tourmaline, are often found in the above-mentioned zones, which seems to indicate that these elements played an important part even in the first stage of igneous concentration. These impregnated zones constitute workable ore deposits only in exceptional cases. In the case of cassiterite impregnations, however, they are of the greatest importance, as they are the source of most of the workable tin deposits. Within these zones a further igneous concentration of metals seems to have gone on in certain instances by a further differentiation of the magma, the result of which was the formation of an extra-fluid magma—the residual magma which in consolidation formed "pegmatites." In the case of tin-bearing magmas, the following kinds of deposit are the results of this further concentration:—Tin-bearing pegmatites proper, lode-like pegmatitic zones, pockets, lenses and pipes of tin ore associated with minerals usually found in ordinary pegmatites. These deposits, on account of their irregularity in shape and distribution, are rarely the objects of mining operations on a large scale, in spite of their richness. One of the most remarkable exceptions is the pipe and lense occurrences of the Potgietersrust district. After the complete solidification of the rock masses, a further concentration of the metallic parts they contain can take place by means of hydrous solution only, by lateral secretion. The majority of the workable lodes have been formed by lateral secretion, either within the rock masses, which contained the metals in a less concentrated state, or within neighbouring rocks, which were connected with the former through fissures. The grade of concentration of metals, in lodes formed by lateral secretion, depends in the first instance on the grade of concentration of the metals in those parts of the rocks which are affected by lateral secretion. Workable lodes will be formed by lateral secretion mainly in those instances where the rock masses undergoing the leaching and concentrating action of moving solutions contain the metals in some concentrated state already, in igneous rocks where a concentration by differentiation of the magma has preceded the concentration by hydrous solutions. The formation of the South African tin deposits is due to magmatic differentiation only, or to the combined action of magmatic differentiation and lateral secretion. The following table shows the grouping of the main South African deposits according to their probable origin as indicated in this paper :—

SOUTH AFRICAN TIN DEPOSITS.

A.—Origin due to magmatic differentiation :	*Examples :*
I. of *Granite* :	
1. Inclusion or so called impregnations.	Groenfontein, Zwartkloof.
2. Segregation in situ.	
a. Irregular pockets, lenses, pipes, etc.	Zaaiplaats, Groenfontein, etc., Elandsfontein, Zwartkloof, Quaggasfontein, Vlaklaagte.
b. Vein-like bodies.	Groenvlei, Enkeldoorn.
3. Segregated masses removed from place of segregation—	
a. Pegmatite dykes.	Oshoek, Swaziland.
b. Aplite dykes.	Zululand.
c. Lodes in sedimentary formations.	Rooiberg lodes ?
II. of *Felsite* :	
1. Inclusions or impregnations.	?
2. Segregations in situ—	
a. Irregular pockets, nodules, lenses, pipes.	Doornhoek, Kromkloof.
b. Vein-like bodies.	Elandsfontein.
3. Segregated Masses removed from place of segregation, felsite dykes.	?
B.—Origin due to lateral secretion preceded by magmatic segregation :	
I. In granite :	
I. Lodes.	Vlaklaagte, Kuils River.
II. In sedimentary formation :	
1. Where the original seat of SnO_2 is *granite*—	
a. Lodes.	Rooiberg.
b. Bedded impregnations.	Rooiberg.
2. Where the original seat of SnO_2 is *felsite*—	
a. Lodes.	Doornhoek.
b. Stockworks in shale	Doornhoek.

—R. RECKNAGEL, *South African Mining Journal*, March 12, 1910, p. 25. (A. R.)

PREVENTION OF ANKYLOSTOMIASIS.—"The ankylostoma, or miner's worm, has been for over two years the subject of experiments at the College of Medicine, Newcastle, England, by Sir Thomas Oliver and Hermann Belger. The chief object was to find a way to prevent infection of wet and ill-ventilated mines. On January 21, 1910, Sir T. Oliver read a paper before the Society of Tropical Medicine. According to the statements in this paper, of the 41 disinfectants tried by Mr. Belger, the best was sulphate of iron. This costs $39 a ton. A one per cent. solution of it would cover 100,000 sq. yd. of floor 1 cm. deep. It would prevent the development of any eggs that might be present and kill larvæ within a day. Almost equally effective are cinders. Old cinders are better than fresh, because richer in sulphate and poorer in chlorides. Ankylostoma larvæ are often found in the grooves at the sides of main entries; these grooves should be filled with cinders. Cinders should be sprinkled round the sanitary pails, both above and below ground.

Other Disinfectants.—Sea water is a third possible disinfectant; it kills larvæ within an hour. Creosote also kills larvæ quickly. Hence, where the air is moist and there is no danger of fire, the lower end of props should be creosoted to a height of about half a yard. Ankylostoma larvæ are fond of climbing, and wooden props easily become reservoirs of them. Managers of collieries and brickfields in cold climates are apt to think that their winter secures them from the ankylostoma, but Oliver and Belger found that larvæ frozen for six days revived and were little the worse. Tenholt found larvæ in miner's worn boots, although these had not been in the mud in which the men had been working; this suggests that the larvæ may migrate in boots from one mine to another. The interval between the infection of a man (through his mouth or skin), and the appearance of eggs in his feces, is 30 to 35 days."—C. HARPUR.—*Engineering & Mining Journal*, May 7, 1910, p. 976. (A. R.)

SILICA STANDARDS OF LENGTH.—"Silica standards of length are quite a novelty. As mentioned in the recent annual report of the National Physical Laboratory, careful experiments have demonstrated that silica is superior to glass as to thermal hysteresis, the co-efficients per degree Centigrade being: Silica, from 0·5 to 1; Jena glass, 59^{III}, 2·7; Jena glass, 16^{III}, 3·5, all expressed in 10^6 units. The 1-m. silica standard of the National Physical Laboratory is a tube supplied by the Thermal Syndicate, 2 cm. in diameter, having half discs fused into its ends. The difficulty of getting good defining lines on the silica has been overcome. Ordinary scratches chipped off in a day or two, and could not be seen when the tube was immersed in water. The ends have, therefore, been platinized, but on the lower surface; the graduation is cut through the platinum film into the silica, and the protecting glass cover rests against the lower platinized surface. The reading from above is, therefore, not affected by any tilt in the cover glass." —*Metallurgical and Chemical Engineering*, May, 1910. (J. A. W.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

THE MINER'S GUIDE. By F. P. MENNELL, F.G.S. A Practical Handbook for Prospectors, Working

Miners, and Mining Men generally. Fully illustrated. 4s. (London: Gerrards & Co.)

"This clearly written little book is to be regarded as an elementary introduction to the principles underlying metalliferous mining in unexplored countries, and should be a useful guide to prospectors and those taking up mining without having any preliminary training. The origin and characters of ore-bodies are briefly discussed: methods for sampling, testing, prospecting, and easy tests of the recognition of minerals and precious stones are given. Since the book is practically a third edition of the author's "Rhodesian Miner's Handbook," many of the examples cited are chosen from South Africa, and those interested in the mineral prospects of the less known and undeveloped regions will find much useful information.—*London Mining Journal*, May 14, 1910. (A. R.)

The Valuation of Mineral Property. Rules and Tables. By T. A. O'Donahue, M.E., F.G.S. 7s. 6d. (London: Crosby Lockwood & Son.)

"In a paper upon the 'Valuation of Mineral Properties,' read in 1906 before the North of England Institute of Mining Engineers, Mr. O'Donahue contended that the published tables for discounting deferred values of mineral properties at high rates of interest were not based upon sound principles. That paper the author has expanded into the present work, and he has had the satisfaction of being informed by the authorities of the Estate Duty Office that he correctly gives the objections which they raise to certain tables, and that his rule embodies the principle for which they contend. The work contains numerous definitions, rules, and examples illustrating and explaining the principle advocated, and is furnished with an appendix of interest tables and of logarithms, with a particularly clear note as to the use and value of the latter calculations."—*London Mining Journal*, July 2, 1910. (A. R.)

Modern Coking Practice, including the Analysis of Materials and Products. By T. R. Byrom and J. E. Christopher. $3.50. The Norman W. Henley Publishing Company.

"This book, as is stated by the authors, is a handbook for those engaged in coke manufacture and the recovery of by-products. The volume is broad in its scope, treating the subjects of coal washing, classification of fuels, sampling and valuation of coal, coke, etc., as well as the science of coke manufacture. The illustrations showing various modern plants are excellent, and the drawings have been well selected and carefully executed. Not least in merit are the many useful tables, showing specific gravities, melting points, weights of materials, heat units, calorific values, etc."—*The Engineering and Mining Journal*, May 21, 1910. (W. A. C.)

Text Book of Ore Dressing. By Robert H. Richards, assisted by E. S. Bardwell and E. G. Goodwin. 16 × 24. Cloth. XI—702 pages. 354 illustrations. Price $5.00. McGraw-Hill Book Company, New York. 1909.

"The main subjects treated are as follows: General Principles. Part I.—Preliminary Breaking. Part II.—Final Crushing. Rolls, Gravity Stamps and Amalgamation, Grinders other than Gravity Stamps, Laws of Crushing. Part III.—Separating, Concentrating or Washing. Preliminary Washing and Hand Sorting, Preparation of the Crushed Ore for Concentration, Principles of Screen Sizing and Classifying, Coarse-Sand Concentrating, Fine-Sand Concentrating, Slime Concentrating, Miscellaneous Processes of Separation. Part IV.—Accessory Apparatus. Part V.—Mill Processes and Management. Management. Mill Principles and Processes, General Considerations. Part VI.—Coal Dressing.

Inseparable and indispensable as the industry of ore dressing is to other industries it constitutes a distinct branch of engineering art. As a study it may properly be classed as a separate branch of applied science. Between the subjects of mining and metallurgical engineering ore dressing is the connecting link, as indispensable to the one as to the other. No course either in mining or metallurgy is complete unless it deals very considerably with the preliminary treatment of ores.

Prof. Richards' book is the first comprehensive text book on ore dressing, and should arouse great interest, especially among teachers and students of this and allied subjects. It will be welcomed as coming from so eminent an authority and teacher. The author's successes in practice and in writing are too well known for recital here. The new book is a condensation of the author's well known treatise, 'Ore Dressing,' now filling four large volumes. It has the clearness, thoroughness and reliability of his former books without being burdened with details of practice. Much descriptive matter and data relating to machinery and practice in different localities is eliminated, and consequently the book will not soon be out of date. It would have been better, perhaps, if the process of elimination and condensation had gone further, resulting in still greater emphasis on the principles. While such a criticism would be made by some teachers, all will concede that the subject is treated thoroughly in both the academic and the technical sense. Expressed in a few words, Prof. Richards' book is a practical presentation of the principles and practice of ore dressing."—*The Chemical Engineer*, March, 1910, Vol. XI., No. 3. (W. A. C.)

The Cyanide Handbook. By J. E. Clennel, B.Sc (London), Associate of the Chemical, Metallurgical and Mining Society of South Africa; author of "Chemistry of Cyanide Solutions," etc., etc. Published by the McCraw-Hill Book Company, New York and London. 520 pp. 21s. nett.

A much larger share of attention than is usually given in works of this description is devoted to the chemical principles underlying the practice of the extraction of gold and silver by cyanide solutions. Almost half of the volume is occupied by the history, manufacture and reactions of cyanogen and its compounds, their analysis and behaviour in working solutions. The compounds noticed include cyanates, isocyanates, ferricyanides, manganicyanides and many other such combinations and many facts of interest to the scientific metallurgist, such as usually require a long and painful search through technical literature, are here for the first time brought together into one convenient work. This forms the principal reason why it deserves a place on the bookshelf of all who are interested in this branch of our industry. An outline only is given of practice in modern cyanide plants, and even this is by no means up to date, and such things as diaphragms, collecting tables, and the use of heavy stamps with coarse screening do not appear to have reached the author's notice. For details of Rand milling practice reference is made to papers written in 1903, and the use of the 'Alkali Wash' is stated to be still common practice. Very good chapters on assaying and sampling will be found, as also a useful section on analysis of bullion and other products. We note, however, that the author while condemning the use of bicarbonates as fluxes on page 335 admits their use in his own work on page 357, and quotes with approbation their use in many other instances. The work is well produced and free from typographical errors. The spelling is English with the exception of 'Niter,' which is consistently so spelt. A few tables of general interest are given, and the index provided seems well up to the mark. This work will take a high place in the ever increasing volume of cyanide literature, and we offer the author our hearty congratulations upon it. (H. A. W.)

Abstracts of Patent Applications.

(C.) 274/10. Sven Emil Sieurin. Process of producing metals.

This application refers to a method of treatment of metallic ores by heating them with a solid reducing material, such as cheap coal containing much ash, to such a temperature that the metal is reduced but not melted. In this way the metal is obtained free from contamination, and may be obtained by physical means from the reduction mixture. The addition of lime in alternate layers is proposed where sulphur is to be eliminated.

(C.) 278/10. M. P. Boss. Improvements in stamp mortars.

This application relates to certain variations in the forms of mortar box liners and in the arrangement of screens for same.

(C.) 279/10. H. A. Humphrey. Improvements in methods of raising or forcing liquids, and in apparatus therefor.

This invention relates to a method of raising or forcing water or other liquids by the energy developed by the periodic explosion or combustion of a combustible mixture and to apparatus for carrying out the method.

It is essential for the carrying out of the method that the apparatus should comprise a combustion chamber which is provided with an inlet valve for combustible mixture and an exhaust valve and communicates with a delivery or discharge pipe or duct leading to a place of higher level or higher pressure and of a length sufficient to ensure the acquisition by the moving liquid of the momentum necessary for the desired effects to be produced, the length depending principally upon the desired speed of operation.

(C.) 305/10. Janus Simpson Island. A process of extracting precious values from ores. 21.6.10.

This application relates to a chloridising process for the extraction of precious metals, in which chlorine and sulphur dioxide are separately generated and delivered to a chamber containing camphor (the precise purpose of which is not stated), from whence the combined gases are led into a converter containing pulverised ore, preferably heated, steam also being projected into the ore simultaneously. The soluble salts of the precious metals which are formed are subsequently leached out in separate vessels.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 339/10. Frank Humphris. Improvements relating to tooth driving gear. 14 7.10.

(C.) 340/10. Edward Jonah Wiggins. Improvements in the method of and apparatus for preparing liquid hydro-carbon for combustion. 14.7.10.

(C.) 341/10. Fred Bedford (1), Charles Edward Williams (2). Improvements in and connected with the catalytic synthesis of methane. 14.7.10.

(P.) 343/10. Gaston Jacquier. A new or improved alloy. 16.7.10.

(P.) 344/10. Victor T. Frolich. Corrugated metal rollers for use in connection with circular ropes used in mechanical haulage. 18.7.10.

(C.) 345/10. Thomas Henry Bradbury. Improvements in rock drill forging and sharpening machines 18.7.10.

(C.) 347/10. Hans Charles Behr. Improvements relating to bearings for stamp cam shafts. 19.7.10.

(P.) 348/10. Andrew Frederick Crosse. An improvement in the treatment of auriferous and argentiferous ores containing graphite and other carbonaceous material. 20.7.10.

(P.) 349/10. Edward Henry Johnson (1), Frances Allen (2), Hugh McMenamin (3). Improvements for feeding ore to the stamps of stamp mills. 21.7.10.

(P.) 351/10. William Cullen. Improvements in the treatment of cyanided residue. 21.7.10

(C.) 352/10. Sandys Stuart Macaskie. Improvements in or relating to superheaters. 22.7.10.

(C.) 353/10. Giovanni Cornaro. Improvements in the manufacture of explosives. 22.7.10.

(C.) 354/10. Charles Anderson Case. Improvements in a panning and concentrating machine. 22.7.10.

(P.) 355/10. Frederick George Fell (1), G. D. Hock (2). An improvement in the means of loosening cams and the like. 23.7.10.

(P.) 356/10. John Archibald Maclachlan (1), Andrew Trimble (2). Improved apparatus for ventilation of workings by means of vacuum. 23.7.10

(P.) 358/10. Arthur J. Dickens. An improved mechanism for elevating water or other liquids from a flowing stream. 25.7.10.

(C.) 359/10. Samuel John Herbert Wilkes (1), George Wilkes (2). Improvements in nut locks. 26.7.10.

(P.) 360/10. Edmond Erskine (1), Cecil Murray (2). Spiral tube scraper. 26.7.10.

(C.) 361/10. William Badger. A shark grip split, serrated, segmental, adjustable, eccentric, turned, self tightening, easy releasing, bolted driving cam for stamp batteries. 27.7.10.

(C.) 362/10. Ernest Albert Brinkman. Tube mill liner. 27.7.10.

(P.) 363/10. Hans Charles Behr. Improvements in the construction of stamp batteries. 27.7.10.

(P.) 364/10. Hermann Daetwyler. Improvements in fluid actuated boiler scaling tools. 27.7.10.

(P.) 365/10. Allan Thomas Cocking. Improvements in torches for fuse lighting and analogous purposes. 28.7.10.

(C.) 366/10. Arthur Merriam Wight. Improvements in collapsible boxes. 29.7.10.

(C.) 367/10. William Woodward Robacher. Improvements in or relating to filtering apparatus particularly intended for the cyanide process of extracting precious metals from their ores. 29.7.10.

(C.) 368/10. William Woodward Robacher. Improvements in or relating to filtering apparatus particularly intended for the cyanide process of extracting precious metals from their ores. 29.7.10,

(C.) 369/10. Pedro Pingjauer. Multiple rotary steam engine. 29.7.10.

(C.) 370/10. Lewis Condict Hayles. Improvements in combined throttle valves and oilers. 29.7.10.

(C.) 371/10. Albert Henry Midgely. Improvements in dynamo electric machinery. 29.7.10.

(C.) 372/10. Arthur Francis Berry. Improvements in or relating to apparatus for use in the distribution of alternating electric current. 30.7.10.

(C.) 374/10. James Alexander Borbour. An improvement in valves, taps, and articles of a like nature. 3.8.10.

(C.) 376/10. William Banks (1), Alexander Warrell (2). A new or improved compound of substances for use as a packing and analogous purposes. 3.8.10.

(C.) 377/10. George Alfred Julius. Improvements in and relating to apparatus for printing and issuing tickets or checks of different denominations and for registering and totalling numbers and indicating the totals. 4.8.10.

(C.) 378/10. Kenneth Bertram Lamont (1), Arthur Henry Winlerton (2), Gordon Travers Easton (3). A mechanical device for separating the fines from the rock before entering mortar box. 4.8.10.

(C.) 379/10. Ivor Scott Winby. Improvements in or relating to devices for securing the rails of permanent ways or similar tracks. 5.8.10.

(C.) 380/10. Dr. August Volker. A new or improved method of forming bodies from semiliquid quartz and like material. 5.8.10.

(C.) 381/10. Alom Ward Ambler. Improvements in means for propelling vehicles, implements an the like. 5.8.10.

(C.) 383/10. Frank George Symmonds Price. Improvements in apparatus for mixing and consolidating concrete and like materials. 5.8.10.

(C.) 384/10. Hans Peter Hansen. Improvements in and relating to the manufacture of pipes, tubes and the like from wood and in apparatus therefor. 5.8.10.

(P.) 386/10. Reginald Shadforth Scott. Improvements for linings for tube and similar grinding mills. 6.8.10.

(P.) 387/10. Junius Hamilton Anstruther Macadam. Improvements in pebble catching devices and like mills. 6.8.10.

(P.) 388/10. Junius Ford Cook (1), Arthur Frederick Lord (2). Improvements in the construction of gravity stamp batteries. 6.8.10.

(P.) 389/10. Ernest Alfred Newton. An improved spitzkasten for the treatment of ores and the like. 6.8.10.

(C.) 391/10. Morley Punshon Reynolds. An improved metallic screen or sieve. 9.8.10.

(P.) 392/10. William Cullagher. Improvements applicable to rotary deep boring apparatus. 10.8.10.

(P.) 393/10. William Cochran Boyd. Improvements in rollers. 10.8.10.

(C.) 395/10. William James Barnett. Improvements in acetylene generators. 12.8.10.

(P.) 396/10. Charles William Dowsett (1), Henry Saunders Kenny (2). Improvements in ore concentration and means therefor. 12.8.10.

(P.) 397/10. William Rundle (1), John Main Ellen (2). Improvements in railway vehicle couplings. 12.8.10.

(P.) 398/10. Arthur Chicele Plowden (1), George Taylor Philip (2). Improvement relating to coupling (automatic) for railway trucks and the like. 13.8.10.

(P.) 399/10. Francis Allen (1), Gordon Travers Euston (2), Hugh McMenamin (3). Improvements in driving means for ore feeders. 13.8.10.

(P.) 400/10. Francis Allen (1), Gordon Travers Euston (2), Hugh McMenamin (3). Improvements in means for feeding ore to the stamps of stamp mills. 13.8.10.

(C.) 401/10. Julius Stromeyer. Improvements in and relating to brake shoes. 13.8.10.

(C.) 402/10. Joen George Inshaw (1), George Richard Inshaw (2). Improvements in apparatus for pilgering metal blooms, tubes or rods. 13.8.10.

(P.) 403/10. Reuben Jones. Improvements in stamp tappets. 17.8.10.

(C.) 406/10. Edmund Scott Gustave Rees. Improvements in ejectors, condensers, and air pumps or compressors. 19.8.10.

(C.) 407/10. Ciro Fidel Mendez. Improvements in and to relating compositions for use as a building material paint and the like. 19.8.10.

(C.) 408/10. Charles Walker. Improvements in and relating to linings for pits, shafts, and the like. 19.8.10.

(C.) 409/10. Carl Giesecke. Process for dressing ores. 19.8.10.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

BLEWITT, E., c/o West Nicholson, *to* Wolfshall Mine, Selukwe, Rhodesia.

CROPPER, C. H., *l/o* Mexico; c/o Lucky Chance Mine, Dubbo, via Narazuta, Banchi Province, Northern Nigeria, via Tokoja.

IMPEY, R. L., *to* P. O. Box 108, Germiston.

LARET, H., *l/o* Bulawayo; Planet Mine, P. O. Arcturus, Salisbury, Rhodesia.

LEYSON, WM., *l/o* Johannesburg; Monomotapa Concessions, Ltd., M'Toko, via Salisbury, Rhodesia.

LICHTENSTEIN, A., *l/o* Benoni; c/o Dr. Simmons, Main Street, Belgravia, Johannesburg.

PIASECKI, S., *l/o* Nigel; City and Suburban G. M. and Estate Co., Ltd., P. O. Box 1026, Johannesburg.

SPANDAU, H. J., *l/o* Johannesburg; Elandsdrift G. M. Co., P. O. Sabie.

TRUSCOTT, S. J., *to* 43, Threadneedle Street, London, E.C.

WARD, H., *l/o* Roodepoort; West Rand Consolidated Mines, Ltd., P. O. Box 38, Krugersdorp.

WARTENWEILER, FRED., *to* P. O. Box 2476, Johannesburg.

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OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

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VOL. XI. SEPTEMBER, 1910. No. 3.

Proceedings
AT
Ordinary General Meeting, September 17, 1910.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the South African School of Mines, on Saturday evening, September 17th. Prof. G. H. Stanley (Vice-President) in the chair. There were also present:

43 Members: Messrs. F. F. Alexander, R. Allen, Tom Johnson, A. Richardson, H. A. White, Prof. J. A. Wilkinson, J. Littlejohn, A. F. Crosse, E. H. Johnson, H. R Adams, E. Blume, B. V. Blundun, A. J. Bowness, F. W. Cindel, M. H Coombe, J. Daniel, R. Gascoyne, J. Gray, A. B. Inglis, J. H. Johnson, G. A Lawson, H. J. Lea, J. Lea, G. Melvill, J. T. Mitchell, S. Morison, W. Nicklin, F. D. Phillips, W. S. V. Price, J. F. Pyles, G. A. Robertson, H. Ross, G. H. Smith, T. F. Smith, B. C. T Solly, J. A. Taylor, A. Thomas, C. F. Thomas, O. Tonnesen, Chris. Toombs, A Wilkinson and L. J. Wilmoth.

17 Associates and Students: Messrs. A. R. Adams, C. A. Damant, F. E. Doble. B. Hasseriis, B. W. Holman, W. J. R. Hunter, A. King, G. W. Leach, R. Lindhorst, G. H. Little, H. Lomberg, H. J. v. d. Merwe, F. J. Pooler, R. E. Sawyer, G. L. Shearer, H. Stadler, and L. A. Womble.

12 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as printed in the August *Journal*, were confirmed.

NEW MEMBERS.

Messrs. J. Gray and Andrew F. Crosse were appointed scrutineers, and after their scrutiny of the ballot papers, the Chairman announced that all the candidates for membership had been unanimously elected, as follows:—

ALLEN, VIVIAN STEVENS, P. O. Box 369, Salisbury, Rhodesia. Mining Engineer.
BRODIGAN, VINCENT FREDERIC, East Rand Proprietary Mines, Ltd., P. O. Box 24, East Rand. Mining Engineer.
JONES, WILLIAM, P. O. Box 42, Randfontein. Mine Captain.
ODGERS, WILLIAM HEARLE, Village Deep, Ltd., P. O. Box 1064, Johannesburg. Cyanide Foreman. *(Transfer from Associate Roll.)*
SCHWARZ, ADOLF, Meyer and Charlton G. M. Co., Ltd., 10, Janie Street, Jeppestown, Johannesburg. Cyanider.
STEVENS, THOMAS BARNBROOK, Oroya Links, Ltd., P. O. Box 57, Kalgoorlie, W. Australia. Metallurgist. *(Transfer from Associate Roll.)*
TRIGGS, JAMES THOMAS, Village Deep, Ltd., P. O. Box 1064, Johannesburg. Cyanider.

The Secretary: Since the last meeting of the Society the following have been admitted by the Council:—

As Associates.—

BRINSDEN, FREDERICK GEORGE, Kalgurli G. M., Ltd., Fimiston, West Australia. Metallurgist.
THORNE, THOMAS LLEWELLYN, Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Cyanider. (*Transfer from Student Roll*).
TUCKER, PERCY ALEXANDER. Consolidated Gold Fields Laboratory, P. O. Box 108, Germiston. Metallurgical Chemist.

GENERAL BUSINESS.

PRESIDENT INDISPOSED.

The Chairman: I am sure you will be sorry to hear that Dr. Moir is not very well to-night, and is therefore unable to take the chair. Consequently it falls upon me, as the only Vice-President present, to occupy that position.

THE DISTRIBUTION OF PULP FOR TUBE MILLING.

By G. A. ROBERTSON (Member).

Mr. H. Stadler in reply to the discussion before another technical society on his paper, "The Computation of the Crushing Efficiency of Fine Grinding Machines," stated with reference to tube mills that there was still "ample room for further great improvements, especially with the regulating and maintaining of the recognised most suitable conditions of the feeding." I do not know what the author had in his mind at that time, but no doubt during his investigations into tube mill efficiency on behalf of the Mines Trials Committee he found room for a more even distribution of the coarse products. If the tube mill is most efficient on particles of ore up to 0·27 in. diameter it is only reasonable to suppose that in order to get maximum efficiency from a battery of tube mills that each tube mill should have an equal share of these coarse particles or its *pro rata* share in accordance with its grinding capacity for the time being.

Whilst various methods might be employed for distributing the pulp between tube mills, including the method suggested by Mr. J. E. Thomas,* none of these can be considered satisfactory under all operative conditions, and the difficulty becomes more apparent when, say, from one point we may have to divide a pulp equally between from 10 to 15 tube mills. In order to overcome these difficulties the writer has prepared the accompanying sketches, which will suffice to show the practical utility of the scheme herewith advocated. In cone classification for tube milling we may not get an equal grading analysis from every foot of the cone perimeter, but such grading is much more equal when the underflow is shut off, and taking the necessary precautions with regard to the vertical inflow, etc., there is no reason why an equal grading of pulp at equal distances on a cone perimeter should not be an accomplished fact.

If the pulp for distribution were elevated by a tailings wheel, naturally such pulp would have to be given a vertical fall into the pulp distributor; on the other hand, however, where a pump acted as the elevator it might be convenient to run the pump column up through the cone, the column discharging itself above the cone overflow level. This latter method would be adaptable to diaphragm cone classification as advocated by Mr. G. O. Smart,* which, in new designs, effects a considerable saving in height of elevation of pulp.

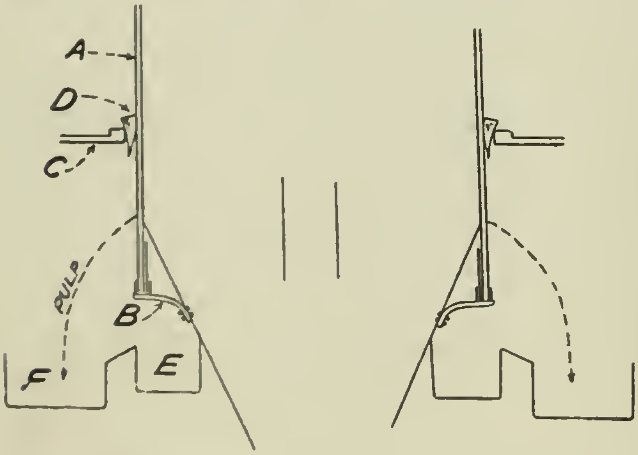

FIG. I.—Elevation of Cone Pulp Distributor.

A—Pulp dividers made of wood. B—Channel iron bracket (with guide plate on inside) fixed around cone. C—Circular platform. D—Wood wedge. E.—Spill launder. F.—Peripheral launder common to the overflows from cone. G.—Pulp inflow from launder.

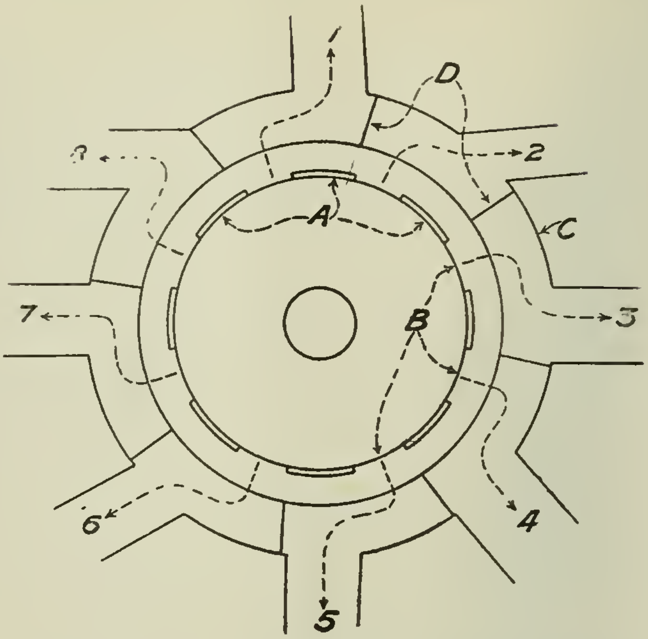

FIG. II.

A—Dividers in position. B—Divided overflows feeding into C—Common launder. D—Guide gates.

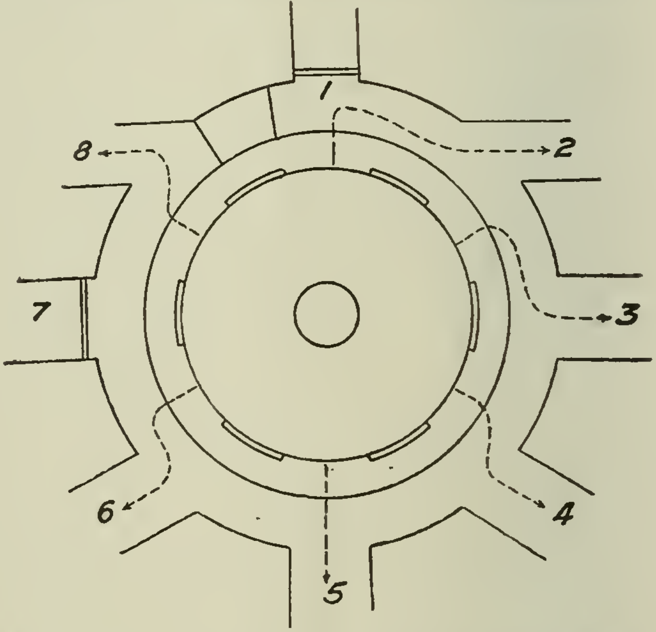

FIG. III.—Shows that only six tubes are in action, Nos. 1 and 7 being idle and overflows again divided.

Since the preparation of these sketches the writer has seen illustrations of another pulp distributor†, one objection to that device being that it cannot divide pulp equally if one or more tube mills were to lie idle, but with a few alterations in the design it could also be made suitable to all conditions.

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* See discussion G. O. Smart, "The Tube Mill Circuit and Classification," June *Journal*, p. 153.

* See this *Journal*, February, p. 282.

† See this *Journal*, July, p. 38.

By this method any number of tube mills can be in or out of action, and the pulp evenly divided amongst the tube mills running.

Mr. Fraser Alexander (*Member of Council*): I should like to propose a hearty vote of thanks to Mr. Robertson for his valuable though short paper. I am sure we all feel very much the want of an adequate distributor leading to the various tube mills. The author does not say whether this apparatus shown in the diagrams is actually at work and doing well, and where it can be inspected. I should like to ask him whether it is entirely theoretical or in actual operation? I think it will be agreed that it is a step in the right direction, as it is a point on which we often meet with a great deal of trouble.

Mr. G. A. Robertson (*Member*): The idea is entirely theoretical. At the same time it is just about guaranteed in practice.

The Chairman: I have pleasure in seconding this vote of thanks. It is one of those little papers which are very useful, and if it helps to solve this practical difficulty of the final division of pulp, the Society and those members supervising the running of tube mills will be under some indebtedness to the author. I do not quite follow personally the course of the pulp through this distributor. I do not see how it gets into this particular device. Perhaps the author could illustrate it on the blackboard?

Mr. G. A. Robertson: The pulp has a clear fall into the common launder from whence it then finds its outlet. You know that with a conical classifier if a piece of wood be placed against the overflow the pulp is divided, so in this case the inflow to the distributor is down the vertical pipe as shown in Fig. I., then between the dividers. It follows, of course, that if several tube mills adjacent to one another are out of action that a few short portable launders would be necessary to receive the pulp and convey it to the outlets on the common launder. Broader dividers would have to be kept in stock in order to meet these requirements.

The Chairman: I think the matter is made clear now.

NOTES ON BATTERY PRACTICE.

By A. R. Stacpoole (Member).

Mr. H. A. White (*Member of Council*) read the following paper in the absence of the author:

The mill manager and amalgamator are rapidly having their power and responsibility usurped by the reduction officer and cyanide manager, but have only themselves to blame for it.

From the early days of the Rand the mill man looked on it as his first and almost only duty to crush, and rush a big tonnage through, paying little or no attention to the extraction of gold; the idea being, that what is not caught in the mill will be caught in the cyanide works. Had the amalgamator looked on the matter of extraction as of equal importance with that of crushing, he would be in a much stronger position to-day.

The first and principal duty of every amalgamator should be *to see that no free gold, amalgam or mercury goes over the end of the amalgam plates.* As a rule the free gold and amalgam which finds its way to the cyanide works requires a much stronger solution and a longer treatment to dissolve it than it gets, so that a large portion of it goes to the dump thereby enriching the residue.

Collection of Sand from Amalgam Plates.—This sand is commonly known as "black sand." The clumsy and imperfect method used in collecting the sand calls for a better and more careful way. In many mills the method used to-day to collect plate sand is as follows:—

The water is shut off, and stamps hung up; then a very imperfect dam is made across the foot of the plate with brushes; the sand is hosed down till stopped by the brushes, but it frequently happens that the flow of water is so great that a large quantity of sand is washed over them, as well as a considerable quantity escaping between and under the brushes, so that probably the quantity of sand which is washed off the plate is equal to the quantity collected. If 50% of the plate sand is worth collecting and treating in a special way, why not catch the other 50% also? To collect the plate sand is a simple operation, and can be done in the following manner:—First hang up the stamps, shutting off the water at the same time; then place a "collecting trough" under the lip and at the foot of the plate, and hose the sand down into the "collecting trough." Care and judgment should be exercised as to the quantity of water used, so that the sand will not be washed out of the "collecting trough." In dressing the plate a quantity of sand is loosened from it. This necessitates the plate being hosed down a second time before the "collecting trough" is removed. When steaming and scouring plates the trough should also be used. Sheet iron would be a good material to construct the "collecting trough" with, as the particles of amalgam which are hosed down from the plate will not stick to it. The collecting trough should run the full width of the plate, and be constructed with an overflow pipe. Fig. I. shows the trough. "A" marks the overflow pipe. The trough should rest on two brackets, and when in position the inside wall of the trough should fit close against and under the lip of the plate, the overflow pipe protruding through a slot in the plate launder. This slot could be closed by a simple and effective shutter arrangement, so that the pulp leaving the plate would not splash through it. The overflow pipe should discharge into a bucket placed on the mercury trap. By this means any sand washed from the collecting launder would be retained in the bucket. Fig. II. shows the collecting launder in position. This arrangement would necessitate a slot being cut in the plate launder. The supporting brackets for the trough should be so arranged as to give strength to the side of the plate launder, where it was weakened by cutting the slot in it.

To remove the sand quickly and cleanly from the collecting trough, it will be necessary to have an iron cylindrical tank full of water, about 4 ft. × 1 ft. 3 in. would be sufficient. It should be suspended on a small carriage in the same manner as a truck is. It could be easily wheeled from plate to plate. When the "collecting trough" is removed from the foot of a plate, the sand would be washed easily and quickly from it by dipping it up and down in the tank two or three times. There should be two collecting troughs to prevent delay. In order that no sand shall escape at the corners of the plate and over the ends of collecting trough, two small angular blocks of wood 8 in. long running from 4 in. wide to half an inch at the point, and $1\frac{1}{2}$ in. thick should be pressed tightly on the plate, and screwed to the side of the table. These blocks would be permanent fixtures.

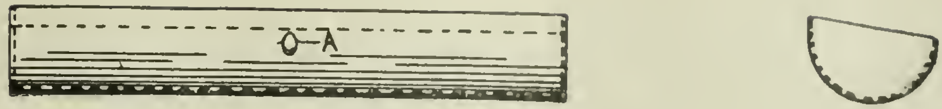
Carrying Amalgam Box, Brushes and Tools from Plate to Plate.—These should be carried on a small trolley table fitted with a sheet iron or copper plate top, and a launder at foot to catch the drainings from the table. If this trolley were supplied to the amalgamator when dressing and scraping plates, it would prevent all loss from dripping brushes, which are full of amalgam, and are frequently carried around in the hand. The trolley should be small and handy, and as high as the amalgam table.

Changing Screens, etc.—When changing screens, working at shoes and dies, etc., there is always far too much scratching and tearing up of the plate and amalgam by throwing tools, screen pins, pulling coarse gravel out of mortar box, etc., on to the plate. This should all be avoided, and can be with a little care. Before opening a mortar box a rubber apron made from insertion should be laid down first over the plate. There should be two aprons kept, one small and one large: the small apron to be used when changing screens, and the large one when working on heads, shoes, dies, etc. The apron should be laid down before the shoe and die board is put on the table. A roller should be attached to each apron, so that it could be rolled up after use.

The gravel or sand which escapes from the mortar box, when the box is open, should not be permitted to come in contact with the plate. Two important points are achieved by preventing this: first, the plate and amalgam are not cut up; second, it will not be necessary to take the gravel to the amalgam barrel, as it has not come in contact with amalgam or mercury. The gravel and sand can be returned direct to the mortar box, thereby reducing by at least one half the quantity of gravel and sand, which the amalgam barrels are called on to deal with. The rubber aprons should be made to fit close up to the lips of the mortar box, and close to the sides of the table. There should be also a shallow iron tray, so made that it will fit close to and against the lip of the mortar box, and run along the full length of box. The tray should have several drainage holes in it.

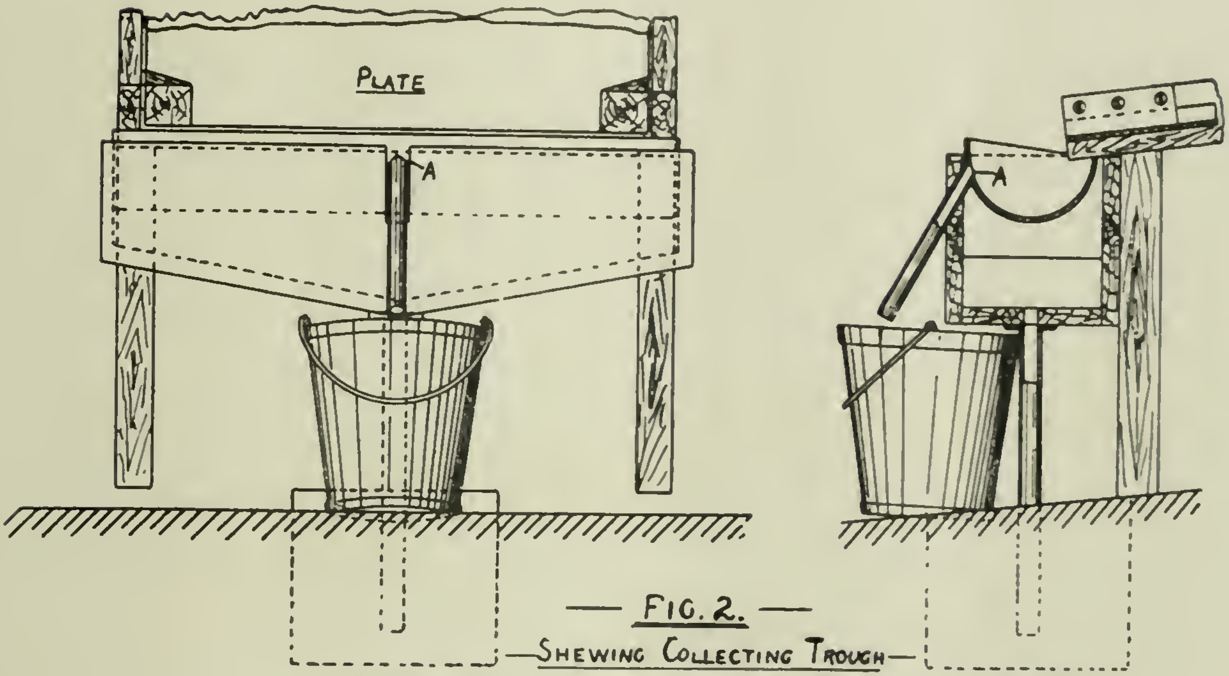
To Change a Screen.—When the stamps are hung up, and water shut off, lay down the smaller of the two rubber aprons, then put the tray in position. The screen pins, splash board, screen and tools can all be laid on the apron without damaging the plate or amalgam. When hosing the gravel off the sill or lip of the mortar box, the tray should be held tightly in position, so that it will catch the gravel and coarse sand. The new screen having been adjusted, the apron should be swept, and the sweepings put on the tray. The contents of tray may be returned to mortar box at once.

Bad Arrangement in "Wash Up" and Amalgam Rooms.—The tanks, bins, boxes, and tubs in which coarse gravel, sand washed from the amalgam, sand taken from mercury traps, and plate sand, are stored until they can be dealt with in the amalgam barrel, are in most cases badly arranged, which entails a very considerable amount of labour, as well as loss in handling. These storages should all be so placed that once the sand has been deposited in them, it could be fed automatically (or nearly so) and directly into the amalgam barrels. The amalgam barrels should be so situated that large storage tanks could be placed directly over them; into these storages the plate sand, and other by-products which are to go through the amalgam barrels should be put, also all by-products from the battery manager's wash-up table. They should be delivered direct into the storage tanks. This arrangement would obviate the necessity of carrying tons of sand in buckets from one place to the other. When this sand has passed through the amalgam barrels, it is again all carried in buckets to a battery specially set apart



— ELEVATION — — FIG I — — END ELEVATION —

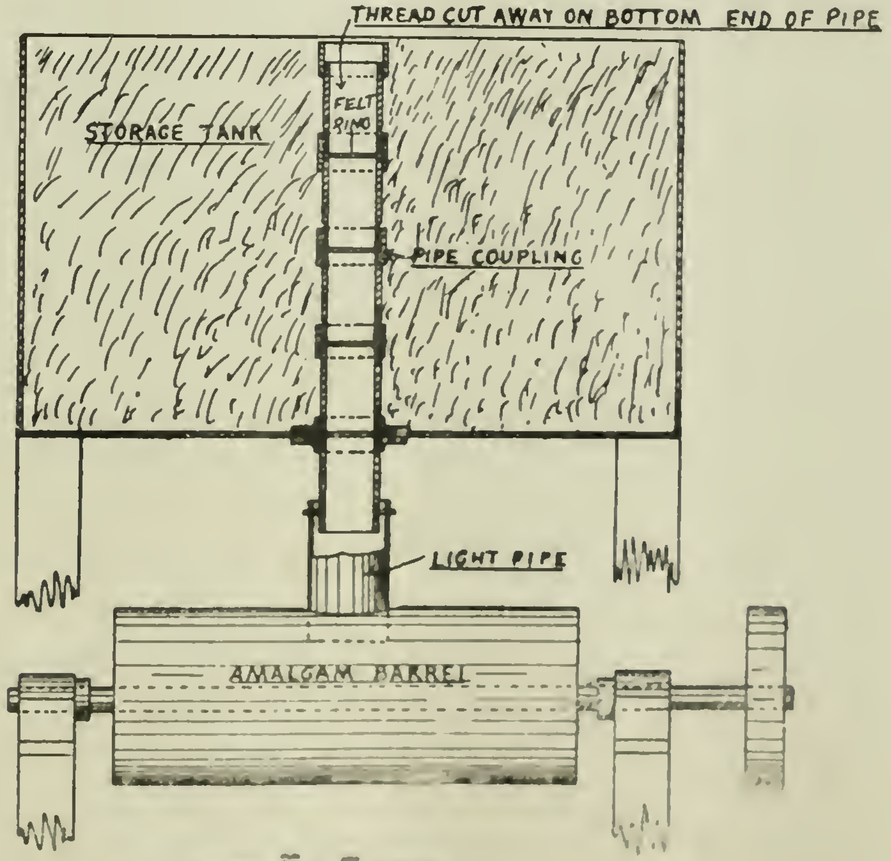
— COLLECTING TROUGH —



— FIG. 2. —

— SHEWING COLLECTING TROUGH —

— ELEVATION — — IN POSITION — — SECTION —



— FIG. 3. —

for it. Instead of bucketting it back to the battery, it should be transferred by some mechanical arrangement, but a better arrangement would be a light battery of three or five stamps, so placed that it would receive the residue from the amalgam barrels direct. The pulp should pass from the plate of this battery into the main tailings launder going to the cyanide works. A simple and good arrangement for feeding the sand from the storage tanks into the amalgam barrels is: To fit each storage with an 8 in. pipe, passing through the centre of its bottom, say 12 in. long on the outside, and 6 in. high in the inside. The inside pipe should be built up of 12 in. lengths till the pipe is as high as the top of the storage, each length of pipe to slip down over the other a distance of 2 in., and to have a felt washer joint. From the end of the pipe should be attached a detachable pipe leading into the amalgam barrel. When it is required to fill the barrel the outside detachable pipe would be placed in position, and the sand shovelled down the pipe. The sand having been lowered a foot in the storage tank, the top length of pipe should be removed, and so on. This would make the discharging easier and quicker. Fig. III. shows storage tank.

There are often by-products passed through the amalgam barrel which are most detrimental to the extraction in the cyanide works. It is a common practice to burn old brushes which have been used in dressing plates, old screen frames, etc., and collect the ash and half charred remains, and put them through the amalgam barrels. Charcoal or ash of any kind should on no account be sent to the cyanide works, nor should old plumbago pots. This description of by-product should be handed to the cyanide manager who can make some return to the battery manager for it. It can be got rid of in the lead bullion furnace to good advantage.

Mill men should join more generally in the discussions on tube mills, and help to solve the knotty question as to their suitability and efficiency. Another point which gives very large scope for debate is the removal of the amalgam plates from immediately in front of the mortar box. Although the plates are being taken out of the charge of the mill man, yet he sits quiet without raising a single point, either in favour of the new scheme or otherwise. Personally, I should not be surprised if it were found to be a mistake to have the amalgam plates away from the mortar boxes, as the free gold will be sure to concentrate in the cement launders, and in every corner it meets on its way to the amalgam table.

Let me conclude with one word of advice to all mill men and that is, to study their extraction, and work as if there was not a cyanide works behind them.

Mr. E. H. Johnson (*Past-President*): In proposing a vote of thanks to the author for his interesting little paper on various stamp mill accessories, I think he has somewhat misunderstood the purpose of removing the plates from the front of the mortar box. Where very high stamp duties are being obtained by the use of coarse screens, the erosion of amalgam by the coarse grit passing over the plates precludes the possibility of successful amalgamation. Hence the relegation of the plates to the tube mill circuit, where the ore is sufficiently comminuted to obtain good amalgamation results.

Mr. Andrew F. Crosse (*Past-President*): I have much pleasure in seconding the vote of thanks, though I am afraid that some of our members will consider the loss of amalgam mentioned is much over-estimated.

Mr. F. W. Cindel (*Member*): I should say the best thing in this paper is the first paragraph, but as I have not studied the paper I shall reserve any remarks for a future meeting.

The Chairman: It is the first paragraph which strikes me. It is suggested that the mill manager and amalgamator are not looking after their extraction. I think they are, but we do not hear enough from them. Mill men seldom join in our discussions, whilst the cyanide men are chasing each other round and arguing the point all the time. I think if the mill men would make themselves heard a little more, they would get on a little more. It is generally the cyanide men who rise to be reduction officers, and I think that is generally due to the fact that they discuss these matters publicly, and people know what they are thinking about.

Mr. Fraser Alexander (*Member of Council*): I cannot agree that this paper is really representative of common battery practice on the Rand, and, to say the least, I think it is rather obsolete but there are a good many points dealt with in the illustrations, which may give men who are not mill men and conversant with the common practice here, some idea how to handle amalgam, black sand and mercury. I do not entirely agree that the cyanide manager or reduction officer usurps the rights of the mill men. There are a great number of very capable mill men but they are somewhat shy and diffident, and as milling often causes a slight deafness, they do not always hear interesting remarks applicable to their case, and to accuse them of not being alive to the position is hardly fair. As regards the conveying of brushes, amalgam, and tools, the author must have been on a very large mill where so much amalgam was collected that it could not be conveniently carried, therefore necessitating a trolley. In fact he might have added a horse to pull it. I am sure mill men will be able to defend themselves quite ably in this matter.

Mr. J. H. Johnson (*Member*): With reference to the author's remark as to rushing a big tonnage through, he is evidently not speaking of batteries where tube mills are working. Now that tube mills have come in they do away with many things of which he speaks. But with batteries where no tube mills are working there is a great idea of getting through a big tonnage and trying to rush the products in the rest of the reduction plant, and the total extraction has to suffer. The mill manager is not always to blame for this as he is very often crushing according to instructions. As regards the handling of tools, etc., I have seen things happening in mills which were not altogether an advantage. Some amalgamators seem to think that once the amalgam is on the plates it requires a charge of dynamite to get it off, except when scraping.

Mr. J F. Pyles (*Member*): The author, like many others who have dared to criticise Rand practice, seems to have stirred up a hornet's nest. While his statements are rather sweeping, I do not believe all amalgamators or mill men are above criticism. From personal experience I know there is a great difference in the results obtained by different mill men under identical conditions. Wherein does this difference lie if not in the dressing of plates and the handling of amalgam? Mill men tell of amalgam recovered from plate-floor sweepings. How does amalgam get on to the floor? Then one sees a modern plate house the floor of which would not hold shelled peas. Does amalgam only find its way on to tight floors? I was almost convinced at one time that amalgam lined tube mill launders were unavoidable. I have since found that this is not true; that there need not be any more amalgam in the tube mill circuit launders than there was formerly in launders leading to the cyanide works. It is simply a question of men.

Dr. W. A. Caldecott (*Past-President*): The policy of entrusting the whole ore treatment, both crushing and cyaniding, to a reduction works manager, was adopted by the Consolidated Gold Fields some years ago. I am glad to say that this system, now quite common on the Rand, offers more responsible and lucrative positions to metallurgical workers on mining companies than was possible with the previous separate battery and cyanide managers. It has in some measure rendered the prospective advantages of promotion and status for surface workers comparable with those obtaining underground.

I trust that Mr. Alexander will supplement his remarks upon this paper by some constructive criticism as well, and give us the benefit of his long experience by describing better devices, if possible, for the purpose of carrying out the various necessary operations brought forward for discussion by the author.

SOME EXPERIMENTS ON SMELTING TITANIFEROUS IRON ORE.

(*Read at November Meeting, 1909, and April Meeting, 1910.*)

By Prof. G. H. Stanley (Member of Council).

REPLY TO DISCUSSION.

Prof. G. H. Stanley (*Vice-President*): I was very glad to hear from Mr. Harbord that he does not share the prejudice with which the use of titaniferous ores, even in small proportions, is viewed in many quarters, and though cautious, and rightly so, with regard to the use of an all titaniferous charge, he evidently preserves an open mind.

So far as I know, no further results of blast furnace smelting of titaniferous ores since Rossi's have been published, but I am able to infer that something is being done in this direction in the U.S.A., as Dr. Waterhouse, of the Lackawanna Steel Co., in writing me with reference to my experiments, mentions that in working on titaniferous ore (no locality stated) the furnace linings give trouble through corrosion, and suggests that the cause may be the basic nature of the slag due to a lower oxide of titanium being formed according to the view I brought forward.

Mr. Crosse shows that when the local ores are worked, as I feel assured they will be at some future time, the local smelters will at any rate have a purer and more likely ore to work on than the buyers of the ores which he analysed.

Mr. Adair's contribution is of great interest, and I certainly agree with him that the physical condition of iron ores has a very important bearing on their ease of reduction. His observations show that ilmenite is much more difficultly reducible than hematite, and the second set of experiments at Pretoria also bear this out and point to the fact that the iron in ilmenite reaches the bosh unreduced, and needs the intense reducing conditions there to effect its reduction, but without enabling it, or giving it time, to take up carbon, or that the oxidising effect of the TiO_2—

itself being reduced—uses up carbon which might otherwise have been absorbed by the iron ; so that his suggestion that titanium interferes chiefly by preventing carburisation appears likely to be near the truth, since if it prevents carburisation it would also tend to prevent reduction : later experiments show that the supposed infusibility of titaniferous slags does not of necessity exist.

I do not think that the silicious slag I proposed would need any special lining if the iron could be reduced out of the slag, as I suppose the lower oxide of titanium to act as base and thus satisfy the affinity of the silica; the corrosion experienced may easily be attributed to the ferrous oxide in the slag. Even if it were necessary to run a basic slag to secure fusibility (whether due to excess TiO_2 in the charge or other base) a more basic lining would be called for rather than clay with added quartz.

In reply to Mr. Greig, I admit at once that he has put his finger on a weak point with regard to the fusibility of most of the slags, *i.e.*, the presence of ferrous oxide as a probable cause ; but it was largely because of this that I made the platinum crucible experiments with pure TiO_2 as a component, and showed that the slags could still be fusible and fluid with scarcely any iron oxide at all present.

The question of fuel was not dealt with more fully as being somewhat outside the scope of the paper, which was only intended to deal with the experiments carried out ; but I may say that both before and since I have made a good many examinations of both Natal and Transvaal coals with a view to determine their coking possibilities, and have obtained some quite good results. I will not go so far as to say that it would be possible to produce coke here of an equal average grade to Durham coke, but coke not greatly inferior with ash not much exceeding 10% and sulphur in the neighbourhood of ·7% or ·8% can, I think, be made from a few coals. At a works in South Wales with which I was connected, the figures for ash in the chemist's daily report sometimes went over 14%, and were certainly underlined as a mark of warning when over 10%, but most of it was used in the blast furnaces of the works and the rest sold to other consumers, and at some foreign iron smelting centres the coke obtained is often inferior to this. Middlesbrough with coke containing under 8% is exceptionally favourably situated.

A limey slag is of course the best way of keeping sulphur low in the iron, but there are other ways in which it can be largely removed, notably by use of manganese in mixers or by Saniter's process, which can remove over 60%, though obviously at increased cost. White iron is not necessarily high in sulphur unless produced from materials rich in sulphur, and when I said that such iron was well adapted for open hearth steel making I referred to the fact that the low silicon content precluded the Bessemer process and necessitated the employment of the open hearth furnace. Obviously it would be a basic furnace to facilitate removal of sulphur and by using fluorspar or calcium chloride in the final slags, they can be made to carry over 55% lime and still be fluid, and thus 90% of the sulphur may be removed, and in this way pig containing ·37% sulphur is being treated in the U.S.A. Sulphur, therefore, although sufficiently objectionable, is not quite the bugbear it used to be, and by a final refining in an electric furnace the sulphur may be practically all removed, and without excessive cost if this be fed with the molten steel from the open hearth furnace.

Coal as a blast furnace fuel is used successfully in Scotland, and I merely suggested in this connection that smelting by coal might be a matter for experiment.

With regard to the silicious character of the dolomite, I looked upon that rather as an advantage, because my ideas required silica to be added as a flux, since the ore was deficient in this respect, and the experiments showed that TiO_2 and CaO would not flux. My conclusion was that the slag should contain about as much SiO_2 as Ti_2O_3, and the silicious dolomite provides silica as well as the other bases required, thus lessening the amount to be provided by other means.

In reply to Mr. Pooler.—I am afraid I cannot yet state definitely what compounds are formed, but that titanium dioxide is reduced to a lower oxide, at least partially, I consider to be proved by the change of colour and loss of weight mentioned when the slags examined were heated in a reducing atmosphere, though the exact loss of weight I have not been able to find time to determine. Further than this, the very marked lowering of the melting point brought about in the same way, indicates the occurrence of a chemical change. Ti_2O_3 seems to be most probably the lower oxide formed and, as shown by this lowering of the melting point, undoubtedly acts as a base with silica as the acid, as, I think, I pointed out ; but it is quite possible and probable that in the complex slags formed, TiO_2 is also present forming titanates. The results of analyses were expressed as TiO_2 because of this uncertainty as to the degree of reduction and the curious coincidence between molecular weights noted by Mr. Pooler was encountered directly I endeavoured to devise a method for determining this with some certainty. The oxides of iron would indeed have to be removed first, and I know of no method of doing so without altering the state of oxidation of the titanium present at the same time. Lastly, may I point out that a reducing atmosphere *is* essential, not with the object of reducing TiO_2 but to reduce *iron*!

I think Mr. Hamilton was a good deal too severe on the Government. Naturally I should have preferred a proper blast furnace to experiment with instead of a foundry cupola: but as nothing of the kind was available I was glad, though not too optimistic as to the result, to make the best of what could be had, and in the limited time available, rather than do nothing at all.

The installation of a small blast furnace on anything like a working scale, and running it for a long enough time to do really satisfactory work would obviously have cost a good many thousands of pounds and occupied more time than I could have given. As there existed also considerable doubt as to whether (even if it were shown that such ore could be smelted) there was any likelihood of iron ore of any kind being treated profitably, because of the limited market available, the wisdom of making such an outlay, from the Transvaal point of view, was questionable.

As I stated in the paper, I had other parts of the world in mind as well, where the question of titaniferous iron smelting is of more pressing importance than here. Had it been otherwise, and the local consumption of the product likely to be sufficient to keep the proposed industry occupied, I have no doubt the Government would have felt justified in so doing. However, the country is going ahead, and with increase of population and industrial activity there must come a time when South Africa will produce its own iron.

I agree that a preliminary calcination of the ore would probably be advantageous, as it is extremely dense in character, and calcination might render it more porous, and perhaps, through oxidation of FeO, less likely to flux and go into the slag, and a blast heating stove would undoubtedly have been a most useful adjunct.

In conclusion, I must express my thanks to the gentlemen who joined in the discussion, and to the members generally for the very appreciative way in which the papers were received.

THE TREATMENT OF ACCUMULATED SLIME, AND THE USE OF FILTER PRESSES FOR CLARIFYING SLIME SOLUTION AND BY PRODUCTS.

(*Read at November Meeting, 1909.*)

By John D. O'Hara (*Member*).

REPLY TO DISCUSSION.

Mr. J. D. O'Hara (*Member*): There having been practically no discussion on my paper, no reply is really necessary from me. The paper was written really for the purpose of giving the experience gained in filter press work and the treatment of accumulated slimes, and to place on record the fact that the clarifying of slimes solution by filter presses here was first carried out by me at the Nigel mine, which has since then been generally adopted on the Rand.

NOTES ON PRECIPITATION.

(*Read at July Meeting, 1910.*)

By F. D. Phillips (Member).

DISCUSSION.

Mr. H. F. Lofts (*Member*): The main advantages set forth in this paper appear to be that less time is taken in "cleaning up," a saving of acetate of lead is effected, which is combined with a complete precipitation, and, more important still, there is less gold carried forward. Though there is no actual proof of precipitation, it is still possible for it to happen on a minute scale, in compartments beyond No. 4, yet sufficient on a big works to affect the figures dealing with actual and theoretical extractions. It is questionable whether the scheme of re-pumping through the extractors two or three times direct from the sumps would not enable one to work with even less than four compartments. My experience is exactly the same as the author's, namely, that all precipitation takes place in the first four compartments: I therefore work with five only, and for forms sake clean up the fifth every three months. To put it briefly, after No. 4 compartment every space filled means more labour, expense and no better efficiency, but waste of time, labour and material. It appears to be open to question whether there is a consumption of zinc where no precipitation takes place. It is well known of course that zinc is soluble in cyanide. Some scientists maintain that the zinc is first attacked and then regenerated, and to prove this, state that in practice working solutions do not carry much zinc.

Mr. A. F. Crosse (*Past-President*): I would make one suggestion. It would be interesting if the author in his experiments on precipitation with solution passing through zinc boxes, made some experiments as to the beneficial effect, or otherwise, of oxygen in his solution. It might in some way assist in the precipitation of the gold by counteracting polarization. In the first box there is oxygen, in the subsequent boxes the solution goes through there is a total absence of free oxygen. Some months ago there was a paper read on a revolving cylinder containing zinc, showing how efficient it was in the precipitation of gold. Perhaps that was chiefly caused by the removal of the bubbles of hydrogen.

Mr. J. F. Pyles (*Member*): Because little zinc is found in the solutions I hardly think it follows that a large quantity of zinc is not dissolved Mr. Brazier showed in his diagrams a great loss of free cyanide in the boxes, a loss, I should say, equal to the total consumption of cyanide on the plant. Now, does not the explanation of so little zinc being held in solution lie in the fact that zinc is precipitated out in the sand and free cyanide regenerated?

Mr. Fraser Alexander (*Member of Council*): It seems to be the general opinion that to have a long box say of 10 or 12 compartments entails an immense amount of labour and expense. Personally I do not find that is so. It is undoubtedly a fact that to dress 10 compartments of an extractor box with zinc where 5 is quite ample to do the work appears wasteful and foolish. Nevertheless, in taking out the 10th compartment traces of gold will be found with practically no action on the zinc taking place. The zinc is therefore serving a useful purpose if only as a filter. It is not always that the extractor boxes are run at an even flow, and cases crop up where the flow is disturbed by one means or another, and precipitated slime is liable to escape from one compartment to the other. Although it may be that precipitation has actually taken place in two or three compartments, it is no detriment whatever to have more than is absolutely necessary to precipitate gold immediately even if the zinc only acts as a filter. In last month's *Journal* Mr. Brazier published some interesting diagrams on the requisite amount of zinc. I do not find that any loss of zinc is sustained by having a little excess in the boxes.

Mr. H. A. White (*Member of Council*): There are some interesting points in connection with the excellent paper read by Mr. Phillips and the discussion by Mr. Brazier thereon, upon which I would like to make some remarks.

In the first place I would refer to the small costs of precipitation which, no doubt, correctly, omit mention of cyanide and lime consumption. It seems clear from the general experience here that the total cyanide contents of the solution leaving the extractors are (within limit of testing error) identical with those of the solution at the head of the boxes. At the same time it is also well known that the "free cyanide" is, especially in the "strong" boxes, much reduced by conversion to the double zinc cyanide which, however, is quite a good gold solvent especially in presence of alkali in excess. The cost, 0·75d., for total handling required is equivalent to a gold value of 0·015 dwt., and it is obvious that solution of this value is the maximum allowable for the sump value in good work. But the real point of interest is the economic limit of reduction, which must be below 0·015 dwt., as that figure would pay for the whole process to be gone through again. Now, Mr. Brazier has clearly proved that zinc consumption is very small when the bulk of the gold has come down, and this is particularly the case in the boxes dealing with solutions, which in the course of treatment will partly remain in the residue either of sand or slime, and whose gold value is thus lost.

It seems clear that if the whole of this gold could be retained in the boxes by mere increase of capacity the chief extra items of cost involved would be extra labour of dressing and loss of interest on zinc shaving, and extra capital for boxes. These items are small, and if it is considered that some insurance for safety against occasional large increases of flow, and other known sources of decreased precipitation is required, it seems that the economic limit is not so much in danger of being exceeded as of not being reached.

In ordinary routine assays an accuracy of 0·01 dwt. on gold solutions shows good and careful work, though this amounts to two-thirds of the cost of precipitation. In experimental work of this kind it is not correct, however, to base deductions on such results, and it is possible by special arrangements, suitable to experimental work upon which important conclusions are to be based, to get an accuracy of ·001 dwt. where large volumes of solution are assayed and the results weighed on the finest balance procurable. Of course such accuracy is only necessary in the case of the last compartments of the "weak" boxes. In such a series of experiments with "slime" boxes I have found that a "trace" with 20 assay tons of solution from the fifth compartment was successively reduced about 50% in each of three more compartments and something less than $\frac{1}{1000}$ of a dwt. could still be detected leaving the extractor, when especial precautions were used with a much greater amount of solution for assay.

It is usual, as the author has done, to quote cubic feet of zinc per ton of solution per 24 hours but as the real criterion is area of contact it is necessary to quote the thickness of the shaving used. In this connection 19·5 lb. to 23 lb. per cubic foot would apparently indicate very coarse material. I find that new zinc cut with one tooth on the "Betty" lathe is about ·0013 in. thick, and weighs dry about 6 lb. per cub. ft., and (taken from middle of "medium" box) 12 lb. to 14 lb. wet. I would suggest that the use of such fine shavings beside giving a greater area of contact per cubic foot of box space, decreases the consumption of zinc in the acid vats and reduces the "carry over" of gold to the minimum while maintaining sufficient strength to enable it to be moved successively right up the series to the top of the "strong" boxes.

I think it would be of great interest to the members of this Society if someone would undertake the tracing out of cyanide consumption. We know it is rare to lose much in the boxes, and that the residue carries only about one-third of the total cyanide used to the dam or dump. It is possible that a careful and accurate investigation of the many other known sources of loss might reveal some gaps that could be stopped with economic advantage, and in any case the scientific interest of knowing exactly where every pound of available cyanogen goes is considerable.

With reference to Mr. Brazier's question as to advantage of heating for purposes of precipitation of gold I think the solution of the problem will be found in the strength of free cyanide maintained. I find that solutions below 20° C. require a strength of 0·010% KCy to render precipitation complete and certain for any length of time and that ·008% or even as low as ·004% "free cyanide" is sufficient with hotter solution up to 28° C. Before using heat for the slime solutions I found in the winter that it was necessary to use as high as ·017% KCy to ensure certainty in precipitation with the same flow as the hotter solutions with maximum strength of ·010% even in the coldest weather.

Dr. W. A. Caldecott (*Past-President*): As regards the influence of thickness of zinc shaving upon weight I believe that the Knights Deep shavings are between ·002 in. and ·003 in. in thickness. Whilst freshly cut filiform zinc may only weigh 6 lb. per cubic foot, yet this weight may be trebled or more for old zinc, such as the author presumably refers to, by compression and breaking up during use. I can hardly agree with Mr. White's view that the cyanide consumption in the extractor boxes is entirely due to formation of potassium zinc cyanide, inasmuch as with weak solution undissolved zinc cyanide may be found and possibly other nitrogen compounds.

Mr. F. D. Phillips (*Member*): I have to substantiate Dr. Caldecott's remark in reply to the point raised by Mr. White, as to the weight of zinc returned to the boxes. The figures referred to zinc that had already been in use—some of it for as long as ten weeks. As regards the assay values mentioned by Mr. White, I may say that we have numerous traces in our reports, but they are always entered up as 0·010 dwt.

Mr. Mather Smith (*Member*) (contributed): I was very interested in this paper, as I think it marks a further step in the right direction. It took years to discover that it was possible to reduce the number of compartments in a precipitation box: many plants are still using weak and slime solution boxes of seven and eight compartments, and I think that if the author were to try again he would have better results. From his own figures, in his third table, the solution leaving the fourth compartment of the lightly packed weak box contained only ·02 dwt. gold per ton, and the last two compartments were useless. Did his tightly packed weak boxes give lower results after the fourth compartments? If not, then the fault did not lie in the lighter packing of the first four compartments, but in the lack of fresh zinc in the fifth, and had the 660 lb. of old zinc been taken out of the 5th and 6th compartments and the fifth dressed with new zinc, he would still (allowing for a 50% extraction in the 5th compartment) have had a ·01 dwt. solution leaving the box, and a saving of the 6th compartment with its 330 lb of zinc.

Lightly packed boxes require to have the zinc well packed into the corners of the compartments and round the frame handles, and with lighter packing the boxes require more frequent dressing but as there is less zinc to handle the dressing takes less time. With lighter packing and more frequent dressing the zinc will always be fresher in the first compartments; this is substantiated by the author's fourth table, which shows that the first compartment of the slime box gave a considerably greater percentage of extraction than the first compartments of the strong and weak, and this is evidently due to the greater freshness of the zinc in the first compartment of the slime box.

Mr. Brazier in the August number of our *Journal* ends his discussion by saying that "where any appreciable quantity of this white precipitate is formed, the author will I fancy, agree that more zinc is required in the boxes than would be the case in clean solution such as at the Knights Deep."

In my paper on precipitation (this *Journal*, March, 1909) I endeavoured to show from practical working tests, that the reverse is the case, and that the greater the quantity of zinc used beyond that actually required for good precipitation, the greater would be the quantity of white precipitate formed.

In Rose's "Metallurgy of Gold" (fifth edition) a table is given on p. 296, which shows that the zinc continues to dissolve after all the gold has been precipitated, and I feel certain that the "appreciable quantity" of white precipitate is caused by excess of zinc in solution.

The Chairman: This has been a very interesting and valuable discussion, particularly to those engaged in zinc box work. Heating the solution certainly does improve precipitation where weak solutions are concerned.

THE ASSAY OF TIN ORES.

(*Read at March Meeting, 1910.*)

By JAS. GRAY, F.I.C. (Member).

REPLY TO DISCUSSION.

Mr. Jas. Gray (*Member*): I fully anticipated when bringing my paper on the above subject before the Society that the discussion would be full and instructive, and I am therefore disappointed to find that only two of our chemical members have come forward with criticisms, and to these gentlemen I have to express my thanks.

Mr. Croghan's opening remarks may be passed over without comment. The statement that perfect fusion, even of a concentrate, can be obtained in ten minutes, using NaOH, is open to serious question and notwithstanding Mr. Croghan's experience, I maintain that complete solution, except in a few isolated cases, is impossible in this time, even with the aid of the blowpipe. I admit that when reduction of the tin solution is effected with iron nails about two drops of iodine solution are necessary to produce the end point, and in ores where the tin content is low this is a decided disadvantage. Since the publication of my paper I have carried out experiments on the nickel reduction and have found that it is much superior to iron. When a mixture of 0·25 gm. of silica and ferric oxide was fused with sodium peroxide and the solution obtained reduced with nickel, it was found that half a drop of iodine solution was sufficient to produce the starch blue reaction. With ores containing arsenic it is advisable to remove it by acid treatment before fusion when the nickel reduction is used.

The first point of Mr. Whitby's criticism is with regard to what he terms my "industry in the bibliographic line," and I would suggest that he might probably have arrived at some reliable method for assaying tin ores by this time, had he been willing to wade, even superficially, into the "mud of history." It will be a source of satisfaction to me if I have been able to assist him in this matter.

I would ask my critic by what authority he takes it upon himself to speak for his professional brethren, when he says: "what we chem sts are looking for is a method which will estimate, with reasonable accuracy, the tin in an ore ranging, say, from $\frac{1}{2}\%$ to 1%." To me, since the adoption of the process outlined in my original paper, tin ores of all kinds present no terrors. Mr. Whitby's assumptions regarding the details of my experiments are incorrect and therefore need not be considered.

To come now to what my critic terms "more serious matters;" I always have a fair excess of HCl present before titration, and indeed during the whole reduction (about 60 cc. of concentrated HCl being used for each estimation), and so far I have never found that tin and arsenic are coprecipitated, even as an alloy. I hope our metallurgical friends will note his suggestion for the preparation of an alloy by precipitation. It is unfortunate that he has failed to bring forward experimental proof regarding this statement. I regret to have to suggest that he should repeat his experiments on the masking of the starch blue colour by the presence of tungsten. I will admit that to the inexperienced eye it is difficult in some cases to note the change in colour when iron nails are used for the reduction but with nickel it is comparatively easy. I would further refer him to Thibault's "Metallurgy of Tin," where it is stated that Ibbotson and Brearley found tungsten to have no interference. A wade into "the mud of history" would have repaid Mr. Whitby in this instance.

I should like to know how Mr. Whitby has "frequently proved" that the peroxide fusion is open to the same objection as the hydrogen reduction method? Has he ever found when using sodium peroxide that some cassiterite was unattacked? I fear he must have been thinking of caustic soda when penning his contribution. From the manner with which he concludes his remarks, it would appear that only lack of time has prevented him from proving, to his own satisfaction at least, that the impurities I have mentioned as having no deterrent action, have a serious effect on the correct assay of tin ores. I sympathise with my critic on his inability to separate "the grain from the chaff," and regret that my efforts in this direction have unfortunately in his case been unsuccessful.

In conclusion, I hope that my endeavours to throw some light on the dark paths of tin assaying have met with some success notwithstanding my regrettable failure in Mr. Whitby's case.

SAND-FILLING ON THE WITWATERSRAND.

(Read at June Meeting, 1910.)

By Edgar Pam, A.R.S.M. (Member).

DISCUSSION.

Mr. Tom Johnson (*Member of Council*): Having recently been doing a little sand filling, I would like to say a little more on the subject.

We want to ascertain what are the best sizes of timber to put in the barriers. The weight of sand saturated with water, and water standing on it will be, say, 90 lb. per cubic foot, this would amount to something like 10,800 tons on a barrier 400 ft. long, 6 ft. high, 200 ft. backs, and a dip of 30°. I know our barriers are not carrying this weight therefore the sand itself must be to some extent self-supporting, probably in the same way as grain does in hoppers, where I think it is found that, after a height of three diameters of the bin is reached, there is no increase of weight on the bottom.

A workable rule for strength of round parts or sticks is: diameter in inches

$$= \sqrt[3]{\frac{\text{load in lb.} \times \text{factor of safety} \times \text{length in feet}}{\text{factor for timber (320)} \times \text{constant for loading (2)}}}$$

This rule calls for much larger sticks than are used in many cases, and I do not think it is the fault of the rule, but the condition of the sand behind the barrier, of which we know little as yet. The cause of leakage of a barrier, which often happens after 100 ft. or 150 ft. of sand has been filled in, is the breathing of the barrier, caused by the greater part of the weight being taken off when the place is idle and the water drained off. When the water is drained off, the sand shrinks and parts from the roof, and on commencing to fill again, the water makes its way over the sand and down to the barriers, and there is a very great weight put on to the barriers rather suddenly. This breathing of the barriers cannot be seen, but it is quite reasonable to suppose it occurs. This movement of the barriers opens the joints at the roof and floor; therefore, to remedy this trouble of leaks, some coarse stuff should be mixed with the sand. It would also make the filling leach quicker, and it would reduce the cost of filling, as the barriers would not need so much particular work on them. To crush this coarse filling would not increase the cost, as the above saving would more than pay for the crushing. Personally, I think we should do away with built barriers as much as possible. For old workings, some drive where the lower rib is solid or mostly so, should be selected, either between the shafts or on the outer side of the shafts, the drive blocked off and the whole filled, blocking up the drives in rotation until the area is finished. After being allowed to drain, any of these drives could be re-opened to get out the drive pillars, or any other ore that was worth the trouble. In new work, the getting out of the ore and filling could be done as mentioned in the June *Journal*, p. 452, in this way only boxholes would need barriers.

Mr. H. A White (*Member of Council*): I should like to ask exactly how the coarser rock is going to help you?

Mr. Tom Johnson: If there is a leak in the barrier the coarser particles will go with the stream towards the leak; these coarser particles will get stopped at the leak, other particles back up against the coarser ones, and so on until slime will not pass, *i.e.* something in the manner in which we would form a filter. Once the filter was formed it would not matter how much breathing of the barrier took place, it would not leak sand, only water which does not matter. Another thing I may mention is that I do not think it is necessary to send sand down with water in our deep vertical shafts. I think it could be sent down dry, in the same condition it is received from the dumps. The question of sending sand down with water or dry, is more a chemical than a mechanical one, due to the nature of the water after use.

Speaking of dumps, on the night Mr. Pam's paper was read, Mr. W. T. Anderson mentioned that he looked forward to the time when all our dumps will have disappeared. I do not know that I should like to live so long, if we have to wait for the sand filling to cause the disappearance. I think it impossible to put the whole of the sand back, and those who expect it, expect too much. The crushed rock is in the ratio of 10 to 6 with the solid rock, and if we allow 40% slime, then the sand remaining is enough to fill up all the excavations made, and we cannot do that. The sorted waste will not affect the question much, as the excavations will be reduced by the weighting of the roof to that amount. I see by one of our technical papers, that some people are of the opinion that sand-filling is more or less a failure. I do not think so, I think it is going to turn out a great success.

Mr. G. H. Smith (*Member*): The costs on the Ferreira Deep are 1s. 8d. per ton. That is as expensive as stonewalling. Another thing is the difficulty of making the bratticing watertight. Why not put in a stone wall right away?

Mr. R. E. Sawyer (*Associate*): I should like to ask Mr. Johnson if reinforced concrete could not be used instead of timber.

Mr. Tom Johnson : That is a matter of expense ; also the movement of the roof or floor would crack and crush the concrete, or any solid barrier, where timber gives way an appreciable amount before breaking. Using concrete at the roof or floor is liable to fail for the same reason. I think the best way is to take advantage of any natural barriers such as faults, dykes, etc., and use as little artificial barrier as possible.

Mr. B. C. Travers Solly (*Member*) : I should like to know how Mr. Johnson would propose to carry the dry sands down the shaft ?

Mr. Tom Johnson : The sand would fall like rain. It would not drop in a lump. You can stand in a shaft and allow the fine sand to come from the top and drop on you. It must be understood that the sand would be passed through a grating.

Mr. F. W. Cindel (*Member*) : In regard to falling sand, I notice that in the discharge of cyanide tanks the hole in the bottom of the tank gradually becomes choked up. Now as Mr. Johnson proposes to take the sand from the dumps, there is of course a certain amount of moisture in the dump, and it seems to me that if this sand is taken directly from the dump, and not being dry enough it is liable to choke up the pipes.

Mr. Tom Johnson : There might be a difficulty at the entrance of the pipe, if using sand containing a large amount of moisture for so called dry sand, but there would be no difficulty below, as the sand would begin to lose its moisture in rushing through the air. The choking up of the tank outlet is generally caused by the bottom layers of sand in the tank, these bottom layers usually being much wetter than the average. For passing dry sands down a shaft large pipes, or a whole compartment would be used.

Mr. F. W. Cindel : I should like to ask if this method has been tried ?

Mr. Tom Johnson : I do not think so.

Mr. M. H. Coombe (*Member*) : The point that troubles me is as to what happens to the sand in a stope that carries a watery footwall or hanging ? The whole thing would become a huge quicksand which could not possibly support the hanging and would in its turn become a danger and standing menace to the levels below when the pillar or stull crushed or decayed above the level.

Mr. Tom Johnson (*Member of Council*) : That is certainly one of the dangers we have to guard against ; it is recognised and will be provided for at the Ferreira Deep. The water must be led away to the outside of the barrier, and not allowed to stay in the sand. This can be done by pipes of iron, wood, clay, etc., or by laying a track of rocks covered by matting, etc., to keep out the sand, or the stope could be left unfilled if necessary.

Mr. M. H. Coombe : The whole thing hangs on our mining methods. We are losing the art and have ceased to mine. In other countries where mining methods are adopted one does not hear of the ventilation question. The stopes are filled with rock and the air currents can be controlled and taken where desired. Here, where such huge mills have to be supplied with rock, everything is done at such a rush underground that we have not time to apply common-sense methods in our mining practice, therefore everything is gutted out and left open—"If the hanging stays, all right ; if it does not, let it come." That is our mining on the Rand to-day, and sand filling in our flat stopes is not going to be a cure but rather add greatly to the troubles of our underground men.

Mr. Fraser Alexander (*Member of Council*) : I think the average miner in dealing with sand filling is rather getting out of his element. I do not want to disparage Mr. Johnson's remarks, for he is experienced in both practices, but we know from the experience of handling sand dumps, that it is not every man who can handle large quantities of sand economically. I think we have probably reached that stage in sand filling in the mines where the miner finds that handling sand is somewhat different to handling rock, and I believe in a combination of the two. I agree with Mr. Johnson's remarks that he requires a filter bed that will assist leaching in the mine, and a man thoroughly acquainted with the handling of sand could assist the miner in this.

Mr. Tom Johnson : The point seems to be forgotten that if you put in a pack wall, you still have to put in a wood barrier making a double expense. A pack wall newly put in, would not stand the weight of sand and water, which on the Ferreira Deep has several times amounted to as much as 1,400 tons of sand in four hours or so, plus the water (calculate the weight when the stope is about full.) The trouble is the movement of the barriers : coarser stuff to form filters is the help we need. If we had packs in that had been compressed with the weight of the roof, it would be feasible to use them.

The coarse crushing I speak of is not by the battery, but by a crusher. I think I mentioned in my last contribution to this paper, that at one colliery they passed their filling through $3\frac{1}{2}$ in. apertures. At the colliery I visited in Belgium, 30% of the filling was from $\frac{3}{8}$ in. up to 2 in., this latter is what I should like to see going in with the sand. I believe there is no place in the world where they have tried to water-fill stuff of such average fineness as we are doing here.

These remarks to-night apply to the deep levels. Outcrop mines have a pull over the deeps, as they do not need to use so much water as the deeps do, for they can go more direct to the stopes.

The Chairman: It seldom happens that a new process for doing anything can be introduced without unexpected difficulties developing, and I hope that our mining, metallurgical and mechanical engineers will all be able to do something towards solving this problem.

Mr. Robt. Allen (*Member of Council*): In connection with the author's interesting paper, a short description of the rill system of stoping and filling, a method largely used in Western Australia, may be of interest to some members.

When working under ideal conditions, the ore body, if continuous, is divided into blocks by equidistant levels, about 200 ft. apart, and by equidistant winzes on the hanging wall side of the lode at about from 150 ft. to 200 ft. apart. The reason for having the winzes on the hanging wall side is that the stopes are thus filled with the least amount of handling. The accompanying diagram shows how a mine is blocked out in the above manner. It is assumed in it that only one winze is connected with the surface. This serves as a pass for filling, but the number of passes from the surface is purely a matter of convenience.

The filling usually consists of fresh residue, which may be of sand, or of roasted "slimed" ore or of raw "slimed" ore, and which may contain up to 25% of moisture. If the residue contains too much moisture there is a danger of it clogging the passes, so that sometimes it is necessary to stack it on the surface for a short time previous to delivery to the mine, and by this means also much of its residual cyanide content is destroyed. No chemical treatment whatever of the residue to destroy its cyanide contents is practised in Western Australia. The residue received from the surface can be distributed to the various winzes by means of a belt conveyor system along a disused level above the topmost workings.

The methods of stoping and filling on the rill system are as follows:—On any particular level a leading stope is taken out below the ore to be mined, when the drive is timbered usually, either by single stulls, or when the lode is over 14 ft. wide, by saddlebacks, at intervals of 5 ft. The latter consist of pairs of stulls sloping towards one another like the rafters of a roof, and bearing upon a longitudinal ridging of sawn timber 2 in. thick. The stulls are lagged with poles about 4 in. in diameter of a local wood called gimlet wood, or with old iron pipes. The lagging is in turn covered with old filter cloths, or the sides and linings of cyanide cases or any other inexpensive material which serves to prevent the residues from falling through.

Two alternative methods of stoping the ore are shown at A and B on the diagram. In the former all the holes are "down" holes and can be drilled wet, which is an important consideration in view of the necessity of reducing dust production. In this method the benches are taken out at an inclination slightly flatter than that of the natural slope of the filling, which in the case of residues is about 45°. This is the more commonly used method.

In the method shown at B the ore is mined by a series of horizontal cuts, and some of the holes drilled must be "uppers" and drilled dry.

Usually during the timbering of a drive ore chutes are put in at distances of about 50 ft., one (P) midway between the winzes (W), the others (Q) being intermediate. As stoping proceeds passes about 4 ft. × 4 ft. in the clear are built above these chutes, usually of 7 in. logs, but sometimes of 9 in. × 3 in. sawn timber. Each winze is also "cribbed" up except when it will not be required later on for passing "filling" to lower workings or for ventilation. In such a case the timbering of the level below the winze can be closed up and the winze filled up.

The breaking of the ore and the filling of the stopes with residues succeed one another alternately. Before the benches of ore are blasted eucalyptus saplings or slabs are laid on the sloping surface of the residue filling. These serve to keep separate to a great extent the broken ore from the residue, and assist in its "rilling" into the passes, very little labour being then required. The passes are then built up close to the working faces and covered over to prevent residues from entering them, and the poles or slabs are removed. More residue is then dumped down the winzes into the stopes, filling them up to a convenient distance from the faces. When a stope has assumed the appearance shown at C, when all ore can be rilled to the passes P, the intermediate passes Q are no longer entirely necessary. As stoping proceeds the appearance of the stopes becomes similar to that shown at D. It is usual to stope a series of blocks on the same level simultaneously, so that the filling of the stopes with residue on both sides of the winzes and the building up of the passes can be carried on symmetrically. When the stopes are nearly beaten out, as shown at E, it is usual to sink subsidiary winzes R in the triangular blocks of ore left

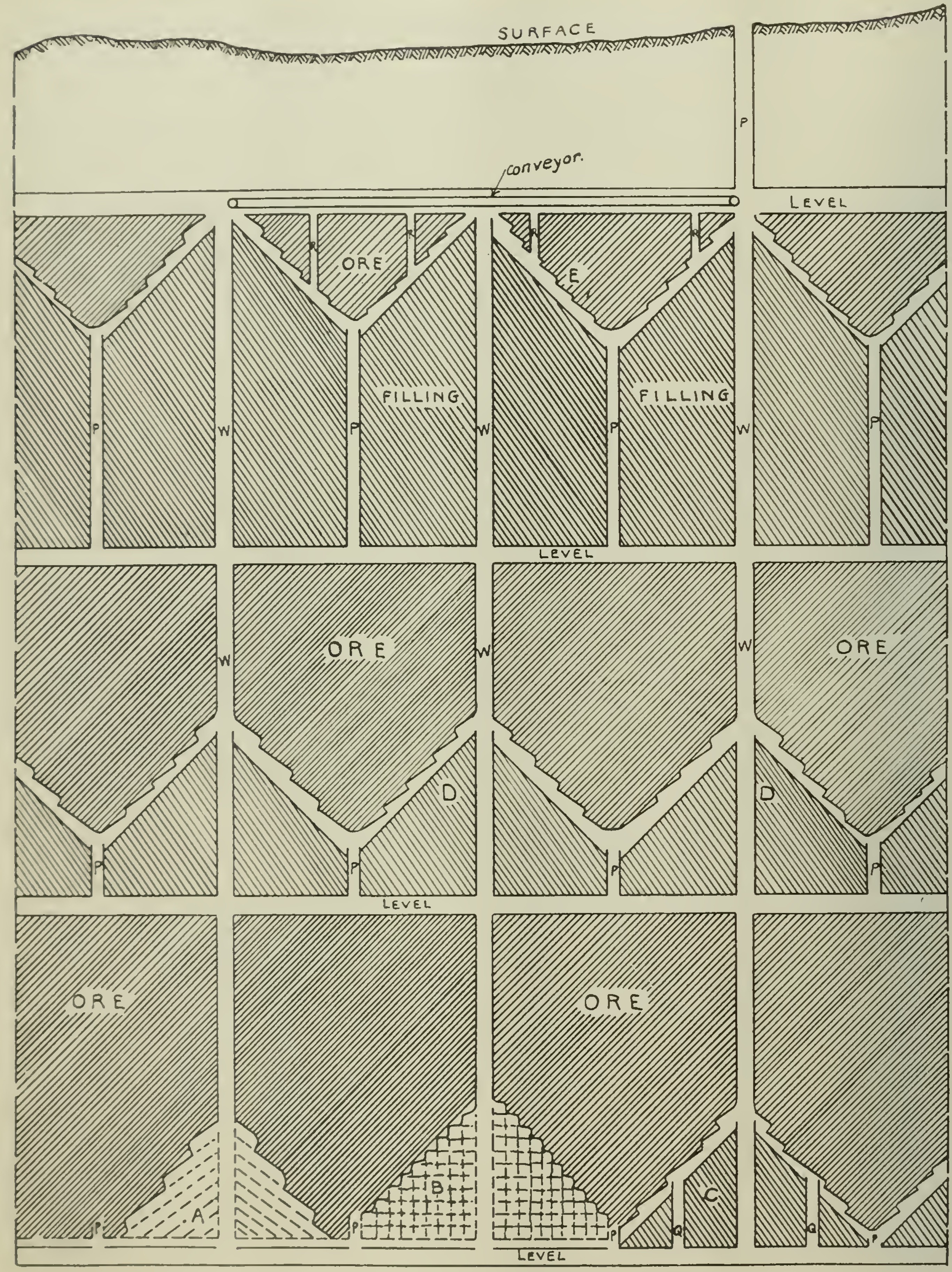


Diagram showing the method of Stoping and Filling on the Rill System.

below the level above, through which the residues for filling can be dropped. The lodes in Western Australia are usually very steep, and the rill system of stoping and filling is generally particularly applicable to them. Generally, as the dip of the lodes decreases below 45°, more and more shovelling is necessary to assist the rilling, and the method becomes inapplicable when the dip is less than about 35°.

Filling with residue in Western Australia has been in use for about 13 years, and its present cost is about 10d. per short ton of ore mined.

The accompanying photograph is of an underground belt conveyor installed on the Great Boulder Perseverance mine for distributing residue, the drive it was installed in being specially driven for it.

A Member: What is the average dip of their stopes?

Mr. R. Allen: They are very steep, quite different to what they are here.

Mr. G. H. Smith (*Member*): What is the average dip of the face from the winzes?

Mr. R. Allen: About 45°.

THE EFFICIENCY OF LABOUR UNDERGROUND.

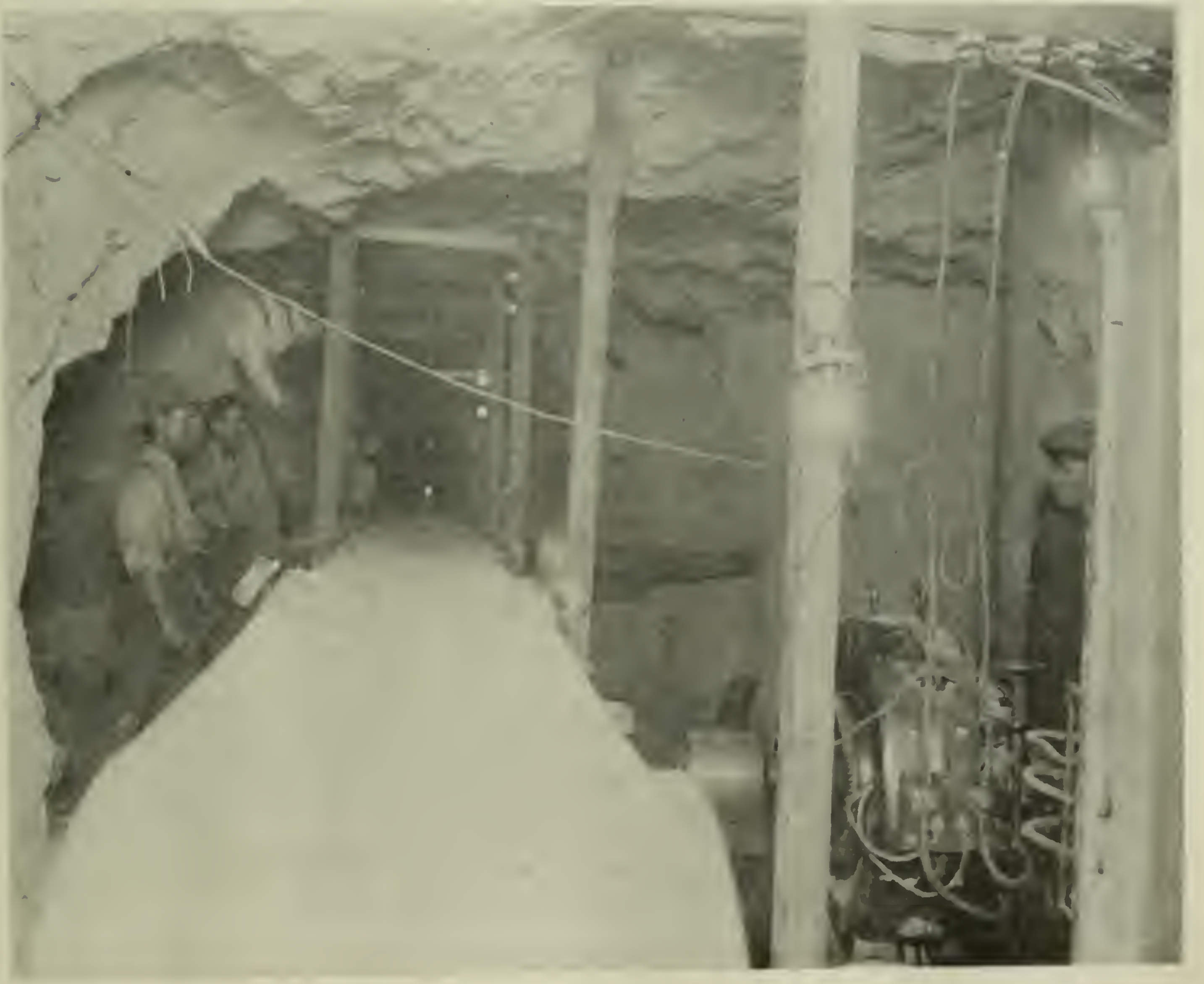
(*Read at August Meeting, 1910.*)

By Tom Johnson (Member of Council).

DISCUSSION.

Mr. W. S. V. Price (*Member*): I was more than pleased to find that a phase of mine economics (I had almost said mine economy as practised on the Rand) was to be dealt with at our last meeting, and introduced by so able an exponent as the author.

It may be taken as an axiom that economy without efficiency is only another way of spelling extravagance, yet to the average mining-man it



Underground Conveyor on the Great Boulder Perseverance Gold Mine, Kalgoorlie, W. A., for Distributing Residue for Filling.

would appear as if most of the mine managers studied economy instead of efficiency with regard to their white workers. Possibly the reason for this is the same as that alluded to by the author with regard to a certain class of shift-bosses, viz., lack of experience in themselves. There are some wide-awake managers who do try to gauge a man's capabilities and who pay him accordingly, and, as was pointed out, he will and does get better value for every pound he pays. He gets and keeps a good, efficient and trustworthy lot of men who are proud to work under a boss who knows his own work and knows when his subordinate does a good day's work.

On the mine where I am working there is a system of calculating all the expenses connected with the stopers. Every day returns are made out showing number of boys, number of holes, inches, and the amount of gelatine used. After measuring-up day, it is ascertained exactly how many boys were required to break a fathom of ground, or a ton of rock; and the number of pounds of gelatine to break a fathom of ground; the men being paid bonuses on these figures. Making all allowance for different kinds of rock, hard or soft, etc., a very good idea of a man's efficiency can be formed. For instance, in one case it had taken a great number of boys to break a fathom of ground, but the amount of gelatine per fathom was very low. What did that show? That the man had poor boys, but being a good miner, knowing his boys could not drill long holes, he only put the burden on them that a short hole could carry and then gave it just sufficient to break it. So I think this is quite a step in the right direction.

It may not be necessary for the supreme head of a concern (mine or store) to have worked in each and every department under him, although it would doubtless be of great advantage, but I certainly think that those immediately in charge of any workmen should be men who have done the actual work themselves, so that whatever may go wrong they could tell what to do to right it. Some may think this a pretty tall order in mining, but when I was at sea I used to think that the mate of a sailing vessel who could not do anything that was required on board of his own vessel, was not fit for the berth, whether it was to make a sail or spar, caulk a deck, or put a few rivets in the hull, and the same principle applies to a boss in a mine. A few years ago I heard a shift-boss say that he "was not in the mine to teach miners their work, they were supposed to know their work before they started on the mine." That may have been all right then, but even with my limited experience I have found that a good many men need quite a lot of coaching, and if by doing that a shift-boss increases the miners' efficiency, I claim he is only doing his duty, for I take it that that is why a man is put in such a position—to get the best work done. There is one very big mistake often made by otherwise zealous shift-bosses, they seem to forget that the white miner cannot do (is not supposed to do) the work himself, but to make his boys do it. He will, as the author has pointed out, get in a temper and bully a man before his boys, which can only spoil discipline, which is already too lax. Or if a man complains of laziness or insolence on the part of the boys, he only laughs at him, whereas if a native complains that he has to *work* for his pay, he appears to side with him. The *native* is the unit, *not* the white man. Train the white man how to get the last ounce out of those under him, and the only way to do that is for the white man himself to have learned by actually doing the work, so that he shall not expect an impossibility. The same thing holds in this case as I mentioned before, for after all, the white man is not the *miner* as we mean the word at home, but ganger or boss, and I hope he always will be in this country.

The author spoke of the way men are paid a fixed or standard wage, no matter how much or how little work is done. We know that is foolish, as the good man gets discouraged and the lazy or otherwise inefficient is satisfied to do as he is doing, seeing that he is as well paid as a first-rate man. There is another way whereby good men are discouraged. The same rate will be paid for stoping or development whether the ground is hard or soft, tight or easily broken! One man may have the luck to get a big cheque for very little work, and another, perhaps better, man will slave for barely day's pay. Perhaps I am digressing from the point at issue, but a little common-sense would prevent such a piece of injustice.

When I spoke at our last meeting I said that the author's paper was practically bound up in that of Mr. Penlerick's. I mean this. Improve the conditions underground and you will increase the efficiency of all, of coloured first, for a native cannot drill as well in a gas-laden, foul atmosphere as he can if the air approaches something akin to purity; the boss too will feel more like hustling round a bit if his head is clear of smoke, etc., instead of being half "dopy." We know that mine gas is not the only thing that makes men "dopy" underground, they *have* been known to get it in other ways, even sometimes in a canteen; but what has the average miner got to keep him out of the canteen? Go to the average mine single quarters and see how snug and homelike they are! I say nothing about boarding houses, for some are very well kept, and all things considered, such as the erratic hours of mining men, etc., they keep very fair tables. There are exceptions of course, where the boarding-house keeper is first cousin to a shark and anything is good enough for a common miner ; but in most cases it is easy for a man to board elsewhere if he is not satisfied. But a man's life is something more than work, eat and sleep, and what has the miner got to keep him from the canteen, which spoils the efficiency of many of our very best men ? We heard about having better rooms, efficient change-houses, etc., at our last meeting. The inefficiency of the latter makes many men keep their digging clothes in their rooms in spite of the law. I remember that Dr. Macaulay, in his paper on mine gases, a few years ago said it was up to the miners to use the means at their disposal to prevent dust and gas, etc., and I quite agree with him, although in this case many men keep their clothes in their rooms because they are not safe in the change-house, or may be owing to the stupid or defective system of heating. Their clothes are damp when they go to put them on, and that is a very good way to get a touch of the pneumonia we heard so much about last meeting. I have known change-houses where the steam pipes were up in the air over the tops of the lockers. You would think they wanted to warm and ventilate the roof of the change-house. The engineer that invented and installed such a system evidently knew very little of the action of heat and evidently thought it struck down. In other ways I admit the miner does neglect the means at his disposal to maintain his health, and thereby increase his efficiency. I am sure that I have inhaled far more dust, gas, smoke and foul air generally in six months' shift-bossing than I did in six years' mining. I need to keep telling men to use the water hoses, etc., at their working places. How very rarely we find men keeping the face of the rock wet when collaring a hole, and if spoken to about the dust from it, they will tell you it is to be a wet hole when it is in far enough. I have repeatedly told men, and insisted on it, that water be applied to the face as soon as the drill strikes the rock, *whether it is for a wet hole or a dry one.* It is necessary from the miner's point of view to prevent dust, and it is equally important from the owner's point of view, for it helps to keep the temper of the drill and so increases the efficiency of the drill, and thereby the miner. The drills need all the nursing we can give them to keep their temper and incidentally the cutting edge, for goodness knows the tempering of the drills is seldom troubled about in the shops, they are not paid to do that, and so long as they look pretty and the number is turned out, all is well. In this way the efficiency of all connected is seriously impaired. To my mind, the whole system is entirely wrong. Instead of the drill-sharpener being paid so much per score or hundred, he should be paid so much per machine ; then far more attention would be paid to the tempering and steel would be tempered to cut the particular kind of rock for which it is to be used. The steel bill per month would be decreased by quite 50%, for drills that now get stamped up with a few taps of the machine would do duty for two or three holes, and even then would not come out of the hole like so many we see—either flat or else the corners all broken off. The amount of steel wasted each month must be enormous, to say nothing of the smith's time in cutting off damaged ends, etc., and the miners' time changing drills in the machines. The same thing applies to hand-drills, a white man does the mechanical part in the drill-sharpening machine and a nigger looks after the tempering, if any. How does this affect the dust question ? A well-tempered drill, drilling a dry hole *cuts* the rock and the chips fall to the ground, but the badly-tempered drill has to pound and pulverise the rock and then the fine particles float about in the air and get inhaled by the miner and his boys. I like the author's idea of the shift-boss, or what would be his equivalent, getting paid a bonus for the efficiency of those under him, and would like to see it put in force now, but—and it is a very big *but*—only one man would have to have the handling of one squad of men, they would always have to be on the same man's shift, and no one else must be allowed in their working place, or we should find exactly what is happening all along the Rand where men are working double shifts, *i.e.*, the loafer would get the same as the hustler's share of the cheque, while the hustler's average was reduced to the ordinary dead level. One man one job is a pretty good principle anywhere, but especially in mining, where we so very rarely find two men who do the same work in exactly the same way. There are so many ways whereby the general efficiency of the mine workers could be improved that it would perhaps be wearisome to continue, but one thing stands out very strongly in my mind as a general efficiency destroyer, and that is that curse of the Rand, Keep the mill going ! It does not matter (at least it looks that way) what goes through the mill, but the directors must not on any account hear that the mill has been stopped. Seven days a week it has to be fed with something. Would it not be better to run it only five days a week and put only very finely sorted ore through, leaving out everything except pay-dirt ! Of course we cannot sort fines, but if closer attention was given to mining only the pay ore, more resueing done, greater care in developing so that while the mine is being opened up the thought of handling the broken rock from the stopes will be kept in mind, efficiency would be improved, dividends would rise, and the wages need not come down at all, but if anything increase to those who put their hearts and their backbone into their work.

Mr. E. J. Wiseman (*Associate*): It is with some hesitation that I, as a young man, venture to add any remarks to this excellent paper; but every practical miner who gives this subject a little serious thought will agree with the author that there still remains ample scope for improving our efficiency underground.

In one part of his paper he deals with methods whereby better results would be obtained in the breaking of ground, and I will endeavour, if possible, to emphasise his remarks in this direction by bringing forward some ideas I have heard expressed on the matter by old experienced miners, in the way of comparisons between hand and machine stoping. One cannot but be very forcibly impressed by the great difference existing between the efficiency we obtain from hand and machine stoping. A good hand stoper on contract would not expect a good cheque if he did not break five fathoms of ground per case of explosives used, while his fellow workman, the machine stoper, is well satisfied if he manages to break only three fathoms per case. Both men may be equally energetic and capable miners, and have similar working conditions as regards stoping width, dip of reef, and breaking qualities of ground. Naturally one asks why this disparity should exist, and why we have this expenditure of energy in one case, without a compensating efficiency. Is it due to the two different ways the men stope, and does the system of handstoping possess any advantage over the methods prevailing in machine stoping? The machine stoper drills his four, five and six holes per bench, as the case may be, each shift. It is not so long ago since he used to fire *all* these holes at one blast, with the result that long sockets were often left; the whole bench would sometimes be hung up, or one hole would cut out another, and so leave a misfire. As this did not pay him he resorted to the scheme of blasting only the front holes on one shift, leaving the back holes for the next, and found he was then better able to judge the burden on the holes and the formation of the ground, and so obtained better results for the amount of explosives he used. This is the system of blasting now in vogue. But still the hand stoper can show greater fathomage per case of explosive used, and why is it so? If the machine stoper pondered the matter over he, would probably conclude that the hand stoper's method must be better than his own, and set himself to find out where it was so.

The hand stoper drills only one hole a shift on each bench, and blasts it separately. Sometimes he does drill two holes on the bench, not more, and even then, after blasting them together, he will likely find that he has not obtained as good results, as if one hole at a time had been drilled and blasted on that bench. Would it therefore not pay the machine man to take a leaf out of the hand stoper's book, and instead of trying to drill as many as six holes to a bench, put in *only two*, and blast *one* each shift? By giving him extra bars, he could still drill the same number of holes per machine each shift, and I think the results would amply repay him for the extra trouble he is put to, in rigging up his machines twice, or even three times, a shift. He would gain further efficiency if he could have his working face to himself, single shift.

Another comparison can be drawn in regard to the sockets of holes left after each blast. In hand stoping we have the holes of almost uniform gauge right through, so that the explosive can be evenly packed, right from the bottom of the hole, thus allowing a well-placed hole to break clean out. On the other hand, in proportion to the depth and burden of machine holes, what a large number of sockets, 12 in. to 18 in. deep, are left standing, especially if the ground does not happen to be exceptionally good breaking. These sockets represent the depth drilled by the long chisel bits, so here again we see a waste of energy. Even if the chisels are well sharpened and tempered, they lose their proper gauge quickly, and the bottoms of the holes drilled by them are then too small to receive the sticks of machine gelatine. If the stoper fails to notice this, he gets a cushion of air left between his charge and the bottom of the hole, and a deep socket is left standing after blasting. On the other hand, should he procure hand gelatine or else split his machine gelatine, so as to get it right into the bottom of the hole, it simply means that the portion of the hole drilled by the chisel is of such narrow gauge as really to represent only a hand hole, but it is given the burden of a machine hole. So he gets another deep socket left after blasting it. Under these circumstances, would it not be much better to dispense altogether with the long chisel bit, and use in its place a long star bit? I believe that the holes would then break better, and deep sockets would then be conspicuous by their absence.

I put forward these ideas for what they are worth, and am only sorry I cannot express them more concisely. I believe that, given an impartial trial, they would lead to improved efficiency. If, on the other hand, a test should prove them impracticable, then we should have the satisfaction of knowing that the present method of drilling over our benches must be as good as we can have, as far as machine stoping is concerned.

The meeting then closed.

Visit to the City Deep, Limited.

Some two hundred members of the Chemical, Metallurgical and Mining Society paid a visit to the City Deep on Saturday afternoon, the 10th September, to inspect the new surface plant, which is now in course of construction. The visitors were conducted over the works by various officials of the company, Mr. Whitford (Mine Manager), Mr. A. M. Robeson (Consulting Mechanical Engineer), Mr. E. J. Laschinger, Mr. Simons (Mine Secretary) and others, and visited the mill, the tube mills, the gold recovery house, sand plant, slime plant, and power arrangements. A few of the more venturesome climbed the big cantilever boom, which reaches to a height of 200 ft. and which will be used for forming the dump.

In welcoming the members of the Society on behalf of the directors of the City Deep, Mr. R. W. Schumacher said he thought all would agree that these visits of engineers and scientists to the various mines were a very excellent thing for the industry. He would not dwell at length on the special points of the new plant, but he thought his hearers would agree that those responsible for its erection might well be proud of their work. He referred in the first instance to Mr Robeson, who was back at work again, after a short but serious illness, also to Mr. Laschinger, Mr. Bowen, Mr. Henderson, and Mr. Ross, all of whom shared the responsibility of the construction work. He thought the City Deep had the finest mill in the world. They would have noticed the steel work and the use made of reinforced concrete, and Mr. Laschinger's cam shaft support which was an entirely new feature. However proud they might feel of the work that had been accomplished, he thought if they asked Mr. Robeson that gentleman would tell them he was confident of being able to do even better work next time. Mr. Robeson had several new ideas up his sleeve and he (the speaker) would not give him away, but at their next plant, the Modder B, there would be a number of novel features of great importance. One of these would be the installation of the Butters vacuum filter process. This had been tried successfully in the States and in the mines of Mexico, and they had the pleasure of welcoming the inventor (Mr. Paterson) among them that afternoon.

It was a process for separating the gold solutions from the slimes, and, speaking as a layman, he had no hesitation in saying that it struck him as being one of the most important innovations introduced since the tube mills. It was installed at a Barberton mine (the French Bobs) and at the Crown Mines almost simultaneously. They were sometimes accused of being too conservative, but he did not think the reproach was deserved, for during the last few years the industry had made very great strides. A few years ago, for example, they knew very little about tube mills, but these were now the most important feature of a reduction plant. The plant of the City Deep was laid out with the original intention of having as the first unit 200 stamps and nine tube mills. He thought, with the experience they had, it might be asserted with fair confidence that they would change their programme to a very great extent, in enlarging existing plants. He did not think it likely that the City Deep would ever add another stamp. They would add tube mills, but no more stamps, and he thought the same policy would be applied in very many other mines. Apart from the surface construction, in which he was glad they took such an interest, there was another consideration, and one he thought not unimportant, that was the mine. Mr. Whitford and Mr. H. Stuart Martin, gentlemen in whom they had the fullest confidence, said that the City Deep was going to be one of the great mines of the world. He wished all success to the Chemical, Metallurgical and Mining Society, and coupled with the toast the name of its president, Dr. Moir.

Dr. Moir, in the course of a brief reply, said the works of the City Deep had apparently been built with a view to an existence of several thousand years. He feared that archaeologists of the future would be sorely puzzled by the ecclesiastical appearance of the concrete buildings, and be led astray in their speculations by taking the vats for baptismal fonts. He wished all success to the City Deep.

Mr. Whitford and Mr. Robeson also responded.

A Short Description of the Ore Reduction Plant.

General.—This plant is designed to handle and treat 65,000 tons of ore per month, and in general arrangement and details of construction presents some novel features which members of the Society will doubtless recognise when looking over the plant.

Sorting and Crushing Station.—The ore from the mine at the western shaft is dumped by 5 ton skips over grizzlies into fine and coarse ore bins built up against the steel headgear. Thence the fines are taken by a 20 in. belt direct to the main ore bin. The coarse ore is taken by *four* 36 in. inclined sorting belts of an average length of 150 ft. to the crushers, which are set on the top of the main ore bin. Each belt feeds *three* 12 in. × 24 in. jaw crushers, one of which is used as a spare. The return portion of each sorting belt acts as a conveyor of the sorted-out waste and delivers the waste to a 20 in. waste belt (common to the four belts) delivering to the dump. The main ore bin is 113 ft. × 23 ft. × 16 ft deep in the middle, capacity 1,000 tons, and is massively constructed in reinforced concrete so as to give ample strength and weight to minimize vibrations due to the crushers. The bin has hopper bottoms fitted with heavy doors operated by compressed air. Through these doors the crushed ore and fines are discharged into the trucks for transport to the mill. The eastern shaft is to be equipped with crusher station when the scale of operations is increased.

Ore Transport to Mill.—The railway line to the mill is 6,500 ft. long and of 3 ft. 6 in. gauge, formed mostly in embankment. Maximum grade 2·65%. A train will consist of *four* or more 40-ton all steel Klussman trucks, with with automatic bottom—side discharge doors. The electric locomotive consists of similar 25-ton 6-wheeled halves coupled as a unit and fitted with vacuum brake gear for the control of the whole train. Each half loco. has a 150 H.P. induction motor operating the driving wheels through gearing and side rods. The two overhead wires will be from 16 ft. to 17 ft. above the rails. The novelty of the system as far as South Africa is concerned is that heavy electric locomotives will operate on 2,000 volt 50-cycle three-phase current as furnished by the power company. As one leg of the line must be earthed, the current for the railway is taken through buffer transformers. The loco. will not run under the crusher station ore bins, thus obviating any danger of contact with live wires round the station. The ore transport line will also connect with the mill stores, general store yard and workshops, S.A.R. siding, and shafts.

Stamp Mill.—The mill consists of 200 stamps, arranged in units of *ten*, each unit being driven by a 50 H.P. motor. Weight of stamps when new, 2,000 lb. The stamps have long heads and short stems. The mortars are short with special housing round the heads. A layer of half-inch felt is placed between the mortar bases and the concrete foundations. There are no king-posts, as the concrete foundations are carried up 14 in. wide, with indented steel bar reinforcing to above the level of the mortar-box tops. On the tops of the foundations is bolted a heavy cast-steel frame which carries the cam shaft and stem guides. Each cam-shaft rests on 11 bearings, as beside the 3 main bearings there are bearings intermediate between the cams. This is to minimize, if not entirely obviate, cam-shaft breakages by reducing vibrations. Cam-shafts are riffled and hollow. Stems are 4 in. × 13 in. long running in cast-iron guide blocks bolted to the steel guide girts, but with a wood cushion between.

As these stamps are designed for very heavy duty, each set of five stamps is provided with two challenge feeders to feed behind the second and fourth stamps of each mortar-box.

The whole skeleton of the mill-bin structure and battery house and roof is of steel, wood being only used for bin-lining, floors and girts for fastening on the corrugated iron. The ground floor of the mill is graded and will be concreted throughout. This makes for cleanliness, and the whole building is well lighted and space allowed for easy access to facilitate inspection and proper attention. All launders have 9% grade and are formed in concrete. Lines of crawls are provided for moving all heavy parts in the battery.

Pulp Elevating Plant.—As the site of the plant is very flat ground, it was found advisable to give two elevations to the pulp, the first between the battery and tube-mills and the second between the shaking tables and sand collectors. In each stage the elevation is done by one 12 in. Robeson-Davidson sand pump, with another pump as stand-by.

Tube-Mills.—There are *nine* 5 ft. 6 in. × 22 ft. Eckstein standard tube-mills. These are each driven by 100 H.P. slow- speed motor through a Citroen gear reduction. The driving gear is located at the outlet-end of the mills, thus avoiding the crowding together of classifying cone, pebble feed and driving arrangements at the inlet-end. It is at the head of the tube-mill where attention is most required and where things mostly go wrong, and the placing of the driving gears also at this end is both inconvenient and dangerous to the workers.

The tube-mill pebble-bin is built into the mill bin framing at one end, and a monorail arrangement provides for the transport of feed ore to the hoppers of 1 ton capacity at the head of each tube-mill.

Gold Recovery House.—Immediately behind the tube-mills is located the house where all the gold is recovered and handled. In this house are placed, under one roof, the amalgamating shaking tables, extractor boxes, clean-up machinery, strong room and refinery ; also conveniently and incidentally the sand solution pumps, second pulp elevating plant, zinc lathes, and office of the ore-reduction officer.

There are *seven* shaking tables, 5 ft. × 12 ft., installed for each of the 9 tube-mills, and room left for the addition of an eighth table to each one, and the whole of the mill pulp runs over these tables as no amalgamation is done in the battery. Easy and clear access to each table for the dressing and scraping of plates has been provided for, and the whole floor of the gold-recovery house has been concreted on solid ground.

There are *eighteen* extractor boxes, 5 ft. × 30 ft., each having 10 compartments for zinc shavings, but each box is partitioned off in the middle, thus virtually making two boxes set end to end. The bottoms of the boxes have a side slope one way only, and a cock is fitted to each compartment on its deeper side for discharging into a lock-up gold slimes launder running the full length of each box.

The main part of the gold-recovery house is built in wood and iron, but the refinery, which is partitioned off by heavy wire netting, is constructed of concrete and steel, the only woodwork being the purlins of the roof. Special attention has been paid to the ventilation of the refinery. An enclosed slag-yard has been provided outside the refinery door

Solution Sumps.—The solution sumps behind the gold-recovery house are in excavation, lined out with reinforced concrete, and the bottoms have also bitumen sheeting embedded in the concrete. The sump dimensions are: *Three* sand solution sumps, each 48 ft. × 47 ft., and *one* slime solution sump, 122 ft. × 47 ft., all sumps 12 ft. 9 in. deep. The ample capacity thus allowed is to meet future requirements as regards increase in the scale of operations without adding to the sump capacity.

Sand Plant.—The collecting plant consists of one row of *six* vats, 50 ft. dia. × 10 ft. deep, built together with their supports and the trestling for the Blaisdell excavators, in reinforced concrete. A 24 in. conveyor belt running under the centre line of these vats takes the sand, elevates it, and delivers it to another belt, running at right angles, to the top of the sand-leaching vats. This leaching plant comprises *two* rows of *six* vats each. Vats are 56 ft. dia. × 12 ft. deep, the sands being filled in by a Blaisdell distributor. The vats are of steel, set on a floor and supports of reinforced concrete. The trestles and runway for the sand distributor and sand excavator are also of reinforced concrete tied into the vat supports. The solutions leached from the vats all go to a small solution classifying house where the classification of solutions going to the extractor boxes is made. The steady head tanks are installed in this house.

Sand Residue Disposal Plant.—The sand is discharged from the leaching vats by means of a Blaisdell excavator, the central discharge chutes delivering to a 24 in. conveyor belt, one under each row of vats. A short cross belt at the end brings the delivery to a common point; thence this sand is conveyed and elevated to the tailings dump by a 24 in. inclined belt carried on a steel cantilever frame instead of trestling. The delivery of this belt will, when the plant begins operations, be over 100 ft. above the ground, and the cantilever support will not be buried in the sand.

Slime Plant.—This consists of *four* conical-bottom collectors, 60 ft. dia. × 10 ft. and 16 ft. deep, built on the ground of reinforced concrete. These have inside peripheral overflow launders of steel. There are *two* steel conical-bottom air-agitation vats, 32 ft. dia. × 30 ft. and 38 ft. deep, for the aeration and agitation of charges, and *eight* steel conical-bottom 1st and 2nd wash vats, 70 ft. dia. × 16 ft. 6 in. and 23 ft. 6 in. deep. The pumping plant consists of *two* 12 in. Robeson-Davidson sludge pumps, *two* 9 in. Roturbo solution pumps, and *one* 6 in. highlift Roturbo sluicing pump. The air for slime agitation is supplied by a small Bellis & Morcom compressor belt driven. An emergency service is also provided by a pipe laid from the mine air service.

Mill and Return Water Service.—The mill service tank is 50 ft. dia. × 12 ft. deep, of steel, and supported on steel trestling. The two return water tanks are of steel set in excavations, and are 50 ft. dia. × 15 ft. deep. The pumps are *two* 11 in. Roturbos. The water-piping in the mill is formed as *two* ring mains, and is made up of exact duplicate lengths to facilitate any repairs or cleaning out.

Power Arrangements, etc.—All power for the operation of this plant is purchased electric power. All machinery is therefore driven by electric motors. In all cases except the tube-mills, the motors drive through belts. The power is distributed from the transformer station and switch house by underground cables.

It will be noted that room is left in all parts of the plant for extensions, and so that these extensions can be conveniently carried out.

Visit to the Simmer and Jack Proprietary Mines, Ltd.

On Saturday afternoon, the 24th September, about a hundred members of the Society spent a couple of instructive hours in an inspection of the reduction works at the Simmer and Jack Gold

Mine. They were received by Mr. O. P. Powell (Acting Manager), Mr. Wilson (Resident Engineer), Mr. Greathead (Mine Captain), Mr. Lea (Cyanide Foreman), Mr. Dixon (Mill Foreman), Mr. McArthur Johnston (of the Goldfields Laboratory) and Dr. Caldecott (Consulting Metallurgist to the Consolidated Goldfields). The visitors went about the property in groups, each of which was attended by one or other of the gentlemen named, and all were able to gather a comprehensive idea of the reduction part of the surface works. There was no time to go underground. One of the most interesting features inspected was the contrivance by which the disused workings below are being filled with sand in which all traces of cyanide have been destroyed.

After the inspection, the visitors assembled on the stoep of the recreation hall where refreshments were served. Amongst the members of the Society present were: Prof. G. H. Stanley (Vice-President), Messrs. F. F. Alexander, E. H. Croghan, Tom Johnson, J. E. Thomas, A. Whitby, H. A. White, A. F. Crosse, J. S. Cellier and Fred. Rowland (Secretary).

In a brief speech, Prof. Stanley expressed the thanks of the Society to Mr. Powell, Dr. Caldecott, Mr. Wilson, and Mr. Smart (Reduction Works Manager), the last-named of whom he was sorry to say was ill, and whom he hoped would soon be about quite well again. They had had an instructive and enjoyable afternoon. One of the things that had forcibly occurred to him (Prof. Stanley) was the free and open way in which information was given of the work done on the different plants. There were no secret processes at all, and there was a spirit of progress which gave a denial to the charge that on the Rand they did not take advantage of new practices. They had been thoroughly abused for not taking the lead in new ideas, of which, of course, they could not have a monopoly. In his opinion a large part of the progress since the early days was due to Dr. Caldecott and his able lieutenants, some of the innovations in the plant which marked a distinct advance in the metallurgy of gold-mining being directly due to their initiative.

Mr. Owen Powell (Acting Manager) heartily agreed with Prof. Stanley's remarks. When he (Mr. Powell) came to the mine the plant was already under construction, and his own work had only been to carry out what his predecessor had arranged. At all reasonable times, the Simmer and Jack, above or below, was open to members of the Society.

Dr. Caldecott said that the satisfactory working of the plant was largely due to those who were associated with him, especially Messrs. Smart, Wilson, and McArthur Johnston. He was pleased to recognise the value of pure chemical research in regard to the problems of the gold industry of the Rand. In regard to sand-filling the cyanicides they had found satisfactory and employed in daily practice on the Simmer and Jack were those discovered after long research by their friend, Mr. H. A. White.

Mr. Powell thanked Mr. Wilson and others who had assisted to show the visitors about the property, and Mr. Crosse maintained that by means of the successful chemical research employed on the Rand they were able to treat, at a profit, thousands of tons of low-grade ore.

Mr. H. A. White added a few words of thanks to the officials and directors, the latter of whom he likewise congratulated upon their enterprise and confidence in their technical advisers. After this a photograph was taken of the gathering, and the visitors returned to town at dusk.

Technical Details.—The following particulars of portions of the reduction works will be of interest:—

Stamp Mill.—Twenty-four batteries are equipped with short wooden observation tables only, without amalgamated plates, the pulp being elevated to classifiers for amalgamation of the metallic gold on the plates in the tube-mill circuit.

Tube-mill Classification.—One diaphragm cone classifier (6 ft. x 9 ft.) is employed for each tube-mill, as described on p. 282 of this *Journal*, February, 1910.

The classifier underflow is about 400 tons of sand per 24 hours as a pulp containing about 27% moisture.

Sand-filling.—The method employed for current sand residues was fully described in this *Journal*, August, 1910. The 60% of clean sand yielded by the crushed ore occupies about the same space in the mine as the ore before it was mined a fortnight earlier. This sand drains readily underground, yielding a clear effluent free from cyanide. The 4 in. centrifugal pump in use is capable of pumping 32 tons of sand per hour as a pulp containing 60% of moisture.

Continuous Sand-Collecting Plant. — Three sand filter tables, 25 ft. in diameter, with 3 ft. filtering breadth, are installed, of which one is a spare; the two others readily handle the total production of 1,300 tons of sand daily. The sand from the filter tables is transferred with cyanide solution as a pulp to the cyanide treatment vats. No washing cones are employed in addition to the four 8 ft. × 10 ft. diaphragm cones above each table. Six similar return sand cones following the table cones are in use. By means of this installation the renewal of a large number of old wooden vats has been avoided and better extraction obtained, apart from securing other advantages. The method was fully described in this *Journal*, August, 1909.

Battery Sand Clean-Up Plant.—This constitutes a miniature reduction plant for the rich mill by-products, and includes a 3 ft. 6 in. × 6 ft. tube-mill, followed by a small shaking amalgamated plate. The very fine tailing pulp, after this crushing and amalgamation, is pumped to 2 air-lift cyanide vats, 10 ft. in diameter by 18 ft. deep, one of which is used for collecting and the other for treatment. These vats have 60° bottom grades, and are fitted with Adam's cut-off gates. A considerable amount of mercury and amalgam is recovered in these vats or dissolved in the cyanide solution, and subsequently on the zinc shavings, from which it is obtained later by condensation during calcination of the zinc gold slime. The cyanide treatment in the air-lift vats yield about 98% extraction and comprises repeated agitation by air and the circulating pump, with alternate settlement and decantation of the gold-bearing solution.

Obituary.

The deaths of the following members are recorded with much regret :—

Mr. **John Radford Streeter**, the mine captain of the Village Main Reef G. M. Co, Ltd., died on the 9th of September of miners' phthisis.

Mr. Streeter was elected a member of the Society in January, 1909.

Mr. **Alexander Holmes Hartley**, the assayer of the New Goch Gold Mines, Ltd., died in Durban on the 27th. September, after an operation for some internal trouble. The deceased was one of the older members of the Society, having been elected a member in June, 1896. He was a contributor to the discussions on several metallurgical papers read before the Society, and in May, 1899, submitted an original paper, "Notes on the Precipitation of Gold from Cyanide Solutions." Mr. Hartley, who was 51 years of age, was the son of the late William Hartley, J.P., of Annadale, Co. Down, Ireland, and leaves a widow and two young sons to whom the Society tenders its sincere sympathies.

Notices and Abstracts of Articles and Papers.

METALLURGY.

Reduction Works of the Randfontein Central.—"*Metallurgical Treatment.*—One departure has been made in the existing methods of recovery by plate amalgamation. It was felt by your technical advisers that the general system of plate amalgamation in coarse crushing and tube milling plants was susceptible of improvements, and they therefore made extensive investigations, beginning in Feb., 1909, at the North Randfontein Gold Mining Co., with a view to carrying this into effect. The experiments conclusively demonstrated that no provision for amalgamation should be made until after the pulp has passed the classifiers, and that owing to the fineness of the sand in the overflow from the classifiers the area of the copper required is greatly lessened, while in the tube mill circuit all that is necessary is sufficient plate area to prevent enrichment of this circuit. This provides a method of treatment that is altogether superior to existing practice, and it has had its commercial value proved by months of daily usage in all the four mills on the Randfontein South G. M. Co. Incidentally, it is also found that there is no necessity to give a shaking motion to the tube mill plates. The net result is, that while the extraction is slightly improved, the capital cost is greatly reduced, the aggregate crushing time of the mill is increased, there being no stoppages required for dressing plates, the whole of the amalgamation is done in one building, and the maintenance and power costs are reduced through the disappearance of the machinery required for shaking the tables, and it becomes possible for the plant as a whole to be kept in a far better state of repair, owing to the gold recovery and the maintenance of the plant being each attended to by differently trained men. In view of the importance of this subject, many metallurgists and engineers have visited the mills of the Randfontein South, with a view to studying the method and results. The new system has now been running satisfactorily at each of the four mills for several months past and is now freely adopted on other groups. The railway system converges into a terminus at the reduction works, all rock from the shafts being dumped at this point into a large receiving bin, from which it is taken by conveyor belts into the classifying, sorting and crushing station, where there are six lines of sorting belts, delivering into the six jaw crushers, which are the largest that have yet been used on the Witwatersrand. Each line acts independently of the other, and under normal conditions, the whole of the plant here, as elsewhere, will be kept at work, but the capacity has been so arranged that in the event of any section having to be shut down for repairs the remainder can deal temporarily with the resulting overload. The waste rock is taken by a mechanical haulage to a separate dump to the east, while the product for the mill will be taken to its destination by conveyor belts.

The Mill and Tubes.—The mill itself consists of 600 stamps, each weighing 1,650 lb., arranged in batteries of 10 stamps in two lines, the mortar boxes being set on massive concrete blocks. The main building is a composite structure of steel and timber, covering an acre of ground, the length of the building being 630 ft., but the mill bins are entirely of timber framing, lined with steel, and these will hold 34 hours' supply of ore. In front of the mill, and in the centre of its length, is the tailings pump-house, where there will be six pumps for the coarse sand and four pumps for handling the fine sand, all of the pumps being of the centrifugal type. In front of this again is the tube mill building, which is a steel structure, also covering an acre of ground; in it are 16 tube mills placed in two parallel rows of eight, and arranged between the tubes are all the amalgamating plates. The reduction officer's headquarters are in this building, enabling him to supervise simultaneously all the work going on in the plate and tube mill house, at the sand and slime plant, at the residue dump, while the pump-houses, extractor-house, mill and crusher station are easily accessible, being but a short distance from his office.

Sand and Slime Plants.—The sand plant is arranged in two rows, each row being almost at right angles to the battery, and this plant consists of 35 steel tanks, 60 ft. in diameter, of which 12 tanks are for collecting the sands, and these are arranged at a higher level than the treatment tanks; the system of transferring the sand will be by means of trucks, and the residues will be taken to the tailings dump by mechanical haulage. As the sand must be all got rid of in the hours of daylight, the quantity reaching the large total of 400 tons per hour, two dumping faces will be provided, which will gradually converge into one as the dump grows larger. Between the two lines of sand tanks is arranged the slime plant, consisting of 23 tanks each 70 ft in diameter, of which six are collectors, and situated in the middle of the slime plant is the slime pump-house, where the pumping operations connected with circulation, transferring and emptying the charges is concentrated. Immediately below the slime plant are the extractor-house and solution sumps; the extractor-house, besides being equipped with the most modern plant, also contains the furnaces for the smelting of the cyanide gold, and the whole is amply safeguarded against theft. In connection with all the above machinery and plant, the necessary appliances for rapid handling and replacing of parts have been amply provided for." — DAVID GILMOUR. — *South African Mining Journal*, Feb. 12, 1910, p. 609. (A. R.)

FILTER FRAME AND CLOTHS FOR LEACHING TANKS.—"In the designing of filter frames for leaching tanks every inch or less of vertical height is of considerable moment when it is considered that all absolutely unnecessary space occupied by the filter frame represents so much available space lost just as many times per month as the tank or tanks in question are filled. As in many leaching plants (cyanide or other kind) there is a considerable tendency to use shallow tanks, the space occupied by the filter frame represents quite a considerable proportion of the total available space in the tank. The writer has found that in practical work one easy way at least of always having the total possible filtering area, and always having the filtering cloths free from slime or hard packed sand, and never after any length of time having slimes or sand passed through the cloths is as follows:—

The wooden filter frames are put in as usual in sections of convenient size, and the complete circle or rectangle, as the case may be, is centred by wooden wedges between the side of the tank and the caulking edge of the frame, thus leaving a caulking space about $\frac{7}{8}$-in. wide. Cocoanut matting to exact size of the filter frame is laid down; this matting need not be sewn into one piece, but can be in pieces of convenient size lightly held together, and with a few stitches here and there. On top and all over this matting is laid jute cloth or duck in sections overlapping each other by a few inches, and, if preferred, loosely held by stitches here and there so as to form a lap joint. This jute cloth or duck is caulked tight into the space between the caulking edge of the frame and the side of the tank with $\frac{3}{4}$-in. manilla rope. The jute cloth or duck should be cut full and large enough, and should lie quite slack above the cocoanut matting in the tank so as to allow for shrinking, also several inches of spare end of cloth should appear above the caulking rope, thus insuring a good caulking joint. This jute cloth or duck forms the fixed filter cloth. On top of this in similar sections is laid a similar cloth with plenty of slack in it everywhere. This latter cloth need merely be tucked in by hand into the caulking space. This constitutes the loose or easily replaceable cloth. No shovelling laths (which diminish filtering area and get hard unleachable sand packed between them) are put on top of of the cloths, nor is any means used to protect the cloths, except that a constant small quantity of sand is always left in the tank on top of this upper loose cloth.

If at any time leaching is not as it should be, all sand is removed from upper cloth and the cloth is removed, cleaned, and replaced again, or a similar ready spare cloth is put in. This can be done very quickly seeing that there are no shovelling laths to move and replace, and because the cloth is in sections that can be handled by one man, and is not a large unwieldly sewn cloth of, say, 40 ft. or more diameter.

It is not found in cases where shovels are used for discharging that the upper cloths get cut; this is partly because the cloth is laid slack and not stitched tight, and partly because an inch or less rather of sand is always left.

The cloths can be kept in perfect filtering condition, and there is no lost filtering area, due to shovelling laths and packed sand between them, and as a result it is found in the case of sand being leached by cyanide that on discharging the residue the assay value is practically the same, regardless of whether the sample is taken from the upper or lower portion of the tank; this cannot be the case when the filter arrangements are imperfect, or of unequal resistance in different portions of the filtering area.

In order to be sure of keeping cloths and the space under them perfectly clean, it is essential in the case of a tank freshly filled with sand that before pumping on solution sufficient solution should be put in from the bottom through the leaching pipe so as to fully cover the upper filter cloth. It is also important that in case of a dry tank solution should not be pumped on when the leaching cock is open. With these latter precautions, the writer has found it impossible to get solutions leaching cloudy, or to get solid matter through the filters. If the upper loose cloth is cleansed occasionally, the fixed jute cloth (or duck) and the cocoanut matting will not require cleaning, or attention, or lifting till such time as, through rotting, they are too weak to carry the charge weight.

Some operators prefer to do without cocoanut matting; in this case another jute cloth (or duck) may take its place, but this is not necessary except with very deep tanks.

These precautions, etc., may sound totally unnecessary, but there is no doubt that the actual leaching and filtering arrangements are, through stress of work, not always what they should be.

If all the lower portion of the charge in the tank does not, after treatment, assay practically the same as the central or upper portions (that is assuming the charge to be divided for sampling purposes into three equal horizontal layers) then improvement may be possible.

In the case of a new charge starting treatment (other things being equal), the metal value of the leaching solution should rapidly rise to its highest value, and then regularly fall to its lowest value at the time of discharge, when treatment is finished.

If the leaching solution values are plotted they should show rapid attainment to the highest metal contents, and then the gradual and regular diminution of leaching solution values. Granted that there is no other cause for irregularity, bad filters alone are sufficient to cause irregular leaching values."—*The Mining Journal*, Feb. 12, 1910, p. 197. (W. A. C.)

FLOTATION PROCESS FOR THE CONCENTRATION OF TIN ORES.—"The Elmore oil flotation process has recently found a new application to the concentration of tin ore. There is an old tin mine in Dolcoath, Cornwall, where for many years past those parts of the workings have been abandoned which contain mixtures of the ordinary tin oxide (cassiterite) with sulphides of various metals, such as copper, zinc, iron and arsenic. Such mixed ores were valueless, because if concentrated on water-dressing appliances, the sulphides, or, at any rate, a large proportion of them, were collected with the tin oxide, and the resulting product was too impure for the tin smelter. For the past two years an Elmore vacuum plant has been in operation there with complete success. The complex ore, after crushing, is treated in the vacuum machine, with the result that practically all the sulphides are removed as a concentrate which is almost free from tin, and substantially all the tin is left in the tailings from the Elmore machine, these tailings being passed on to usual water-concentrating appliances for the recovery of the tin, which is found in practice to be practically free from sulphides. Some of the Dolcoath tin ores are free from impurities except copper sulphide. From this ore the vacuum process produces a high-grade copper concentrate, in some cases as high as 19% of copper, which is readily saleable to the copper smelter.

Following the success at Dolcoath the process is about to be applied to two other widely separated parts of the world, viz., the Straits Settlements and South Africa. With the tin concentrates from South Africa a working trial of the Elmore process on about 35 tons of this material has recently been carried out, the results being given below.

	S %	Cu %	As %	Sn %
The impure tin concentrate treated assayed	1·69	0·26	2·91	72·38
The sulphide product assayed	20·77	2·78	33·0	3·64
The cleaned tin product assayed	0·12	0·03	0·14	74·9

The loss of tin in the sulphur product amounts to only 6 lb. per ton of material treated, while 95% of the total impurities have been removed."—E. S. ELMORE.—*Mining Journal*, Aug. 28, 1909; *Metallurgical and Chemical Engineering*, April, 1910, viii. 4, p. 204. (J. A. W.)

CYANIDING SILVER ORES IN MEXICO.—"The ores from the San Rafael mines are derived from a system of veinlets which form the Vizcaina vein, or mother lode, of the district. They contain from 70 to 75% silica and from 10 to 20% calcite as gangue. The silver exists mostly as sulphide Ag_2S associated with varying proportions of iron sulphide FeS_2, lead sulphide PbS, and zinc sulphide ZnS. The gold occurs in nearly constant proportions of 4 or 5 gm. per ton in high and of 3 or 4 gm. in the low-grade ores. All ores carrying more than 300 gm. of silver to the ton are cyanided.

The ore is sorted by hand; crushed in Blake crushers, and sampled in the mine yard, after which the treatment is as follows: (1) Pulverised with stamps in a solution containing 2.5 kgm. of potassium cyanide to the ton of water; (2) concentrated on Wilfley tables; (3) the tailings are classified; (4) the sands from the classifiers are ground fine in Krupp tube mills; (5) the material from the Krupp mills is settled in pulp thickeners; (6) the thickened pulp is agitated in Pachuca pneumatic tanks; (7) the slime is separated from the solution by Moore vacuum filters; (8) the gold and silver in the filtered solution is precipitated by zinc filaments. The mill is driven by electric power, the motors for the most part being of the C.C.L. type of the Westinghouse company. Each 20 stamps has a 75 h.p. motor, and each Krupp mill a 100 h.p. motor The water supply comes from the mine and is slightly alkaline. The surplus water from the filter plant is returned to the water tank, which has a capacity of 400 cub. meters.

The pumps used are of the vertical triplex power type with solid water end. The product of the tube mills and classifiers is handled with 5 pumps 10 in. × 54 in. The filter plant is served by one centrifugal pump, and the filters are worked by two Gould vacuum pumps, 14 in. × 14 in., although a third vacuum pump is held in reserve.

There are 80 stamps in use, 40 weighing 850 lb. set on timber foundations; and 40 weighing 1,250 lb. bolted to concrete foundations. The lighter stamps drop 7¾ inches, the heavier 6¾ inches, and both drop 104 times per minute. The El Oro solid guides with back plates are in use. The shoes and dies, which are of forged steel, last from 90 to 100 days.

The 850 lb. stamps crush from 3 to 3½ tons of ore daily through 10-mesh, 18-wire, 1·32-millimeter aperture screen.

The 1,250 lb. stamps crush from 6 to 6½ tons daily through the same mesh screen, and from 5½ to 6 tons daily through 12-mesh, 20-wire, 1·067-millimeter aperture screen. The total capacity of the stamps through 10-mesh screen is from 300 to 400 tons per day, but at present crushing is limited to 350 tons to prevent overloading the tube mills. From 7 to 8 tons of solution is used in the mortars per ton of ore crushed. One pair of rolls 36 in. × 16in. and two additional tube mills have been ordered, and when these are installed 8-mesh screens will be placed in the mortars, and it is expected that the capacity of the stamps will be raised to 500 tons of ore daily.

The following is a sizing test of the battery pulp when using 10-mesh screens of 1·32-millimeter aperture: +40, 37%; +60, 9%, +100, 7%; +150, 13%; +200, 4%; −200, 30%.

The total consumption of power in the mill is 1·68 horse-power per ton of ore treated, of which 1·05 horse-power is consumed in crushing. The mill solution is contained in two tanks 24 ft. × 20 ft. having a capacity of 200 cubic meters each. These tanks are used alternately to supply the stamps, Wilfley tables, classifiers and tube mills, and to receive the barren solution, being worked in cycles in order to avoid enrichment of the solution. With this system the working solution contains not more than 50 grams of silver per cubic metre. According to the class of ore treated, from 6 to 12 kilos of lime is added in the bins per ton of ore. The alkalinity of the solutions is kept at 1 kilo of calcium oxide *CaO* per ton of ore.

The mill solution averages ·25% potassium cyanide KCN to the ton; and there is a cyanide loss of about 600 grams of KCN per ton of ore treated in the stamp battery. The extraction of silver and gold in the crushing operations is high, averaging one month 38·3% of the silver and 70% of the gold contained in the ore.

One Wiltley table takes the pulp from 5 stamps. The 16 tables employed each receive from 17 to 30 tons of ore per day without being overloaded. The tables are also used as auxiliary classifiers, the slime overflow being sent to the pulp thickeners.

When the tables receive material at the rate of 30 tons daily, about 22% of the mineral is recovered as concentrate which contained 1 month: Silver, 25·586 kilograms per ton; gold, 126 grams per ton; iron, 34·45%; silica, 12·25%; lime, 3·40%; sulphur, 40·50%.

The concentrate is sold to the smelter, and, as previously stated, the tailing goes to the classifiers, five of which discharge their product into one tube mill. The proportion of solid to solution is at this point of the system about 1 to 2·3, but as it leaves the classifiers the proportion is about 1 of solution and 2 of solid. From 50 to 70 tons of pulp and all the return from the tube mill are fed to each classifier. The classifiers discharged the following material during 1 month: 2·5% remained on 100 screen, 15·5% remained on 150 screen, 7·5% remained on 200 screen, 74·5% passed a 200 screen.

The five Krupp tube mills, which are 4 ft. in dia. and 20 ft. long, have Neal's baffle at both ends and El Oro liners. Each of the present mills is supplied with 100 horse-power motor, but this being unnecessary the new mills will be supplied with 75 horse-power motors. The original steel lining of the Krupp mill lasted 90 days, while the El Oro lining lasted 8 months.

About 350 tons of material pass through the 5 mills daily, but their capacity for regrinding battery sand is rated at 50 tons daily, the heads averaging: 48% on 100 screen, 39·6% on 150 screen, 4·2% on 200 screen, 8·2% passing 200 screen.

The sand entering the mill contains about 60% of moisture, and but 8·2% is finer than 200 mesh. The slime discharged from the classifiers had 74·5% that would pass a 200 mesh.

The pulp leaving the tube mill and entering the pulp thickeners has a consistency of 1 dry slime and 10 solution, but leaves the thickeners at 1 dry slime and 1·2 solution. The pulp is unwatered previous to agitation in the Pachuca tanks. The pulp thickening or unwatering tanks are five in number; three 20 ft. in dia. and 10 ft. high receive 70 tons dry pulp each, and two 24 ft. in dia. and 10 ft. high receive 100 tons each per day. The solution from the tube mills is raised to the unwatering tanks by 3 pumps, two 8½ in. × 10 in. and one 7 in. × 9 in. The capacity of the two first pumps is 350 gallons per minute, that of the second pump 250 gallons per minute.

Ten 15 ft. × 45 ft. Pachuca tanks are used to agitate the pulp at present, and two more have been ordered. All the pulp deposited in the Pachuca tanks is intended after agitation to go to the filters. The Pachuca tanks have a capacity of 183 cubic meters and are given an average charge of 100 tons of dry slime and 120 tons of solution. The proportion of 120 tons of solution and 120 tons of slime was tried, but it was found difficult to discharge the unwatering tanks, and the proportion of 1 part slime to 1·2 parts of solution was found more satisfactory. The charges mixed in the proportions of 1 dry slime to 2 and 3 of solution were also tried but gave more or less trouble in filtration.

For agitating the slime in the Pachuca tanks, and to supply the Moore filter tanks, one Ingersoll-Rand air compressor with double cylinders, 18¼ in. dia. and 16 in. stroke, delivers 700 cub. ft. of air at 9,000 ft. elevation, when making 100 revolutions per minute. When making 130 revolutions per minute the capacity of the compressor is 900 cub. ft. This quantity of air is however not required until the capacity of the plant is enlarged. The volume of air required for energetic agitation in the Pachuca tanks is estimated at 100 cub. ft. per minute with pressure at 25 lb. per sq. in. The pulp is agitated 36 hours with ·3% KCN solution and allowed to stand at rest from 12 to 24 hours, as it has been found that during this period the silver extraction increases about 20 grams. The compressor receives power from a 100 horse-power motor. There is about ·18 horse-power per day consumed per ton of ore agitated by air. Before agitation is commenced the solution is brought up to ·3% KCN and 250 grams of acetate of lead per ton of ore is added. The average value of the solution and mineral in the Pachuca tanks in October was 337 grams silver and ·33 gram of gold per ton. After the material had been treated there were 71 grams silver, and after the slime had been discharged from the filter it contained 55 grams of silver per ton.

The filtration plant contains two units of Moore A No. 3 type of vacuum filters; the filter leaves which are 10 ft. × 6 ft. have a capacity of 2·5 tons of dry slime per day. There are 160 filter leaves in operation and 80 held in reserve. The leaves are arranged in four baskets of 40 each. Each unit has three tanks 12 ft. × 22 ft. × 7 ft. for the filter leaves, and each tank has a capacity of 75 cub. meters.

In order to move the baskets there are two 25-ton capacity travelling cranes, each supplied with a 30 horse-power variable speed General Electric Co. motor for lifting, and a 5 horse-power motor for travelling and transferring the baskets.

Filtration is divided into three periods: the first, formation of the slime cake, which requires from 20 to 30 minutes to become 1 in. thick, according to the condition of the filters. In the second period 1 hour is consumed in washing the cake with barren solution. The third period is a 15-minute water wash given to the cake on the filters.

The first tank contains the agitated slime pulp to be filtered; the second tank contains the barren solution coming from the zinc precipitation boxes; the third contains water for washing out the barren solution, after which the cake is discharged from the filters and washed away. The discharged cake contains about as much water as slime, that is 1 to 1. On each basket is a spiral tube 16 in. in dia. and 20 ft. long, in which is maintained a vacuum to hold the slime cake on the filter leaves during the 5 minutes required to transfer the baskets from tank to tank. Each unit has a vacuum pump which draws the various solutions through the filter leaves. During October the water discharged assayed 4·5 grams of silver per ton; the cake washed assayed 50 grams and unwashed 55 grams per ton. In order to clarify the solutions, sand filters are used prior to precipitation in the zinc boxes.

Precipitation.—Zinc shavings, ·006 in. thick, are used in precipitation.

Two tons of solution to the ton of ore are precipitated.

The heads in the boxes assay from 200 to 300 grams of silver; the tailings, 2 grams.

The clean-up is made weekly.

The short zinc which remains on a 20-mesh screen is returned to the boxes ; the shorts on 40-mesh are melted.

The precipitate is pressed and dried to 20% moisture on a 24 in. × 24 in. Shriver press.

Melting.—Coke furnaces and Dixon's No. 300 graphite crucibles are employed.

The precipitate is melted with the following flux : 15% broken glass ; 6% borax glass ; 4% soda carbonate. The short zinc which remains on the 40 mesh screen is fused with 20% broken glass ; 8% borax ; 6% soda carbonate.

The bars, without remelting, assay from 920 to 950 grams silver, and 5 grams of gold to the kilo of silver.

The slags are crushed quarterly in a battery and concentrated on a Wilfley table. The concentrates are melted and the tails treated in the cyanide plant or sold.

RESUME.

The resume of the November results is given below :

Tons crushed, 8,393.

Assays :	
Ore contents :	
Silver	·901 kgm
Gold	4·430 gm
Concentrates :	
Silver	25·348 kgm
Gold	141·600 gm
Heads of Pachuca tanks :	
Silver	·397 kgm
Filter discharge :	
Unwashed, silver	·071 kgm
Washed, silver	·063 kgm
Extraction :	Per cent.
In concentration silver	24·19
Cyaniding :	
Crushing	31·74
Agitation, silver	36·18
Total extraction, by assays, silver ...	92·11
Total extraction, by concentn., silver	91·43
Total extraction, by concentn., gold ...	97·08
Consumption per ton of ore :	
Sodium cyanide, 128 per cent. ...	1·078 kgm
Lead acetate	·317 kgm
Lime	6·706 kgm
Flint pebbles	1·227 kgm
Zinc	·843 kgm
Coke	1·646 kgm
Borax	·084 kgm
Soda carbonate	·044 kgm
The cost per ton was :	
General expenses	$ ·78
Grinding and concentrating ...	1·18
Cyaniding	1·32
Pumps and compressor	·15
Filtration, including royalty ...	·33
Precipitation and melting ...	·44
Total, Mexican money ...	$4·20

—E. Girault, *Mining and Metallurgical Institute of Mexico.*—*Mines and Minerals*, May, 1910, p. 618. (A. R.)

Manufacture of Iron and Steel.—"1. There are large iron ore deposits in various parts of the Colony, but the most accessible, near Pretoria and Airlie, are, so far as outcrop samples indicate, very silicious, and although capable of producing a good pig iron would be expensive to smelt.

2. The limestone deposits suitable for blast furnace purposes are very limited, and owing to their occurrence the mining costs are high. There are large deposits of silicious limestone, but these could not be economically used alone for iron smelting, although to a limited extent they might be used in admixture with the more pure varieties.

3. It has not yet been definitely determined if a hard metallurgical coke suitable for blast furnace practice can be made from local coals, although experiments on a small scale give very promising results. In any case the coke would not be a high-class coke and it would be expensive, owing to the costs incurred when mining only selected portions of the coal seams.

4. Assuming a satisfactory coke can be made, pig iron suitable for foundry purposes and for the manufacture of steel by the Basic process can be made from Pretoria and Airlie ores, in admixture with not less than about 20% of rich non-silicious iron ores which exist in comparatively small quantities in several parts of the Colony. Such a pig iron would cost very considerably more than the same quality of iron produced in Europe.

5. Pig iron cannot be commercially manufactured in the electric furnace from local ores with current costing anything approaching the price at which it can be at present generated in the Colony.

6. Taking into consideration all the local conditions respecting the supply of raw materials, cost of labour, and especially the restricted nature of the present market, and the large variety of finished products it would be necessary to manufacture in order to maintain a fair output, I do not consider that an iron and steel plant to manufacture rails, sleepers, bars, wire, galvanised sheets, etc., could be commercially successful.

7. It would be quite impossible to export either pig iron or manufactured steel in competition with older iron-producing countries much more favourably situated. It would not be possible to compete with imported material near the chief seaports without a very substantial bounty from the Government or protection in some equivalent form.

8. If all the Railway Departments of South Africa would jointly agree to take substantial yearly quantities of rails and sleepers, at fair market prices, this would form a sound basis for the establishment of an iron and steel industry, which, with the assistance of a reasonable bounty, should expand as the requirements of South Africa increase.

9. A small electric furnace plant designed to produce high-class steel from the large accumulations of scrap in the Colony should give excellent results, and such a project should receive every possible support from the Government.

In view of the possibility of future developments, I would strongly recommend that systematic prospecting of the most accessible ore deposits be undertaken without delay, to prove them both as to quality and quantity, and that the question as to the possibility of producing a metallurgical coke be definitely settled by coking experiments in modern coking ovens."—F. W. Harbord.—*Transvaal Mines Department Report*, June, 1910, p. 17. (A. McA. J.)

Studies in Concentration.—"Theodore J. Hoover presents a discussion of various points in connection with the calculation of the proportion of recovery in concentration. He commences with a demonstration of his method of calculating the percentage of recovery and ratio of concentration when only the percentage assay values of the ore, concentrate and tailing are known, and not the relative tonnages. This is quite a simple mathematical problem. If we take x, y and z, as the undetermined weights of the ore, concentrate, and tailing, and a, b and c as the percentage of metal contained in each as ascertained by the assay of a sample, then the constitution of the ore, concentrate, and tailing, and the amount of metal contained in the three, are expressed by the two formulæ:

$$(1)\ x=y+z$$
$$(2)\ ax=by+cz$$

The ratio of concentration is expressed by the fraction $\frac{x}{y}$, and the percentage of recovery by $\frac{100\ by}{ax}$. Eliminating z from the two equations, by multiplying (1) by c, and substracting, we get $(a-c)\ x=(b-c)\ y$. Thus $\frac{x}{y}$, the ratio of concentration, is represented by $\frac{b-c}{a-c}$. Substituting this ratio in the formula for the percentage of recovery we get the percentage expressed thus: $\frac{100\,b\,(a-c)}{a(b-c)}$. These two formulæ are independent of the value of x, y and z. It will be seen also that if we have the percentage assay-values of the ore, concentrate, and tailing, we can deduce the relative amounts of concentrate and tailing for, say, 100 parts of ore. This is done by substituting 100 for x and $100-y$ for z in equation (2) and solving for y.

As a rule the percentage of recovery and the ratio of concentration are ascertained by measuring x, y, a and b. There are many occasions when this cannot be so conveniently and easily done as by this method based on sampling. Mr. Hoover points out that his method depends for its success on the degree of accuracy of the sampling and of the assay; we do not suppose that the possibility of error need be greater than when the ore and concentrate are weighed daily and their contents sampled. The two methods might with advantage be used as a check one on the other. Mr. Hoover proceeds to show how his method can be employed for studying the accuracy of analyses, and how errors are introduced into the figures for extraction by inaccurate samplers and chemists."—*Mining Magazine*, July, 1910, p. 74. (A. R.)

HIGH TEMPERATURE MEASUREMENTS AND GAS THERMOMETRY.—"Before the publication of the recent measurements in the Geophysical Laboratory, the region of accurate gas thermometry ended with the temperature 1,150°. The common practice for some years has therefore been to measure temperatures beyond that point by extrapolating the parabolic curve out to the desired temperature—often as high as the melting point of platinum (1,755°). Extended extrapolation is always a matter of grave uncertainty, but no other convenient method was available, and after all, observations of this kind with the thermoelement could be translated in terms of an observed scale whenever one should be provided. It happens in this particular case that the extrapolation (from 1,150° to 1,755°) comes out just 50° low at the platinum melting point.

Now that the gas thermometer is able to make direct measurements of temperature up to 1,550°, it is merely necessary to include several thermoelements in the furnace with it, to evaluate their temperature curves by direct comparison, and afterward to determine with them a series of standard fixed temperatures like the melting points of metals, minerals, or salts, which are available for general use.

This procedure has been followed in the recent investigation with the results which are tabulated below. The pure metals are readily obtainable and the minerals easily made up, so that anyone may now calibrate his thermoelement by noting its electromotive force at the (known) melting temperature of three or four of these substances, preferably choosing such as will include the region in which he expects to use the thermoelement.

Such a calibration requires to be checked from time to time by repeating one or more of the melting point observations, particularly if the element is exposed to contamination from vapours of other metals. This contamination is particularly disastrous if a thermoelement is exposed to the vapour of iridium, which explains the remark made above, that in our experience a bulb made from an alloy of platinum and rhodium is to be preferred to the iridium alloy in use at the Reichsanstalt.

The standard melting points in the table below are taken from observations by Day and Sosman and include, for comparison, the standard melting points published by the Reichsanstalt in 1900 and now in general use:—

Substance.	Point.	Atmosphere.	Crucible.	Temperature.	The Reichsanstalt Scale.
Zinc	Melting and freezing	Air	Graphite	418·2° ± 0·3	419·0°
Antimony	,,	Carbon monoxide	,,	629·2 ,, 0·5	630·6
Silver	,,	,,	,,	960·0 ,, 0·7	961·5
Gold	,,	,,	,,	1,062·4 ,, 0·8	1,064·0
Copper	,,	,,	,,	1,082·6 ,, 0·8	1,084·1
Diopside (pure)	Melting	Air	Platinum	1,391·2 ,, 1·5	
Nickel	Melting and freezing	Hydrogen and Nitrogen	Magnesia and magnesium aluminate	1,452·3 ,, 2·0	
Cobalt	,,	,,	Magnesia	1,489·8 ,, 2·0	
Palladium	,,	Air	Pure magn'sia	1,549·2 ,, 2·0	1,575·2
Anorthite (pure)	Melting	Air	Platinum	1,549·5 ,, 2·0	
In addition, the following temperatures were incidentally obtained:					
Cadmium	Melting and freezing	Air	Graphite	320·0 ,, 0·3	321·7
Aluminium	Freezing	Carbon monoxide	,,	658·0 ,, 0·6	657·
Platinum	Melting	Air	—	1,755	

A new estimate of the melting point of platinum, which has not yet been directly determined with the gas thermometer, is included in the above list. It was obtained in this way: There is a remarkably close agreement between independent determinations of the temperature interval between the melting points of palladium and platinum:—

Nernst and von Wartenberg	204°
Holborn and Valetiner (at the Reichsanstalt)	207°
Waidner and Burgess (at the Bureau of Standards)	207°

If we, therefore, simply add 206° to the palladium melting temperature (1,549°), we obtain 1,755° as the melting point of pure platinum, with an absolute error of perhaps no more than ± 5°.

By way of conclusion, the following estimate is offered of the accuracy of existing standards of temperature within certain temperature intervals :—

0° – 100°	± 0·002°
100° – 300°	,, 0·05°
300° – 1,100°	,, 0·8°
1,100° – 1,550°	,, 2°
1,550° – 1,750°	,, 5°.

—A. L. DAY.—*Metallurgical and Chemical Engineering*, May, 1910, Vol. viii., 5, p. 260. (J. A. W.)

THE SILICA FILTER IN SLIMES TREATMENT.—"An unusually interesting paper on an improvement in cyanide practice has been contributed by Mr. E. Gybbon Spilsbury to the transactions of the American Institute of Mining Engineers. It relates to a porous diaphragm which has been found of great value in the aeration of slimes, and has been successfully used for revolving filters and other purposes, to which a material of that kind is obviously well adapted. The following notes refer to the employment of the filter in slimes treatment :—

'The recovery of gold and silver from their ores by means of the cyanide process has been so successful in the last few years that any radical improvement would seem impossible ; yet the appliance to which I wish to call attention in this paper is really a radical departure from the methods now in general use.

'The most modern and approved of these, known as the all-sliming method, depends for its success on the grinding of the ore so fine that practically 90% of it will pass through a 200-mesh screen. The slimes thus produced are then agitated and aerated in tanks of various types.

'The main object of this treatment is to insure such a thorough admixture of the pulp and solution that every particle of the ore is surrounded by a volume of solution sufficient to insure the dissolving of the whole of the gold and silver content. In addition, in order to expedite the action of the cyanide, and to oxidise such elements as would, if left in their active state, become cyanicides, air is blown in under pressure. In the Pachuca tank, this air is the active and sole means of agitation.

'The chief objections to all the methods of agitation used heretofore are the expense of operating and maintaining the mechanical devices for keeping the pulp in suspension, and the length of time required to obtain a fairly complete extraction of the values. These objections are inherent to any method of agitation effecting a circulation of the pulp and the solution together by giving the whole mass a circular movement, as in the low-tank system, or a verticular circulation, as in the Pachuca-tank system. In both methods, the particles of ore are kept travelling in the same direction as the solution, and with very little difference of speed ; so that, while the whole mass is in violent motion, the relative positions of the ore-particles to their surrounding medium of suspension change but very slowly, and consequently the length of time required for a given extraction is much greater than would be necessary if the movements of the solution and the pulp were not coincident.

'The improvement here described is a purely mechanical one, devised to meet this requirement. It rests simply on the discovery of a method of manufacturing a diaphragm of silica sponge, which, while strong enough to support heavy weights, is so evenly porous throughout that air can be passed through it with practically little resistance, and in which, nevertheless, the pores are so minute that no solid matter, however, finely divided, can pass through or even into it.

'In practice, this diaphragm is placed in the tank as a false bottom, resting on light channel-iron bars, 4 in. above the real bottom. The plates are either 12 × 12 or 12 × 20 in. in size, and are secured to the channel-iron supports, along the lines of intersection, by 0·25 in. carriage bolts. When the plates are all laid, oakum is driven into the joints, which are then made completely tight by pouring in liquid cement.

This simple operation completes the whole installation. The charge is now run into the tank, and air is admitted from below, under a pressure of from 2 to 6 lb. only, depending on the depth of the charge. Immediately the whole charge becomes a seething mass of uniform but gentle agitation, in which every particle of ore is in constant motion.

'No pressure is exerted on any of the air particles after they have passed through the pores of the diaphragm. They simply levitate up through the mass by reason of their lower specific gravity. No distinct streams or lines of agitation are preceptible to the eye, but the charge shows a very distinct increase of volume, amounting to more than a foot of height in a 6 ft. tank : and the surface becomes covered with a coating of foam or air-bubbles, the thickness of which depends on the volume of air blown through the diaphragm. The full capacity of the sponge varies from 5 to 5·5 cub. ft. of air per minute per square foot of area at 1 lb. pressure.

'Under these conditions the action of the cyanide is very rapid and intense. In 80% of the runs made an extraction of over 50% of the combined gold and silver values has been obtained within the first hour of agitation ; and while, in experimental work, our treatment is usually carried on for 12 hours, six hours will probably be found to be the economical period in general practice.

'It is found that the consumption of cyanide per ton of ore treated under this method is much less than in either the Pachuca or the mechanically-agitated tanks. This, I believe, is due chiefly to the briefer exposure of the cyanide to the oxidising effect of the air, but also to the circumstance that we are able to treat effectively a much thicker pulp than the other methods of agitation will permit. With a proportion of 1·5 of solution to 1 of ore, we can obtain the quickest extraction ; but, for facility of charging and discharging tanks, we generally make the mixture 2 to 1.

'Records of work done with this porous diaphragm on a commercial scale, in one of the large mills of the Guanajuato Development Co., show what remarkable extractions are obtained by this method.

'One of the important questions we had to study in the use of this material was, whether the pores of the sponge would not sooner or later become filled and choked by the very finest particles of ore, thus impairing the efficiency of the plates.

All our experience hitherto goes to prove that no such stoppage need be feared. The pores are so minute and so irregular in shape that apparently no solid matter can find entrance. In treating certain classes of ore we do find that, after a certain number of hours, the air-pressure begins to increase, by reason of a closing of the surface-pores of the diaphragm ; but examination under the microscope has shown that this is due to a gradual deposit of lime carbonate on the surface of the plates, formed at the moment of contact with the air with lime in solution. The removal of this coating, however, offers no difficulty. It can be done, between charges, either by sweeping the surface with a wire broom or by washing it with a weak solution of hydrochloric acid. In either case, the complete removal of the deposit takes place instantaneously, and the air-pressure drops to the normal.

'The credit for the successful application of this porous medium to the cyanide process is due to J. E. Porter, of Syracuse, N.Y., who, having acquired the material for an entirely different purpose, conceived at once its possibilities in the cyanide field, and, by a long series of careful experiments, developed the many advantages of its present adaptation to that field.

Besides the employment of this silica sponge in the treatment tanks, its usefulness has been demonstrated in the filtering of the solutions, and as a clarifier. Several types of silica-sponge filters are now building of which probably, the simplest consists of a table, from 25 to 30 ft. long and 8 ft. wide, the top of which is an air-tight pan, connected with an exhaust pump. The pulp being run over the table, the solution is drawn through, leaving a dry cake of the desired thickness. Wash-water is then flowed over this cake, and it is washed in the usual manner. When the cake is finally dried, the table is tilted to a vertical position, and the cake is blown off by air-pressure. The mineral sponge always contains a certain amount of moisture, which causes a film of water to exude when the air-pressure is turned on: and this film, acting as a lubricant, aids the cake to free itself, so that the separation is immediate.

This filter has a capacity of from 40 to 50 tons of dry pulp per day. The resulting cake contains less than 23% of moisture when discharged. The simplicity of construction of this filter, its indestructible filtering medium, and the absence of the innumerable valves and fittings required by all filters of the leaf-type, will recommend it strongly to mill men generally. I think it will increase the saving of values wherever it is introduced."—E. GYBBON SPILSBURY, *American Institute of Mining Engineers.* — *South African Mining Journal*, June 18, 1910, p. 461. (A. R.)

MINING.

THE RELATION OF COSTS TO PROFITS.—"*Hypothesis.*—Minimum waste milled increases the costs and profits per ton milled (waste and reef), but the profits are increased in a greater ratio, and, although the pay limit is higher, some of the unpayable ore reserves becomes payable, or maximum reef milled decreases the costs and increases the profits per ton of reef milled. and some of the unpayable ore reserves become payable.

Introduction.—Undoubted a new era has been heralded on the Rand recently by the commendable reductions in working costs, and the pace has been set so strong that it may be wise to pause and endeavour to ascertain whether profits per unit of gold may not be sacrificed on the high altar of costs. I am fully cognisant of the fact that this subject has received much attention, but I am not aware of any similar table having been prepared.

Basis of Calculations.—I have taken as the basis of my calculations a mine which has 300 claims, which contain two reef bodies on a 30° dip. The average reef thickness of each ore body is 24 in., which will produce a value of 6 dwt. over a 60 in. stope width. The reduction plant consists of 200 stamps and four tube mills, with a crushing capacity of 40,000 tons per month. From this starting point I have calculated what would result if the external waste (waste above and below the reef channel at the stope face) were gradually reduced from 36 in. to 12 in.—vide columns A, B, C, D, E, F, G,—and at the same time the maximum amount of waste was sorted out on the surface, which will ensure minimum waste or the maximum reef being milled. Columns A and G are examples of two extreme cases, but they have been purposely included in the accompanying table to emphasise the principle I am endeavouring to establish.

Waste Sorted out on the Surface.—The basis mine (A) sorts 20% of waste on the surface from stopes of an average width of 60 in. The C mine sorts 20% also, but reduces the stope width to 54 in. by leaving 6 in. of waste in the hanging and foot, which, when compared to the basis mine, is equivalent to sorting 10% on the surface; this, together with the 20% sorted, makes an equivalent of 30% sorted compared to the 20% of the mine taken as the basis. Mine G only sorts 10% on the surface, but through reducing its external waste to 12 instead of 36 in., is credited with 50% less waste milled.

Ton Duty in Stopes.—I have assumed, for simplicity, that all the rock is broken by machines, and that if the duty in the 60 in. stope were 8·75 tons it would gradually drop to 5·25 tons in a 36 in. stope. I shall provide the reason of this assumption later.

Lb.-inch Duty.—The above ton duty is based on an equal efficiency—that is 292 lb. broken per in. of stope width. The term in. dwt. (should be dwt. in.) is familiar on the Rand, and also, I consider, that as efficiency is the only criterion of work done that the term lb. in. should replace the ton duty. This latter comparison is dependent mostly on the stope width and the duties in two different stope widths are not comparable, but if the ton duty is divided by the stope width and expressed in lb. for convenience, the resultant factor is a direct reflex of efficiency. The formula is:—

Lb. in. duty = lb. broken per in. of stope width per machine per shift.

$$= \frac{\text{Fathoms} \times 36 \times \text{Stope width in in.} \times 2{,}000 \text{ lb.}}{12 \times 12 \times \text{Machine shifts} \times \text{Stope width in in.}}$$

$$= \frac{500 \times \text{fathomage}}{\text{Machine shifts}}$$

One of the most uncertain factors in determining the ton duty is eradicated in the calculation, viz., the stope width. In a similar manner the efficiency cost of breaking ground is expressed by the formula:

$$\text{Cost per ton in} = \frac{80 \times \text{cost}}{\text{Fathoms}} \text{shillings}$$

which also does away with the stope width factor and gives a true reflex of efficiency.

Tons Mined.—In order to mill 40,000 tons per month, the 60 in. mine would mine 50,000 tons, when 20% was sorted, and the 36 in. mine would mine 44,444 tons, as 10% only, or 4,444 tons are sorted on the surface. The first advantage is here noted that 5,446 tons less would be hoisted, etc.

Feet Developed.—As the stope width decreases the lateral development must be increased to develop an equal tonnage. My argument here is that the tons developed should equal the tons stoped plus 25%, inasmuch as 5% of the tons developed would be occupied by pillars, which could only be extracted at a later date; 10% is represented by worthless bodies of reef exposed; and 10% for contingencies, such as dykes, faults, etc. It is, therefore, seen that the development footage would be increased from 1,561 ft. per month to 2,219 ft., and since the tonnage per

Stope widths (in inches)	60—A.	60—B.		54—C.		54—D.		45—E.		45—F.		36—G.	
	Mine taken as the basis.	Costs and Profits.	Equal Profits.	Costs and Profits.	Equal Profits.	Costs and Profits.	Equal Profits.	Costs and Profits.	Equal Profits.	Costs and Profits.	Equal Profits.	Costs and Profits.	Equal Profits.
Percentage sorted on surface	20	30	—	20	—	25	—	15	—	20	—	10	—
Equivalent percentage waste not milled (through decreased stope width)	—	—	—	10	—	10	—	20		20	—	40	—
Percentage decrease of waste milled	20	30	—	30	—	35	—	35	—	40	—	50	—
External waste above reef in stopes	19	18	—	15	—	15	—	12	—	12	—	6	—
Reef	24	24	—	24	—	24	—	24	—	24	—	24	—
External waste below reef in stopes	18	18	—	15	—	15	—	12	—	12	—	6	—
Ton duty	8·75	8·75	6·74	7·87	6·22	7·87	5·69	7·00	4·92	7·00	4·50	5·25	2·63
Tons stoped	46,829	53,643	—	46,483	—	49,651	—	43,418	—	46,137	—	39,940	—
Tons mined	50,000	57,143	—	50,000	—	53,333	—	47,059	—	50,000	—	44,444	—
Tons milled	40,000	40,000	—	40,000	—	40,000	—	40,000	—	40,000	—	40,000	—
Tons developed	62,439	71,524	—	61,978	—	66,201	—	57,891	—	61,516	—	53,253	—
Feet developed	1,561	1,788	—	1,722	—	1,855	—	1,809	—	1,922	—	2,219	—
Value of tons from stopes	6·00	6·00	—	6·59	—	6·59	—	7·33	—	7·33	—	9·54	—
Increase in value	—	—	—	·59	—	·59	—	1·33	—	1·33	—	3·50	—
Value of tons milled	7·05	7·99	—	7·73	—	8·19	—	8·08	—	8·54	—	9·76	—
Value of yield	6·63	7·51	—	7·30	—	7·74	—	7·68	—	8·11	—	9·37	—
Value of yield (shillings)	27s. 11d.	31s. 6d.	—	30s. 8d.	—	32s. 6d.	—	32s. 3d.	—	34s. 1d.	—	39s. 4d.	—
Increase in value	—	3s. 7d.	—	2s. 9d.	—	4s. 7d.	—	4s. 4d.	—	6s. 2d.	—	11s. 5d.	—
Stoping per ton stoped	4s. 6d.	4s. 6d.	5s. 10d.	5s. 0d.	6s. 4d.	5s. 0d.	6s. 11d.	5s. 8d.	8s. 0d.	5s. 8d.	8s. 9d.	7s. 6d.	15s. 0d.
Shovelling and tramming, per ton mined ..	1s. 6d.	1s. 6d.	1s. 6d.	1s. 8d.	1s. 9d.	1s. 8d.	1s. 9d.	1s. 9d.	2s. 0d.	1s. 9d.	2s. 0d.	2s. 0d.	2s. 6d.
Development per ton mined	1s. 9d.	1s. 9d.	1s. 9d.	1s. 11d.	1s. 11d.	2s. 0d.	2s. 0d.	2s. 2d.	2s. 2d.	2s. 2d.	2s. 2d.	2s. 9d.	2s. 9d.
Total costs, per ton milled	16s. 1d.	18s. 0d.	19s. 8d.	17s. 3d.	18s. 10d.	18s. 3d.	20s. 8d.	17s. 7d.	20s. 5d.	18s. 5d.	22s. 3d.	19s. 6d.	27s. 6d.
Total profits, per ton milled	11s. 10d.	13s. 6d.	11s. 10d.	13s. 5d.	11s. 10d.	14s. 3d.	11s. 10d.	14s. 8d.	11s. 10d.	15s. 8d.	11s. 10d.	19s. 10d.	11s. 10d.
Percentage increase in costs	—	12	—	7	—	14	—	9	—	15	—	21	—
Percentage increase in profits		14	—	13	—	20	—	24	—	32	—	68	—
Cost per ton of reef	32s. 2d.	31s. 7d.	—	31s. 1d.	—	30s. 10d.	—	30s. 0d.	—	29s. 7d.	—	26s. 4d.	—
Profit per ton of reef	22s. 11d.	23s. 6d.	—	24s. 4d.	—	24s. 7d.	—	25s. 9d.	—	26s. 4d.	—	30s. 1d.	—
Pay limits at stope faces	3·60	3·61	—	3·85	—	3·86	—	4·12	—	4·09	—	4·82	—
Ore reserves which become payable with decreased waste milled	3·61	3·61	—	3·93	—	3·93	—	4·34	—	4·34	—	5·54	—
Gain in ore reserves	·01	·00	—	·08	—	·07	—	·22	—	·25	—	·72	—
Ore reserves which become unpayable with increased waste milled	3·17	3·17	—	3·45	—	3·45	—	3·79	—	3·79	—	4·82	—
Loss in ore reserves	·13	·44	—	·40	—	·41	—	·33	—	·30	—	0·00	—
Life of mine (years)	23·10	20·22	—	20·83	—	19·53	—	19·64	—	18·48	—	15·60	—
Percentage decrease of life ...	—	12·5	—	9·8	—	15·5	—	15·0	—	20·0	—	32·5	—
Equivalent stamps ...	200	225	—	220	—	231	—	230	—	240	—	265	—
Equivalent tube mills	4	4·5	—	4·4	—	4·6	—	4·6	—	4·8	—	5·3	—
Net life profit ...	5,829,608	5,842,771	—	5,981,064	—	5,965,018	—	6,173,971	—	6,219,185	—	6,676,972	—
Increase in profit	—	13,163	—	151,456	—	135,490	—	344,363	—	389,577	—	847,364	—
Yearly dividend ..	50·47	57·79	—	57·43	—	61·09	—	62·87	—	67·30	—	85·60	—
Present worth—6% and 3%	2,839,694	3,047,613	—	3,074,171	—	3,163,973	—	3,265,845	—	3,384,921	—	3,913,632	—
Present worth per share	£5 13 7	£6 1 11	—	£6 3 0	—	£6 6 7	—	£6 10 8	—	£6 15 5	—	£7 16 7	—
Percentage increase in present worth ..		7·3	—	8·3	—	11·4	—	15·0	—	19·2	—	20·2	—
Ore Reserves—													
Payable ore reserves—Waste . .	4,158,675	4,158,675	—	3,541,249	—	3,541,249	—	2,938,797	—	2,938,797	—	1,524,848	—
,, ,, Reef ...	2,772,450	2,772,450	—	2,832,999	—	2,832,999	—	2,938,797	—	2,938,797	—	3,049,695	—
,, ,, Total .. .	6,931,125	6,931,125	—	6,374,248	—	6,374,428	—	5,877,594	—	5,877,594	—	4,574,543	—
Percentage decrease of waste	—	—	—	14·8	—	14·8	—	29·3	—	29·3	—	63·4	—
Percentage increase of reef	—	—	—	2·2	—	2·2	—	6·0	—	6·0	—	10·0	—
Percentage increase of payable reef claims worked	—	—	—	2·2	—	2·2	—	6·0	—	6·0	—	10·0	—
Percentage decrease of nominal ore reserves	—	—	—	8·0	—	8·0	—	15·2	—	15·2	—	34·0	—

ft. is less, this item would decrease from 62,439 to 53,515 tons, as a minimum of waste would be mined in the latter case.

Value of Tons from Stopes.—If the basis mine produced rock of 6 dwt. over the stope face, assuming the reef value at 31·96 dwt. and the waste value ·7 dwt., then the 36 in. mine would increase its stope grade to 9·54 dwt.

Value of Yield.—The yield for the 60 in. stope would be 6·63 dwt., or 27s. 11d., and that of the 36 in. stope 9·37 or 39s. 4d., or an increase of 11s. 5d. per ton milled. I have assumed the extraction to be 94% and 96% respectively.

Stoping Costs.—If a ton of rock costs 4s. 6d. to stope in the 60 in. stope, I have—for the time being—given a cost of 7s. 6d. or a 3s. increase for the 36 in. stope.

Tramming and Shovelling Costs.—I have added 6d. a ton for the decreased stope width.

Development Costs.—I have used the cost of 56s. per ft., and consequently the 36 in. mine would incur an increase of 1s. per ton milled for the extra development.

Reduction Costs.—The reduction costs would vary but little.

Cost per Ton Milled.—The above costs for the 60 in. mine total 16s. 1d. per ton milled, and they increase to 19s. 6d. for the reduced stope width mine.

Profits per Ton Milled.—The profits are, however, increased when mining the same reef at a uniform value, with similar mining and reduction conditions, from 11s. 10d. to 19s. 10d. per ton milled, or an increase of 8s per ton.

Equal Profits.—I shall now endeavour to prove that although in actual practice the above results may not be achieved, yet at the same time it is possible to prove that the principle advocated is true. I state that the profits would increase from 11s. 10d. to 19s. 6d. Assume that this deduction is untenable, and that the costs would rise higher than 19s 6d. Then, if the profit in the 36 in. stope were only 11s. 10d., the costs could rise from 19s. 6d to 27s. 6d., which would enable the stoping costs to rise from 4s. 6d. to 15s. per ton, when the width was cut down from 60 in. to 36 in., which, in my opinion, is reducing this possibility to an absurdity and quite without the pale of reason. It would also be seen how simple it is to reduce the costs from 19s. 6d. to 16s. 1d., without an increased efficiency. I consequently claim that although I do not maintain that results achieved would be realised to a penny, yet the principle is vindicated.

Percentage Increase in Costs and Profits.—Mine B increases the costs 12% and the profits 14%, or a ratio of 1 to 1·2. Mine C drops the costs from 18s. to 17s. 3d., through shovelling, tramming, hoisting, etc., less waste, which is a 7% increase on mine A, and the profits are increased in the ratio of 1 to 1·9. and so on to the 36 in. mine, which increases the profits over the costs by the ratio of 1 to 3·2, which tends to prove my assumption that by increasing the waste milled by the costs and profits are increased in a greater ratio.

Costs and Profits per ton of Reef Milled.—The stope width of 60 in. is composed of 24 in. of reef and 36 in. of waste, but 20% is sorted out on the surface, which accounts for 12 in., consequently every ton going to the mill consists of 24 in. of waste, that is half reef and half waste. If the cost per ton milled of waste and reef is 16s. 1d., the cost per ton of reef is 32s. 2d. In a similar manner it will be seen that the majority of the rock broken, hoisted and milled in the 36 in. stope is reef which works out at 74% reef and consequently although the cost per ton of waste and reef milled increases to 19s. 6d., the cost per ton of reef decreases from 32s. 2d. to 26s. 4d. And similarly the profits increase from 22s. 11d. to 30s. 1d. for every ton of reef milled. This proves my alternative hypothesis that maximum reef milled decreases the cost and increases the profit per ton of reef milled. Our mines have been laid out to mine and mill the maximum of reef, not waste.

Pay Limits at Stope Faces.—The pay limits rise from 3·60 dwt. to 4·82 dwt.

Value of tons at Stope Faces.—The 6 dwt. value over 60 in. increased to 9·54 dwt. in the 36 in. stope, or an increase of 3·54 dwt. Since the pay limit increases only 1·22 dwt., there is a direct gain of 2·32 dwt. in the increased stope value over the pay limit. In other words, although the pay limit increases, the value of the stope increases in a greater ratio.

Ore Reserves which become Payable.—The 60 in. stope would require 3·61 dwt. to be payable. Cut this face down to 36 in. and the value becomes 5·54 dwt., which is ·72 dwt. greater than the pay limit of 4·82 dwt. in the 36 in. mine. And conversely, if 4·82 dwt. were being broken in a 36 in. stope—this is equivalent to 3·17 dwt. in a 60 in. stope—which means that the narrower stope is mining rock which would be unpayable by ·43 dwt. if the width were carried to 60 in. I consequently consider that the principle is proved which enunciates 'some of the unpayable ore reserves become payable when a minimum of waste is milled.'

Life of Mine.—The 60 in. mine would have a life of 23·1 years, whereas the 36 in. mine would be worked out in 15·6 years, or a percentage decrease of 33%.

Stamps and Tube Mills.—If the plant were 200 stamps and 4 tube mills, and it was required that more reef should be milled, the 60 in. mine would have to add 65 stamps and 1·3 tube mills to mill as much reef as the 36 in. mine. How much simpler and less costly it is to reduce the waste milled. Instead of spending money on extra stamps, spend more money on the sorting plant first. I am conducting experiments to see what additional waste can be sorted out of the product going to the mill after coarse and medium waste has been eliminated.

Increase in Life Profit.—Although the reduced width of mine has a decreased life of 7·5 years, yet yet in the shorter time it would yield £847,000 more profit from the same reef.

Present Worth of Shares.—The present worth of 6% and 3½% is increased from £5 13s. 7d. to £7 16s. 7d., or 20·2%.

Nominal and Actual Ore Reserves.—The nominal ore reserves, that is the reserves of reef plus waste, are decreased from 6,931,125 tons to 4,574,543 tons, but the actual reserves, that is the reef tons, are increased from 2,772,450 tons to 3,049,695 tons, which is accounted for by 10% of the unpayable ore reserves becoming payable.

Payable Reef Claims Worked.—10%, or 30 more claims would be worked.

Conclusion.—In conclusion I am firmly convinced that if the development is kept well ahead, the stope widths are reduced to a practical minimum and the maximum waste sorted out on the surface, the resultant profits per unit of gold will be at their maxima, no matter what the costs per ton of waste and reef are, provided the maximum efficiency in working the mine is attained. Low costs without efficiency will not bring the large tonnage of low-grade Rand ores within the limits of payability."—H. MUSSON THOMAS.—*South African Mining Journal*, April 3, 1910, p. 257. (A. R.)

Hammer Drills in Overhand Stoping and Raising.—"Before the Institution of Mining and Metallurgy, Mr. H. B. Williams, Associate, read a paper from which we extract the following:—The numerous advantages obtained by the recent substitution of hammer drills in place of hand labour and ordinary piston drills having now been sufficiently demonstrated by means of exhaustive trials and practical operation in stoping and raising at the Granite Gold Mines, British Columbia, I herewith append an account of my experience of their performance. Overhand or back-stoping only is practised at these mines, the reef consisting of hard quartz from 8 in. to 5 ft. in width, containing 2% to 10% of sulphides of iron and arsenic, while the country rock is a dark close-grained granite. The vein underlies at angles varying from 35° to 50°. Figures being available to show the cost of stoping as practised in previous years at the mines, this paper is written with the object of a general comparison of results by the various methods employed, and in particular with regard to those obtained with hammer drill. The principal wearing parts are the piston hammer, costing $4, the chuck $3½, tappet $2, and complete valves $10. I now propose to deal with the advantages of the hammer drill over piston drills manifested in the tests underground, which led to the final adoption of the former at the mine for stoping and raising. The lightness and general handiness of the 2 in. hammer drill, its weight being only 60 lb against the 125 lb. of the 2 in. piston stoping drill, is a point in its favour. There is a considerable saving of time in setting up. In a shift of eight hours, it can be safely estimated that the loss of boring time in setting up, changing steel, and taking down a piston drill is two hours; with the hammer drills half-an-hour only per shift is sufficient, less time is lost in changing steel, there being no U-bolts, bars, or mountings to manipulate. Inexperienced hand-labour miners can learn to run the hammer drills in a few shifts, a great advantage in a country where good machine men are difficult to procure. Greater drilling efficiency has been obtained. With the small 2 in. one-man piston drill, a shift's work comprising a total footage bored of between 25 and 30 ft. was considered satisfactory, whereas the hammer drills bore 10 4-ft. holes, or an aggregate of 40 ft. per shift. The bits of the drill steel are not dulled by the quick, light blows of the hammer drill to the extent which occurs with the heavier blows from the piston drill. Owing to the fact that the hammer drill is so compact and requires no bar or mountings, holes may be bored to take the greatest advantage of slips in the ground in low stopes, and greater efficiency in breaking ground has resulted. The construction of the hammer drill renders its adaptability most evident for carrying considerably lower stopes than with the piston drill, a great saving of cost in stoping narrow reefs, such as those of the Granite Mines. I have been struck with the small quantity of fine dust made by the hammer drill in the stopes and rises. I account for this improvement over piston drills by the fact that there is no churning movement of the drill steel in the hole, as is the case with the piston drills, the bit of the drill steel being held continually in the bottom of the hole while boring. The air is exhausted back from the face of the stope and no air can escape at the chuck end of the machine. It may also be noticed that the hammer drills possess the advantage of producing coarser cuttings than the piston drill. In air consumption I have found that with a receiver pressure of 80 lb., there is only a slight advantage in practice in favour of the 2 in. hammer drill over the 2 in. piston drill. The advantage in general handiness, increase in progress and consequent reduction of cost in raising with the hammer drill are even more marked than in the case of stoping by its use."—*S. A. Mining Journal*, March 7, 1910, p. 299. (A. R.)

Colliery Gas Indicators.—"1. In the incandescent wire system we find that two small loops of platinum wire of equal magnitude can be made to glow by the passage of an electric current through them simultaneously. This current is generated by turning the handle of a small dynamo used in the manipulation of the system. One of the loops is surrounded by an air-tight brass cylinder, with a glass disc at one end; the other by a cylindrical copper gauze, also with a glass disc at one end. These two glass discs face each other at a few inches apart, and when the wires are made to glow the light from each falls upon the two sloping surfaces of a small movable A-shaped block covered with white paper which stands between them. When there is combustible gas in the air which circulates through the wire-gauze cylinder, the loop within that cylinder glows with greater intensity, or, in other words, emits more light than the opposite loop in consequence of the combustion of the inflammable gas in contact with it. As a result, that side of the movable block nearest to it is brighter than the other side. If the block is now moved towards the other loop until its two sides are equally bright, then its relative distance from the loops is a measure of the intensity of the light emitted by each, and by an easy process of comparison, arrived at by experiment, can also be made a measure of the proportion of combustible gas in the air. In testing a scale is provided, fixed in a line along which the apex of the block is moved, which is divided and marked to show percentages of the methane. If there is any difficulty in judging the relative brightness of the two sides of the block, the system is defective.

2.—In the alcoholic flame system we have a combination of a complete safety lamp and a small alcohol lamp. The latter may be screwed into an eccentrically situated recess in the bottom of the former, and its wick holder projects up through a tube which it exactly fills and terminates close to the wick holder of the safety lamp. The two wicks are thus brought so close together that if either of them is ignited the other immediately takes fire also. Either of these can be extinguished by drawing down or re-lighted by pushing up. It is thus possible to test for gas with alcohol flame alone, and then to re-ignite the oil flame and extinguish the alcohol flame without opening the lamp. The safety lamp itself is capable of detecting gas of 3% and over, but by the introduction of the alcohol tester fire-damp from 0·5% to 3% may be detected. About one-third of the inside of the glass is enamelled black so that a background may be formed to facilitate observation of the gas caps. In testing by means of the oil flame the wick is drawn down until it just loses its bright tip; the black glass then indicates the gas cap. If no distinct cap is seen, this does not prove the absence of gas, as it does not indicate its presence when less than 3% is present. The brass plug is then withdrawn from the oil lamp, and the alcohol tester screwed into the recess. This, by an ingenious device, opens its own wick tube, and the heat from the oil flame causes the alcohol to ascend and ignite by contact with the oil flame. The oil flame is then extinguished, and the test by alcohol accomplished. The gas caps will be as follows, taking half-inch alcohol flame as the standard :—

Per Cent. of Gas.	Inch Gas Cap.	Remarks.
0·5	·60	Very pale, not clearly seen.
1·0	1·00	Pale blue, clearly seen.
1·5	1·44	Pale blue, clearly defined.
2·0	1·68	Clear blue cone.
2·5	2·00	Distinctly defined gas cap.
3·0	—	Oil flame used.

The alcohol flame is a simple and effective gas indicator, and can be applied very easily as a tester.

3. In the hydrogen flame indicator we have a good practical invention. In construction it is an ordinary oil or benzoline lamp, with a copper tube passing vertically through the oil vessel, and terminating close to the wick holder, and connected to a steel cylinder fixed to the side of the lamp at the other. The steel cylinder contains compressed hydrogen, which is retained in it by means of a screw valve. When it is desired to test for smaller proportions of fire-damp than can be detected by the oil or benzoline flame, hydrogen is allowed to flow through the copper tube by slightly opening the valve. The hydrogen jet thus produced immediately ignites at the flame of the lamp. The wick of the latter is then drawn down until its flame is extinguished, and the hydrogen flame alone remains. The flame is then regulated to the proper height by means of a valve, and the heights of the caps produced by it are read off on a scale at the back of the glass. The percentage is judged as much from the brightness as from the height of the cap. The gas caps produced by a hydrogen flame (reduced) are as follows :—

Per Cent. of Gas.	Inch per Cap.	Remarks.
¼	·7	Pale and hazy.
½	·7	More definite, and outline easily perceived.
1·0	·9	More clearly seen.
2·0	1·2	—
3·0	2·1	Becoming larger and more distinct.

The incandescent system is quoted as one which is very liable to become defective or even useless during a test. In the alcohol flame system it is said when the apparatus becomes even slightly heated, by the heat of the flame or the temperature of the mine, a vapour is given off by the alcohol which is an explosive agent when mixed with air. Further it is liable to become easily extinguished if not handled with extreme care.

For the hydrogen flame it is claimed to make a test with absolute certainty. The flame is non-luminous and capable of emitting considerable heat with safety ; this serves to enhance its reputation as a most favourable fire-damp indicator. It is not readily extinguished and can be taken in safety into currents which would extinguish an ordinary safety-lamp. The hydrogen flame left burning would re-ignite the oil lamp without danger in suitable current. It is capable of long duration of flame in an atmosphere which is irrespirable. Will continue to burn in atmospheres containing from 50 to 60% of carbonic acid gas, when about one-quarter of the amount would extinguish an oil or alcohol flame. It is considered an efficient and effective lamp for gas testing, but condemned by some for the foregoing reason. It is very sensitive when detecting low percentages of CO_2, exhibiting a change of colour, which is more evident as the proportion of CO_2 increases. It is capable of being taken into any mine as a fire-damp detector, and can be utilised under many other conditions. No waste of time in making a test—not necessary to exceed 30 seconds. It is not inferred that it would be prudent to place the apparatus in an atmosphere where the oil flame becomes extinguished ; I only quote its capabilities."—T. H. KNEEBONE, *Science and Art of Mining*, March 5, 1910, p. 356. (A. R.)

THE TREATMENT OF NATIVES ON MINES.—"We take the following from the report on the health conditions on Rhodesian mines :—

The members of the committee were Dr. A. M. Fleming, C.M.G., Chief Medical Officer, Rhodesia, Dr. Donald Macaulay, M.L.A. (Transvaal) and Dr. W. Gladstone Clark, M.A. (Bulawayo).

Prevalence of Scurvy.—The figures given in the last reports available, namely, for the year 1908, show that there were 1,671 cases of scurvy on the mines in Southern Rhodesia, with 207 deaths, and this may be considered a low estimate, as many cases are unreported, or died of secondary complaints such as dysentery and pneumonia, and in proof of this, during our commission we found on several mines a large number of cases of undoubted scurvy which did not figure on the hospital sheets. We have no reason, moreover, to suppose that there has been any appreciable diminution in its incidence since this report was published.

Infectiousness.—Evidence taken both in Rhodesia and in the Transvaal is unanimous as to the fact that a large proportion of the alien recruits from the tropical zones arrive either with the disease upon them or develop it within the first three months of service. Outbreaks occur from time to time amongst native miners who have been resident for longer periods than this, but careful enquiry will generally bring to light some error in the dietary existing on the particular mine concerned.

Deferred Pay.—It would be convenient here to refer to the system of deferred pay, as this would appear to have a certain bearing upon the incidence of scurvy. It is a striking fact that habitual labourers and those natives who add to their diet by purchases of foodstuffs from local stores rarely suffer from scurvy, whilst those natives whose pay is reduced by remittances to their homes or who are of a parsimonious nature, are particularly affected.

Diet.—We would especially emphasise the necessity for the issue of fresh vegetables, further reference to the growing of which will be found in clause 65. As far as possible, vegetables for which the natives have a predilection should be given, and fortunately these are the most easily grown. We would specially mention potatoes, pumpkins—in fact, all gourds—sweet potatoes, wild spinach ; also leeks and onions, where obtainable, are valuable anti-scorbutics. The difficulty of obtaining fresh vegetables for the mines has been impressed upon us, and we therefore recommend that at seasons of the year, and in districts where fresh vegetables are unobtainable, ½ lb. of meat and a double ration of Kafir beer be added to the diet scale as a substitute.

Kafir Beer.—The nutritive value of Kafir beer and its action as an anti-scorbutic have been amply demonstrated ; we consider that it should be a compulsory ration on all mines. The preparation should be in the hands of the mining authorities, and it should be issued with regularity and under supervision. In this connection we found on visiting the Rand that an excellent beverage called "mahauwe" was prepared in some of the compounds from the residue of the mealie porridge. This is a mildly fermented beverage, is easily prepared, and has a distinct nutritive value. A description of how this should be prepared will be found in Annexure "C."

Native Diet in Kraals.—It was alleged by some witnesses that the diet of mine natives compared more than favourably with the diet the native was accustomed to in his kraal. We have enquired into the native kraal diet, and though a greater quantity of food may be consumed by the native on the mine, still, in his kraal his food is richer in those fresh constituents to the presence of which we attach so much importance. Apart from this consideration, it seems superfluous to insist that the scale of feeding obtaining in the native kraal is no criterion as to a native's wants whilst engaged in mining.

Prevalence of Pneumonia.—During the year 1908 there were reported to the Government Health Department no fewer than 2,251 cases of pneumonia, with 686 deaths, giving an incidence of 72·93 per 1,000, and a death rate of 22·23 per 1,000 labourers. Our investigations have shown that these figures are an under-estimate.

Overcrowding as a Factor.—Overcrowding in compounds produces ideal conditions for the spread of an admittedly infectious disease. This overcrowding is brought about in two ways; firstly, by the provision of inadequate accommodation and insufficient air space, as we found on some mines; secondly, by the habits of the natives, who prefer to crowd into huts with their brothers rather than occupy the places set apart for them. There is more pneumonia during the winter months, when labour is most plentiful and overcrowding most prevalent.

Structural Defects of Dwellings.—The effects of overcrowding are aggravated by the structural nature of the huts and the personal habits of the native. We consider that the ordinary hut full of the heterogeneous collection of odds and ends with which the native surrounds himself, badly ventilated and badly lighted, with an absorbent floor which he uses as a spittoon, and which he impregnates with nutrient material from the refuse of his food, forms an ideal incubator for the germs of an infectious disease.

"*Stricter and More Intelligent Compound Inspection.*"—We recommend that, where possible, there should be a daily inspection of all natives in the compound, and an inspection by a medical officer or other competent person once a week. The sanitary condition of many of the mines we visited is open to improvement. Clothing parades and disinfection of clothing, and the prohibition of natives changing their clothes with natives, should be enforced.

General Conclusions.—Food should be abundant and properly cooked. The fine grinding of meal is of the utmost importance. The cultivation of mine gardens for the supply of vegetables should be encouraged. Natives who have their wives on the mines seldom suffer from sickness, and tend to become habitual labourers. The office of compound inspectors should be removed from the Mines Department and placed under the Health Department. Such officers should be given powers of summary jurisdiction in cases of minor offences under the regulations."—*S. A. Mining Journal*, May 7, 1910, p. 286. (A. R.)

HYDRAULIC IMPREGNATION OF THE COAL FACE AND HYDRAULIC BLASTING IN COAL SEAMS BY THE MEISNER PROCESS.—In 1890 Meissner devised a process for rendering coal dust harmless, by forcing water under a pressure of 10 to 20 atmospheres into a borehole about 40 in. in depth for a considerable time. The mouth of the borehole was fitted with a stuffing gland. The water impregnated the coal face in such a manner as to enable the coal to be brought down free from dust. The process, however, suffered from certain deficiencies and after a time was given up.

On account of the Rodbad explosion last year experiments were again started at a colliery near Dortmund, and later were extended to a number of other collieries in Westphalia. Deeper boreholes were used than in 1890. The packing gland was also improved and extended further into the borehole. In dusty bituminous seams the experiments have been quite successful, but in compact non-porous structures the method has been found impracticable.

When the coal is porous and mild, a pressure of 20 atmospheres are found sufficient, but in more compact seams 40 atmospheres are often necessary. Again, in some seams the impregnation and loosening of the coal are complete in about 10 minutes whereas in others 5 to 6 hours are necessary.

As impregnation is proceeding the coal can be heard to tear, break and even detonate. The process is attended with great difficulty in seams where the adjacent rock is cleaty and friable, since under these circumstances the rock can be easily wetted and even begins to creep, necessitating great care. The main results of the process are greater safety in shot firing, reduction of consumption of explosives and absence of dust during loading and haulage.—BERGASSESSOR TRIPE, Dorstfeld.—*The Colliery Guardian*, July 8, 1910, p. 67. (T. D.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

ORE MINING METHODS. By WALTER R. CRANE. Cloth, octavo, pp. 225. Ill. (New York: John Wiley & Sons. London: Chapman & Hall.) Price 12s. 6d.

"This book contains descriptions of methods of timbering and supporting rock in place, methods of stoping and mining in narrow and wide veins and bedded and massive deposits, including stull and square-set mining, filling and caving, open-cut work, and a discussion of costs."—*Mining Magazine*, July, 1910. (A. R.)

GEOLOGY: SHORTER COURSE. By Profs. CHAMBERLIN and SALISBURY. 21s. (London: John Murray.)

"The scientific and literary partnership of Profs. Chamberlin and Salisbury, the heads of the Department of Geology in the Chicago University, is a felicitous one, and has been productive of excellent results upon geological teaching in America. Their advanced course not only achieved American celebrity but speedily attained to considerable vogue in this country; indeed, we can hardly remember another foreign work of the same type which sprang so instantaneously into prominence and received so frank and cordial a welcome. The shorter course has been prepared in response to an obvious demand, and is also excellent, the thankless work of abridgement having been carried out with judgment and with skill. We have, in fact, nothing but praise for the book, which is adequate in its presentment of principles, informed with a multitude of illustrations drawn from the labours of the best equipped geological survey in the world, and written in a manner which administers a useful stimulus to the imagination, without in the least trangressing the restraint proper to scientific writing. The stratigraphical record as here presented is, naturally, almost exclusively American in its inspiration, and to English students, accustomed to the English classification, the descriptive names will jar with the unfamiliarity of a foreign language. But such differences are inevitable to the science, and form its chief, perhaps its only, drawback. Moreover, they tend to destroy provincialism of thought, and are thus of independent utility. A point which constantly recurs in this work, which is of great importance, is the presence or absence of deep-sea deposits in the sedimentary record. Our authors hold—and M. Emil Haug in his recently published traité partially concurs with his view—that, so far as at present determined, the derivative rocks exhibit no examples of formations parallel with those now accumulating upon the ocean floor. Even the true white chalk, which occurs in the Secondary rocks of the United States, though not in such great vertical extension as with us, the authors attribute to conditions of comparatively shallow but clear water, and not to the true ocean depth. This view, which is gaining constantly greater currency among geologists of the modern school — that the gigantic changes of sea-level formerly postulated never occurred — is obviously fundamental, and carries with it the conclusion that the relative positions of sea and land have varied in geological time only within comparatively narrow limits."—*London Mining Journal*, June 11, 1910. (A. R.)

MEASUREMENT OF PULP AND TAILING. By W. J. SHARWOOD. Pamphlet, pp. 24. Ill. (London: *The Mining Magazine.*) Price 2s.

"This is a reprint of Mr. Sharwood's Articles that appeared in *The Mining Magazine* for November, December and January. The information contained in the articles has proved to be of great value to metallurgists and this issue in separate form has been made to supply a demand."—*Mining Magazine*, July, 1910. (A. R.)

NATAL: REPORT ON THE MINES AND MINERAL RESOURCES (OTHER THAN COAL). By F. H. HATCH, Ph.D., M.I.C.E. Published by order of the Natal Government. (London: Natal Government Agency.)

"As the result of a personal examination the public would naturally hope for some definite expression of opinion from a trained and experienced expert as to the probable result, favourable or unfavourable, of further development on those deposits which have been partly opened up. It is remarkable in a report of this kind that as regards the majority of the properties described there is not the faintest indication of opinion as to what might be the outcome of further development. Only in instances where developments have proved conclusively the worthlessness of a property is there any definite opinion expressed.

Plans and sections of development work in progress were in the majority of instances non-existent; and this deficiency, due to the existing regulations of the mining department, is pointed out in the report, but the writer of it should have supplied the deficiency so that his readers might obtain a clear insight of the structure of the ground being explored. Such insight can very rarely be obtained from verbal descriptions. They leave too much to the imagination of the reader. Facts conveyed by correct plans and sections are definite in themselves and in their relation to one another.

From the descriptions of the iron ore beds and of several of the non-metallic minerals it is evident there are indications of deposits on which further investigation might lead to the establishment of profitable industries, but, as is properly pointed out in the report, their exploitation is hampered by the lack of any geological maps. Without reliable surveys no systematic records, either geological or statistical, can be maintained. A proper survey of the Colony would be of the greatest possible benefit to individual enterprise.

To those intimately associated with Natal the inconclusive nature of this report must be disappointing; but it should in no wise act as a deterrent on their efforts to develop the resources of the Colony, and it is to be hoped that the Government will, by a proper organisation of the department concerned, render more efficient assistance to prospectors and miners than they have done in the past."—*London Mining Journal*, July 9, 1910. (A. R.)

REDUCTION OF WORKING COSTS ON THE RAND. By HENRY HOWARD. 1s. (London: *Investors' Guardian*, Limited.)

"This is dated May, 1910, and is an examination of existing conditions and the prospects. The author states that the work, which has appeared in the form of contributions to the *Investors' Guardian*, is the result of information gained during his last visit to the Rand in the autumn of 1909. The brochure sums up very well what has been done under the big mills low grade low costs policy of the last two or three years. From 1902 to the autumn of 1909 the average cost of all the Witwatersrand mines was reduced from 25s. 9d. to 17s. 2d. per ton. Authorities are cited to support the expectation of a further reduction, but it may be well to remember that during the last few months the leaders of the industry have been giving intimations of changes in policy, which would involve higher rather than lower costs."—*London Mining Journal*, June 11, 1910. (A. R.)

FROM PROSPECT TO MINE. By ETIENNE A. RITTER. 8s. 4d. (Denver, U.S.A.: The Mining Science Publishing Company.)

"Chapter I. treats on Prospectors and outfits, and these subjects are particularly well dealt with. Chapter II. is on Veins and Ore-bodies; Chapter III. on 'Taking up Claims and the Apex Law." This chapter, based on the laws of the United States, graphically describes all the procedure necessary to be executed and work accomplished before a claim can be said to be properly registered and owned by the prospector or taken up, and this chapter also defines the text of the patent grants of a lode mining claim.

Chapter IV. treats on "Starting work underground," and contains many practical suggestions as to the best course of procedure under different conditions. The author rightly says that a serious and a frequent mistake is not to have the work directed in the first instance by a good mining engineer, who need not stay at the mine all the time but inspects it regularly and keeps posted on the process of the work; and we also endorse the advisability of having assays taken regularly when a vein is being developed. There is also a great deal of truth in the author's view put out—that the future of the mining industry belongs to some powerful prospecting and mining companies able to stand the losses incurred in failing to open a mine from a large amount of prospects, and yet able to keep on until they get a bonanza which will not only make up for all the money previously lost, but bring besides a tremendous profit; in such a company, as in an insurance company, the working of the law of averages will bring success.

Chapter V., on "Mining Companies," furnishes a few useful practical notes of advice, and also words of caution for both the prospector and also the investor as to the flotation of metalliferous mining companies, and the methods adopted for the getting up and the issuing of prospectuses with a view to the raising of share capital.

Chapter VI. deals with Pointers as to Local Conditions, and Chapter VIII. with Mining Machinery.

Chapter X.—the last one in the book, on 'Philosophy and Poetry of Mining'—should be read with the warning note with which this review opens."—*London Mining Journal*, July 9, 1910. (A. R.)

HINTS ON SURVEYING; AND THE KEEPING OF MINE PLANS AND REPORTS. By J. W. TEALE, M.I.M.M. 6s. (London: *The Mining Magazine*.)

"A thin octavo volume containing 18 pp. of subject matter and 21 of specimen forms for monthly returns, etc. The reading matter consists of a few desultory remarks on mine plans in general; the hints on surveying, which occupy the major portion, comprise a list of conventional signs usually employed by mining surveyors. As regards the forms proposed for the returns, they may probably serve their purpose; but these are matters of individual taste, and most managers have some hobby of their own. The book is very neatly bound."—*London Mining Journal*, July 9, 1910. (A. R.)

Abstracts of Patent Applications.

(C.) 245/10. James Dawson & Son, Ltd. (1), James Dawson, junior (2). Improvements in conveyor belts.

The invention for which the above application is made relates to improvements in the construction of conveyor belts.

According to this invention the completed belt is constructed of a belt, the full width of the completed belt, to either or both sides of which are attached, longitudinally, two or any greater number of belts of smaller widths. The smaller widths may be attached to the full width by means of rivets, screws, or sewings, or by means of balata or like substances.

All the widths are constructed of textile fabric, and each may be of any number of plies suitable for the purpose for which the belt is required. The textile fabric is usually impregnated or spread with balata, gutta percha, india-rubber or like substance.

The intention of the invention is to secure greater longitudinal pliability than is possible in a belt of the same strength made in one width, and to increase the life of the belt as if one of the smaller widths wears out it can be easily replaced.

(C.) 325/10. Ottokor Serpek. Improvements in electrical resistance furnaces. 8.7.10.

This invention relates to the manufacture of nitrides, especially aluminium nitride.

A furnace is described which produces the nitrides from mixtures of suitable materials through the agency of the heat produced by an electric current in resistances constructed of carbon and aluminium nitride. These resistances are so shaped as to stir up the melted ingredients when the furnace is revolved, thereby securing an easy and regular distribution of the heat generated.

(C.) 354/10. Charles Anderson Case. Improvements in a panning and concentrating machine. 22.7.10.

This application refers to an improved panning and concentrating machine consisting of a screen and a series of pans, mounted co-axially with openings near the lowest points of the conical pans and a means of applying a circularly swaying motion. Balls are used to aid the separation of the concentrate and to force it through the openings provided.

(C.) 367/10 and 368/10. William Woodward Robacher. Improvements in or relating to filtering apparatus particularly intended for the cyanide process of extracting precious metals from their ores. 29.7.10.

These applications refer to vacuum filters of the vertical rotary type, the lower half of which is immersed in slime pulp, and provided with scrapers acting on the upper part for removal of the partially dried solid. 367 refers to a filter with the filtering medium forming the walls of the rotating member, and 368 to one with the filtering medium forming the periphery. The principle feature of the design is the use in both of a rigid, porous mineral material as the filtering medium.

(C.) 576/09. David Gilmour. Improvements in fixing cam pulleys to cam shafts. 13.12.09.

This refers to a means of fastening pulleys on camshafts, and consists of a tapered bore in the pulley boss to fit on a taper turned on the shaft. The grip of the pulley is obtained by screwing up a nut on a thread cut on the cam-shaft whilst a key may also be fitted if desired.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 410/10. William Arthur Beard. Improvements in and relating to rotary engines and the like. 19.8.10.

(C.) 411/10. William Edge. Improvements in hinging machines. 19.8.10.

(P.) 412/10. Willie Herbert Stewart. Drill heating and smelting furnaces. 19.8.10.

(C.) 413/10. Elizabeth Barnston Parnell. Improvements in the treatment of ores. 19.8.10.

(C.) 414/10. John Gillies (1), Claude Daniel McPhee (2). Improvements in scratch headed fuses. 20.8.10.

(P.) 416/10. Philip Lawrence Lazarus. An improved sanitary pit. 22.8.10.

(P.) 417/10. John Broad Roberts. Anti phthisical lubricator and appliance. 23.8.10.

(P.) 418/10. Adam Richard Stacpoole (1), Joseph Mahoney (2). Improvements in and appertaining to the shoes and dies of stamp mills. 24.8.10.

(P.) 419/10. Norman William Pill. Improvements in apparatus for the manufacture of inflammable gas by carburetting air. 24.8.10.

(C.) 421/10. William Edward Harding (1), John Hunter (2). An improved chuck for rock drills and other purposes. 25.8.10.

(P.) 423/10. William Charles Stephens. Improvements in or connected with cradles for rock drills and other machines. 25.8.10.

(P.) 424/10. Hans Charles Behr. Improvements in pneumatic percussive apparatus. 26.8.10.

(C.) 425/10. William David Coolidge. Improvements in tungsten and the manufacture thereof. 26.8.10.

(C.) 426/10. Emil Deister. Improvements in ore concentrators. 26.8.10.

(P.) 428/10. Cecil Hardwood Fenn (1), David Gavine Hunter (2), John Hathorne Wilson (3). Improvements in rope jockeys. 27.8.10.

(P.) 429/10. Reitz Gannon. Improved process to bring about the complete combination and convert cyanide mine dump sands into hard substances such as concrete waterproof plaster and the like. 29.8.10.

(C.) 430/10 Holberry Mensforth. Improvements in internal combustion engines. 29.8.10.

(P.) 431/10. James Auld. Improvements in driving means for the ore feeders of stamp mills. 2.9.10.

(C.) 432/10. Robert Emmott. Improvements in crushing, pulverising and disintegrating machines. 2.9.10.

(P.) 433/10. William Henry Rufus Munnery. An improved double action stone crusher. 3.9.10.

(P.) 434/10. Arthur Martin Ludwig Dammrich (1), Charles Hansen (2). Improvements in means for conveying and distributing water or other liquids in mines and the like. 3.9.10.

(P.) 435/10. Arthur Martin Ludwig Dammrich (1), Charles Hansen (2). Improvements in means for conveying and distributing water or other liquids in mines and the like. 3.9.10.

(P.) 436/10. Albert Edward Jordan. Improvements relating to hauling and like engines. 3.9.10.

(P.) 437/10. Frank George Seineke. Improvements in means for straightening or altering the direction or course of a bore hole. 3.9.10.

(C.) 439/10. Boris Hellmann. Improvements relating to the treatment of petroleum for the purpose of modifying certain characteristics thereof. 5.9.10.

(P.) 441/10. Anthony Richardson. Vacuum brake improvements. 7.9.10.

(C.) 442/10. Robert James Worth (1), Worth, McKenzie & Co. (2). Improvements in connection with the valve controlling gear of winding engines for collieries or the like purposes. 7.9.10.

(C.) 444/10. William Spiers Simpson (1), Howard Oviatt (2). Improvements in the direct production of iron and steel from the ore. 9.9.10.

(C.) 445/10. Charles Cheers Wakefield. Improvements in or relating to automatic regulators for acetylene gas generators. 9.9.10.

(C.) 446/10. Robert McNitt. Improvements in methods for reducing metals. 9.9.10.

(C.) 447/10. John Collins Clancy. Improvements in the treatment of precious metalliferous ores. 9.9.10.

(C.) 448/10. Michael Joseph Owens. Improvements relating to the production of articles of glass and to apparatus therefor. 9.9.10.

(C.) 449/10. William Emil Bock. Improvements relating to the production of articles of glass and to apparatus therefor. 9.9.10.

(C.) 450/10. Arthur Thomson. An improved means for collecting the dust from holes drilled by rock drills and like machines. 9.9.10.

(C.) 451/10. Thomas Rouse (1), Besseler, Waechter & Co., Ltd. (2). Improved preparation of soluble alkaline silicates containing ferric oxide. 10.9.10.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

Anderson, J. S., *l/o* England; Abosso G. M. Co., Ltd., Abosso, Gold Coast Colony, W. Africa.

Anderson, P. von N., *l/o* Benoni; P. O. Box 678, Johannesburg.

Atkinson, C. E., *l/o* Selukwe; East Gwanda Mines Ltd., West Nicholson, Rhodesia.

Beatty, J. W. S., *to* P. O. Box 1064, Johannesburg.

Brown, J., *l/o* Fordsburg; Village Deep, Ltd., P. O. Box 1064, Johannesburg.

Chomley, W. B., *l/o* Day Dawn; Lancefield G. M. Co., Beira, W. Australia.

Clarence, J. V. G., *to* P. O. Box 27, Jeppes.

Cockburn, P. A., *l/o* Queque, Rhodesia; Le Hang Chang Mine, via Francistown, B.P.

Graham, H. W., *l/o* Johannesburg; c/o Bank of Africa, 113, Cannon Street, London, E.C.

Gray, J. Wensley, *l/o* New Zealand; Wallis Co., Ltd., Ntranang Obuase, via Sekondi, Gold Coast Colony.

Jenkins, C. W. Bowen, *l/o* North Sydney; Yambulla Gold Mine, Yambulla, near Eden, New South Wales.

Johnston, James, *l/o* San Francisco; Room 1002, 30, Church Street, New York.

Jory, J. V., *to* P. O. Box 124, East Rand.

Jupp, H. L., *to* 31a South Street, Vogelfontein, P. O. Boksburg.

Hay, E., *l/o* Johannesburg; Ginsberg G. M. Co., Ltd., P. O. Box 16, Knights.

Laret, H., *l/o* Bulawayo; Monomotapa Concessions, Matokos, via Salisbury.

Meyer, C. E., *to* Three Sisters G. M. Co., Louw's Creek.

Nettle, F. R., *l/o* Benoni; Robinson Deep G. M. Co., P. O. Box 1488, Johannesburg.

Olds, H. F., *l/o* England; Lone Star Mine, Gadzema, Rhodesia.

Oliver, E. L., *to* 9, First Street, San Francisco, California, U.S.A.

Payne, A. E., *l/o* New Primrose G. M. Co.; Consolidated Langlaagte Mines, Ltd., P. O. Box 15, Langlaagte.

Powter, H. B., *to* P. O. Box 5, Knights.

Robertson, J. R. H., *l/o* Penhalonga; East Rand Proprietary Mines, P. O. Box 220, East Rand.

Waterhouse, T. L., *l/o* Peel; Caronleith, Archer Street, Chatswood, New South Wales.

Wertheim, C. J. M., *to* Kaswarie Straat, 3, The Hague, Holland.

Whitehouse, J., *l/o* Utrecht; Village Deep, Ltd., P. O. Box 1064, Johannesburg.

Wright, A. L., *l/o* Florida; P. O. Box 4804, Johannesburg.

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Proceedings
AT
Ordinary General Meeting, October 15, 1910.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the South African School of Mines, on Saturday evening, October 15th, Dr. James Moir (President) in the chair. There were also present :—

70 Members: Messrs. C. B. Saner, R. Allen, E. H. Croghan, Tom Johnson, J. E. Thomas, H. A. White, Prof. J. A. Wilkinson, A. F. Crosse, W. A. Caldecott, Wm. Cullen, E. H. Johnson, W. Abel, H. R. Adam, F. Allen, A. J. R. Atkin, S. G. Bartlett, S. Beaton, E. Blume, B. V. Blundun, F. W. Cindel, M. H Coombe, W. M. Coulter, T. Donaldson, G. C. Fox, R. Gascoyne, J. Gray, F. G. Guthrie, R. E. Hall, B. J Hastings, A. R. Hughes, W. H. Jane, J. H. Johnson, T. S. Jolly, G. A Lawson, Hy. Lea, J. Lea, H. Stuart Martin, J. D. Marquard, S. Morison, P. T. Morrisby, M. T. Murray, W. Nicklin, E. Pam, W. S. V. Price, J. F. Pyles, S. C. Quin, E. T. Rand, A. Rogers, A. H. Scarf, A. Schwarz, A. M Sim, G. H. Smith, S. H. Steels, W. A. C. Tayler, J. A. Taylor, A. Thomas, C. F. Thomas, C. Toombs, F. W. Watson, C. F. Webb, P. H Whyte, A. Wilkinson and L. J. Wilmoth.

29 Associates and Students: Messrs. H. Abao, A. R. Adams, W. E. Burrows, F. C. Carbis, J. Cronin, C. L. Dewar, J. Gibson, C. H. Greathead, B. W. Holman, W. M. Human, A. King, G. W Leach, L. T. Leyson, H. Lomberg, G. F. Mathews, F. J. Pooler, H. B. Powter, J. R. H. Robertson, H. Rusden, R. E. Sawyer, P. Scatterty, H. Stadler, W. R. Tait, A. D. Thomas, D. J. Thomas, J. Thorlund, P. A. Tucker, H. Ward, J. C. Webster, and W. F. Zipp.

18 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as printed in the September *Journal*, were confirmed.

NEW MEMBERS.

Messrs. E H. Croghan and E. E. Blume were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows :—

Duffield, Henry John, Shamva Mines, Ltd., per Private Bag, Salisbury, Rhodesia. Secretary.

Ferguson, Colin Armstrong, P. O. Box 21, Randfontein. Mine Manager

Flander, Robert, Geldenhuis Deep, Ltd., P. O. Box 5, Cleveland. Cyanide Manager. (*Transfer from Associate Roll.*)

Hartley, Frederick Darley, Crown Mines, Ltd. (Eastern Section), P. O. Box 102, Fordsburg. Chief Surveyor.

Hewitt, Geoffrey George, P. O. Rooiberg. Analyst and Assayer.

Knauer, Hermann, Lancaster G. M. Co., Ltd., P. O Box 360, Krugersdorp. Mining Engineer.

Lea, Walter, Messrs. Lewis & Marks, P. O. Box 1030, Johannesburg. Mining Engineer

Milliken, John Tait, Golden Cycle Mining Co., Colorado Springs, Colo., U.S.A. Metallurgist.

Pitt, Charles Kiddle, Lancaster West G M. Co., Ltd , P. O Box 360 Krugersdorp. Surveyor.

Richards, Stephen Herbert, Jumpers G. M Co., Ltd , Cleveland. Mining Engineer.

Richardson, Robert Richard, Jacobs, Durban, Natal. Works Manager and Chemist.

Rishworth, Percy James, Crown Mines, Ltd., P. O. Box 24, Fordsburg. Mill Shift Leader.

Roberts, E., Randfontein Deep, Ltd., P. O Box 45, Randfontein. Mine Captain.

Schoch, Edward Rengers, New Boksburg G. M., Ltd , P. O. Box 209, Boksburg Mine Manager.

Tillard, Max Ogilvie, Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Acting General Manager. (*Transfer from Associate Roll.*)

West, Charles Alfred William, P. O. Box 73, Barberton. Business Manager.

The Secretary: Since the last meeting of the Society the following have been admitted by the Council:—

A, Associates.—

Bass, Kenneth Christian, Three Sisters G. M. Co., Louw's Creek. Assayer. *(Transfer from Student Roll.)*

Bidlake, Albert Ebenezer, Shaftesbury House, Kerk Street, Johannesburg. Metallurgist.

Edwards, Ernest, P. O Box 186, Bulawayo Mining Engineer.

FRY, ALFRED T., Gwalia Consolidated, Ltd., Wiluna, Western Australia. Metallurgist.
JENKINS, ALFRED, New Primrose G. M. Co., Ltd., P. O. Box 193, Germiston. Mining.
KING, DANIEL CHILCOTE, New Primrose G. M. Co., Ltd., P. O. Box 193, Germiston. Mill Foreman.
LEYSON, LEWIS THOMAS, c/o Standard Bank of S.A., Ltd., Commissioner Street, Johannesburg. Analytical Chemist.
McLUCKIE, BARRY WILLIAM, Geldenhuis Deep, Ltd. (North), P. O. Box 5, Cleveland. Cyanider.
OLIVIER, SALOMON HERCULES, Durban Roodepoort Deep, Ltd., P. O. Box 110, Roodepoort. Cyanider. *(Transfer from Student Roll.)*
STIRTON, ARCHIBALD LOCHEE, Consolidated Langlaagte Mines, Ltd., P. O. Box 15, Langlaagte. Mine Foreman.
WOOLF, MONTEFIORE PHILIP, Gwalia Consolidated, Ltd., Wiluna, Western Australia. Metallurgist.

As Students.—

ABAO, HERBERT, Geldenhuis Deep, Ltd. (West), P. O. Box 1, Cleveland. Cyanide Learner.
DALY, RAMSAY L'AMY, Crown Mines, Ltd. (D Section), P. O. Box 156, Fordsburg. Mill Learner.
EDWARDES, ROY DES ANGES, Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights. Cyanide Learner.
SIMPSON, ERIC GERALD, Crown Mines, Ltd. (D Section), P. O. Box 156, Fordsburg. Cyanide Learner.
SMOOK, FREDERICK JOHANNES, Durban Roodepoort Deep, Ltd., P. O. Box 110, Roodepoort. Cyanide Learner.

GENERAL BUSINESS.

The President: I have to express our thanks to the managements of the City Deep, Ltd., and of the Simmer and Jack Proprietary Mines, for their kindness in inviting us to go over their properties, and showing us such interesting works.

Mr. Andrew F. Crosse (*Past-President*): With regard to our very interesting visits to the City Deep and the Simmer and Jack mines, I should like to suggest for such visits in future that it would add very much to the value of the visit if some of the staff would explain the details of the treatment, and also perhaps answer questions regarding the same.

The President: I think that is a very good idea, and we will bear it in mind.

CYANIDE AS AN INSECTICIDE.

Mr. Jas. E. Thomas (*Member of Council*): I should like to mention, with regard to an abstract on p. 86 of the August *Journal*, by Prof. Wilkinson, on the use of "Potassium Cyanide as an Insecticide," that some seven years ago I attempted to rid my garden of ants by sprinkling a 0·001% solution of potassium cyanide on the soil. The ants certainly disappeared, but every plant in the garden was killed Sprinkling a solution of potassium cyanide has also proved an effective method of killing weeds on garden paths and golf greens, so that I am afraid I cannot endorse the statement expressed in the abstract that, "it has no effect on growing plants, even in strong doses."

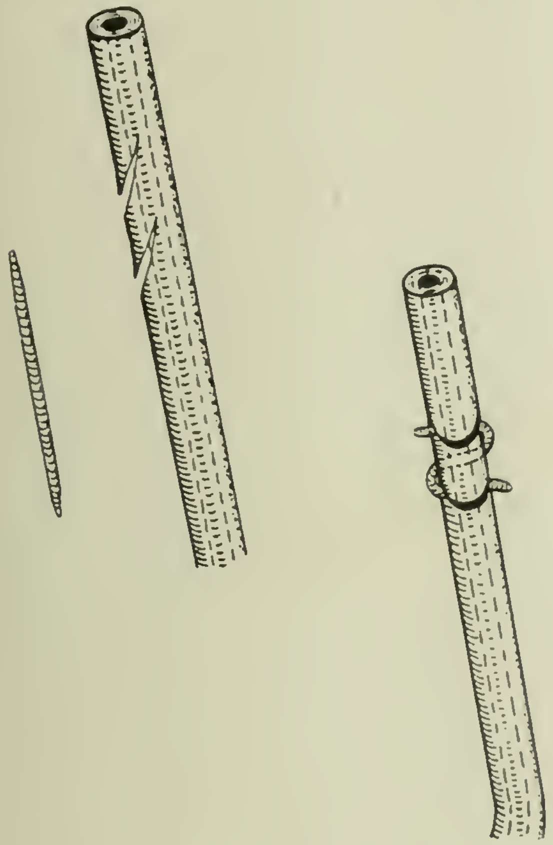
NEW METHOD OF LIGHTING FUSE.

The President: Mr. Weston has a little demonstration to give us in connection with the instructions for the handling of explosives recently issued.

Mr. E. M. Weston (*Member*): I believe you have all noticed that a committee has been appointed recently to draw up instructions with regard to the handling and general use of explosives. I believe that committee was appointed as the result of some remarks I made in a paper about two years ago. Personally, I wish the committee had sent these instructions round to the members for suggestions before printing them. Perhaps the demonstration I am going to give may not be altogether out of place, though it comes too late to be of any use in connection with the printed instructions. Some three or four years ago I think I was the first to draw attention to a very reprehensible practice, which is still in favour in our mines, namely, of using a "cheesa stick" for lighting fuses when firing charges in stopes and drives. The committee say with regard to the lighting of fuses:—

"*Lighting the Fuse.*—The fuse must be prepared for lighting by splitting the end about an inch down with a sharp knife. As soon as an efficient fuse lighter is available the cheesa stick made of blasting gelatine should be prohibited, as poisonous gases are formed by this burning explosive."

I think I can show you that the use of a cheesa stick is absolutely unnecessary except in special circumstances. This is the method which I used underground for a year or eighteen months in actual mining. Here we have an ordinary "cheesa stick" I have made for this occasion, and I am astonished at my own moderation in the amount of gelatine which I used to make it up. It is a common thing for a native to use one small stick of gelatine to make up a "cheesa stick," but there is only about 22 gm. or 25 gm. of gelatine in this stick, whilst there is usually about 50 gm. (stick exhibited). Now it is possible to light up a round of holes conveniently and safely without the use of a stick at all. In the first instance I think the method recommended of splitting down about 1 in. of the fuse is bad, for this reason: where there is any dampness or drip in the vicinity of the holes, quite a large proportion of the core is exposed to the action of the moisture in the air and no matter how vigorously the cheesa stick is applied misfires will occur. The method I used is efficient, and I think it will be agreed that it is a rational method. What I do in the first place is this: I cut two nicks running in the opposite direction to that recommended by the committee, about ¼ in. apart. I then open a stick of gelatine and cut from it very thin strips about the size of whip cord and from 1½ in. to 1 in. long. I then insert one end of this small strip into one cut, and twist it round the fuse so that it cannot be accidentally detached, and insert the other end into the other cut. Then I close both cuts up. It will be seen that I am using a minimum quantity of gelatine, about a 15th of a gram. One gram will be sufficient to fire a whole round of holes. There is practically no vitiation by fumes and I have this advantage also, that I have two chances to light the core if the gelatine in the first nick should fail to light. To light these fuses which are primed with this small amount of gelatine I do not employ a "cheesa stick." The miner who carries a "cheesa stick" inhales the maximum amount of the most poisonous fumes possible to generate in a mine. If a miner burns even the half of one small stick of gelatine he vitiates the air of the drive to an alarming extent. Nearly every miner is supplied with tobacco bags (that goes without saying), and if they will only take the trouble to take a tobacco bag and wrap it round a candle and tie it with numerous separate pieces of string, they have a torch which is very hard to put out and which gives a more convenient light to light up any charges desired. It will last half a dozen or more times, and if required it can be bound by wire to the end of a stick, and with the fuse made up in that way, it may be taken that they can absolutely depend, even more than with a cheesa stick, on getting the holes efficiently lighted. Therefore I think the committee might have recommended that the use of "cheesa sticks" should be absolutely prohibited in our mines, and that this method should be adopted.



The President: I am sure we all thank Mr. Weston for this excellent practical hint, which certainly might be adopted until we get a proper "cheesa stick."

Mr. W. Cullen (*Past President*): I understand Mr. Weston takes some exception to the regulations drafted by a committee of the Council of this Society under the head of the lighting of fuses. Now at the time that particular regulation was drafted, I think I am safe in saying there was no "cheesa stick" on the market, therefore it was decided that until such time as a dependable one was available, blasting gelatine should be continued. Mr. Weston's idea is, I think, admirable and I am certain it would have been incorporated in the report if we had known about it. But the position with regard to cheesa sticks has altered very much since these instructions were drafted. I fully intended to have brought with me to-night two or three samples of "cheesa sticks" which are now in general use. They are cheap and efficient, and have been tried on a great many mines (the Premier mine among them). Probably at the next meeting if anyone has come across the "cheesa sticks" to which I have referred, it may be worth while to have practical experience of them brought before the Society.

Mr. Tom Johnson (*Member of Council*): I have great pleasure in being able to bear out all that Mr. Weston says about this substitute for the "cheesa stick." It has been in use on the Rand for about 14 years to my knowledge, and I think it is a much better system of lighting.

Mr. E. M. Weston (*Member*): I would like to say I do not claim any originality for the torch, but I think my method of putting in two cuts and twisting a piece of gelatine round the fuse is certainly original.

Mr. C. B. Saner (*Vice-President*): I have been trying lately several "fuse lighters" or "cheesa sticks," and they have acted quite satisfactorily when everything was in order. But in one case if a little too much of the mixture was put in the lamp it generated so much heat as to be too hot to hold, or the flame though very strong to begin with soon became thin and small and was put out by concussion. When a certain amount of water was dropping all those tried went out, showing their uselessness in damp places.

I think Mr. Weston is wrong in advocating the prohibition of the "gelatine lighter" in all cases: to take one instance, that of vertical shaft sinking, I have not seen anything to equal a "gelatine lighter" when 50 to 60 holes have to be fired The fuse lighters at present in the market are not "fool proof," but as soon as there is a real proper lighter invented I, for one, will be only too glad to help to enforce its use.

Mr. E. M. Weston (*Member*): In the matter of shaft sinking or where water is present in large quantities, I certainly would not recommend my method in preference to the "cheesa stick." I had not that in my mind when I spoke. I referred to ordinary working stopes and in rises and drives. Desperate diseases require desperate remedies sometimes In wet shafts the fuse is never cut until just before it is lighted.

The President: I am quite certain the "stick" queston is not settled yet, but I think what Mr. Weston has in his mind is that his scheme is not a *gelatine* stick, because it uses so little gelatine.

THE DESTRUCTION OF CYANIDE.

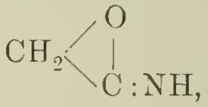
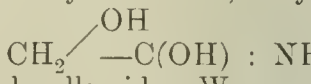
(*Read at June Meeting, 1910.*)

By James Moir D.Sc. (Vice-President) and
Jas. Gray, F.I.C. (Member).

REPLY TO DISCUSSION.

Dr. J. Moir (*President*): There has been very little direct discussion on our paper, if we except the casual references to it by Mr. White. Regarding the latter gentleman's contribution we feel that it is in a sense unfortunate that the custom of the Society precludes the admission for discussion at the same time of two papers on allied subjects, and that in consequence Mr. White's able and thorough investigation into the action of certain "cyanicides" should have to be treated as a contribution to the discussion on ours. He will therefore not expect us to go very thoroughly into his experiments, especially as we shall begin by saying that we are in almost entire agreement with what he has put forward.

With regard to the action of alkaline permanganate, Mr. White has performed a great service in resurrecting the forgotten facts on which it depends. He has shown that the oxidation, by this method, of cyanide to cyanate, a method discovered by Volhard in 1891, is quite remarkably economical and rapid, as in fact the technical success which we saw on the Simmer and Jack mine fully substantiates.* We feel ourselves at liberty, however, to criticise his theoretical chemistry, especially on cyanates He tells us that he attempted to prepare KNCO by a method unspecified, and that the yield was only 6%. By means of permanganate Volhard in 1891 obtained about a 70% yield, and we ourselves find that *gently* heating KCN with excess of litharge yields cyanate quite free from cyanide, and with this product we are totally unable to confirm Mr. White's statement that cyanates are partly reduced (*i.e.*, to prussian blue) by ferrous salts: we cannot get a trace of it, working either in neutral or alkaline solution. We have, generally speaking, to make the complaint against Mr. White that he does not give full experimental details where such are necessary. Thus the $KMnO_4$ titrations on p. 443 would have had something more than technical value if the author had told us how much cyanide he used, and what impurities were in it. The result is that it becomes a matter of chance whether anyone trying to repeat his results will hit on the same conditions, and consequently needless controversy may be excited; a case of which in fact Mr. White has referred to in his remarks. Again we consider that Mr. White's compound MnO_2, MnO_3 is not proven. MnO_2 is not a basic oxide towards weak acids, and therefore such a compound could not hold together in presence of water, any more than, say, aluminium sulphide can. We prefer to think that the precipitate is always MnO_2, and that the excess $KMnO_4$ used is directly employed in oxidising KNCO to carbonate and nitrogen, a reaction which would quite probably be inhibited by coagulants.

As for the action of "formalin," we confess ourselves astonished after our previous experience of cyanhydrin reactions, to find that this one is so rapid and complete. We think that a special explanation of the apparent direct conversion to glycollic acid (possibly glycollamide) is called for, considering as we do that the normal type of equation quoted by Mr. White, is an insufficient explanation. If the intermediate compound had the constitution  it might be expected, similarly to ethylene oxide, to yield at once by hydrolysis  $CH_2(OH)—C(OH):NH$, the tautomeric form of glycollamide. We regret that time did not permit of our following out this interesting reaction, especially as we notice that the substance described in the literature as glycollic nitrile, has properties rather inconsistent with the simple formula $CH_2(OH)\cdot CN$. Nevertheless we had no difficulty in showing the formation of glycollic acid in this reaction. We warmed the mixture of cyanide and formaline, and after a few minutes added an excess of $CuSO_4$, then washed the precipitate well and decomposed it by H_2S and boiled down the aqueous filtrate. We agree that this destruction process for cyanide is, from the practical point of view, an extremely clean one, and by far the most promising if any serious attempt to recover the residual gold is to be made.

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* There is however something wrong with Mr. Powell's figures for moisture on p. 7—perhaps some one will elucidate them.

As for the use of picrates, we consider them to be excluded by the fact that the action is slow and that the engineers appear to be determined to allow only a few minutes for any chemical reaction (in the case of Mr Pam's scheme only a few seconds, unless things can be rearranged so that no one has to work at the exit of the sand bins). Speaking generally of Mr. White's work, we have freely to admit that when we wrote our paper we considered that the only processes economically possible would be the use of $FeSO_4$, of polysulphides and of cuprous salts, and that we never expected that formaline would be rapid enough or permanganate cheap enough.

We think it a pity that the mining firm which is trying the most economical possible method—using the ferrous salt contained in very old accumulated sands to destroy the cyanide in fresh residues—has not come forward publicly to give us its experience. Our only comment is that such a scheme has no chance of success except under the constant charge of a chemist with full powers over the contractors and workmen concerned.

One of us has already explained our attitude towards the use of bleaching powder as "cyanicide." We suggested the use of *acid* solution, because we thought bleaching powder would not dissolve fast enough. Mr. White appears to have surmounted this difficulty, whereon naturally the process becomes probably the best and simplest available at present—at all events, it will be nearly fool-proof, to use Mr. Saner's expression.

At this stage we have to remark that it has been customary in the past in this Society for authors to lighten the solidity of their replies to discussion by indulging in more or less acid repartee at the expense of their critics. Of this custom both of us have taken advantage in the past—sometimes too full advantage perhaps—but we now feel that a certain amount of disappointment might ensue if our reply were as aridly scientific as we tried to make the original paper. We therefore hope that the staider members of the Society will forgive us if we once more let ourselves go. Mr. White shall be the chief victim. Firstly we want to know what on earth induced him, the Chairman of our Standing Committee on pure metallurgical English, to suggest the introduction of such monstrosities as the words *phem* and *phine*. We, as chemists, cannot be expected to approve, when the terminology of our science—probably the most exact terminology in the world—is watered down to the level of the "average man on shift": we think rather, with all due deference, that the gentleman in question might try to attain the mountain-top levels in which the chemist soars. Besides, we fear that the word *phenolphthalein* is not the only one which this type of shiftsman cannot spell, and we suggest that he should thank his stars that he is not making his living out of a European chemical industry where real 'teasers' may occur, *e.g.*, Tetramethylparatolyltriaminodiphenylalphanaphthylcarbinol.*

Secondly we seem to have touched some spot or other in our remarks on correct cyanide titration. Mr. White says he does not think that any cyanider worthy of the name is ignorant of certain facts. Our comment is that since a great many responsible people who do cyanide titrations can be shown not to know these elementary facts, these people are therefore unworthy of the name of cyanider.

What Mr. White contends is that correct methods are on record in our *Journal*, and also, occasionally, in the text-books; whereas we contend that what the Rand needs is that the man on shift shall always know about these correct methods—which is all the more reason why every "man on shift" should join the Society and should study its proceedings, old and new, as if his salvation depended on it. We accept, with due humility, Mr. White's correction regarding the ratio usual between "total cyanide" and "free cyanide," but we should have been more comfortable about it if Mr. White had put on record what he means by "free cyanide," on which subject, doubtless from his long experience, he is much more authoritative than we are: but when Mr. White talks of "fairly definite and consistent analyses" on cyanide works, well, we *hae oor doots*. Perhaps Mr. White keeps a special definition of the term *analysis* for these occasions. We, on the other hand, consider that the methods employed on cyanide works, although they may be of some value, in intelligent hands, for the purposes for which they were devised, because they are moderately self-consistent—that these methods are nevertheless at best mere rule-of-thumb and of no scientific value. In fact we think that Mr White himself, in his more callow days, would have ventured to call the term "free cyanide" an "effete superstition." Anyway it is "up to" his sub-committee to define these terms properly.

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*The base of a blue "aniline dye." The systematic name of its quinonoid salt is probably the longest on record, viz., [illegible]

$Me_2N \cdot C_6H_4 \cdot C \cdot C_6H_4 \cdot NMe_2Cl$

$C_{10}H_6NHC_6H_4Me$

not merely giving the *technique* of the processes which are supposed to determine them

A similar matter in which we think there is a looseness of thought or definition is the idea that atmospheric oxygen can oxidise cyanide to cyanate. So far as we know there is not a scrap of direct evidence for this happening and it is a pity the statement gets taken over from one text book to another without investigation. That the loss of HCN on exposure to air (through CO_2 and hydrolysis) is in itself ample can be easily shown by putting a very dilute cyanide solution in a thin layer in a large dish, acidifying it and exposing it without agitation at an open window. With ·01 strength we found that about 50% of the HCN was lost in an hour (the solution having been made alkaline at the end for titration).

We admit freely that the title of our paper might have been better chosen—we certainly discussed the point enough—but we venture to think that it has served the purpose we had in view. We consider that if and when a cyanide has been converted by any process into another substance which does not give the specific cyanide reactions, it is then to all intents and purposes "destroyed." It is quite true, however, as Mr. White says, that soluble ferrocyanides decompose very slowly on standing with evolution of HCN. We are under the impression that this had been predicted by the pioneers of the ionic theory, and we confess that we considered the change infinitesimal and never expected that the dissociation could be experimentally demonstrated. We now find that the tolidine copper acetate paper (apparently the most reliable of the five) shows the liberation of HCN from even neutral K_4FeCy_6 in quite a short time. From the practical point of view, however, we fear no danger from this dissociation unless the ventilation of the mine also becomes "infinitesimal." According to the Mining Regulations Commission, the Langlaagte Deep mine, the worst as regards air, gets only one change of air a day, but even this, though shocking from the health point of view, would suffice to dissipate any prussic acid arising from dissociation of ferrocyanides. In addition to these facts, it will be remembered that we proposed to use enough iron to produce *insoluble* ferrocyanides, and the latter obviously must dissociate far less than the soluble ones. On the other hand, Professor Findlay, of Birmingham, has suggested that the first precipitate when excess iron is used, may contain insoluble ferrous cyanide. This supposition would account for some of our results where we treated the product with drastic reagents.

We are told that it will be too costly to maintain a high alkalinity in the sands after deposition, but we desire to point out (1) that if the engineering side of the question is properly done, the moisture in the sands need not exceed 15%, and (2) that loss by absorption of CO_2 will only take place at the surface of the stope, and (3) that CO_2 in mines is no longer considered inevitable and will have to be removed for other reasons than that of destroying $\frac{1}{10}$ of a pennyworth of lime, and (4) that the lime lost will pay for itself in preventing corrosion of pumps and other iron by acid water.

As regards the remarkable reaction discovered by Mr. White resulting in the production of HCN from ferric thiocyanate on keeping, we are glad to be able to confirm it at last. By suspending filter paper soaked in caustic soda in the air above $Fe(SCN)_3$ we easily caught HCN and tested it by the prussian-blue reaction, whereas when acidified KSCN was used without iron only thiocyanic acid came up on to the alkaline paper.

[It should be noted that thiocyanic acid reacts like HCN to the tolidine, dianisidine and hydrocœrulignone papers, but does not react appreciably with phenolphthaline or benzidine (pres. of copper understood).] The equation seems to be:

$$6Fe(SCN)_3 + 4H_2O = 6Fe(SCN)_2 + 5HSCN + HCN + H_2SO_4.$$

It is an extraordinary reaction, and we congratulate Mr. White on discovering it, even if the one of us to whom Mr. White refers has to admit his error in questioning the result in 1906—which questioning was only caused by Mr. White's own sketchy and intuitional methods of presenting his results. As for the danger likely to arise from this source in practice, whilst it is undoubtedly capable of becoming appreciable, it ought to be negligible in the future era of good ventilation which has dawned.

We have two small additions to our paper to make. The first is an attempt to cut down the proportion of iron to a minimum. It will be remembered that the ratio Fe : 6KCN was a failure, and the ratio Fe : 3KCN quite successful. We have now tried the ratio Fe : 4KCN with varying alkali (magnesia added after iron).

	Cy′ Fe OH′	Cy′ left		
molecules	4 : 1 : 0	2%	4 : 1 : 2·7	→40%
	,, $\frac{2}{3}$	3 ,,	,, 3·3	→48 ,,
	,, 1	12 ,,	,, 4	→52 ,,
	,, 2	34 ,,	,, 5 3	→54 ,,
			,, 6 7	→58 ,,

The conversion is best when no alkali is present in this case. Working solutions are reduced to ·001% KCN when iron added before magnesia and still lower if magnesia added first.

There is, by the way, a misprint in (λ), p. 439; 61 should read 16.

The other point concerns the theoretical discussion on p. 439. In June we thought that the theory must be wrong because the curve drawn from the mathematical equation agreed pretty badly with the experimental results. We now find that they really agree, but that we plotted the experimental results on a wrong principle We treated molecules of acid as equivalent to minus alkali, although according to the ionic theory, the chemist's sheet anchor in these matters—alkali (hydroxidion) is always present to some extent even in an acid solution. If, therefore, we treat acid as gradually diminishing OH′, as we ought to have done, the curves have the required form all over. We notice, however, that another curve, deduced from the equation $[\mathrm{Fe}^{\cdot\cdot}][\mathrm{Cy}']^4$ = constant, is somewhat similar, and for the present we cannot decide between the alternatives. The new curve is of the form $y^2x=(cx+d)^2$ and taking the particular form of it given by: $\frac{y}{12}=\frac{x+\frac{1}{20}}{\sqrt{x}}$ we find a minimum value (at $x=\frac{1}{20}$ mol. alkali) of 5·4: at $x=\frac{1}{2}$, $y=9{\cdot}3$; at $x=1$, $y=12{\cdot}6$, at $x=2$, $y=17{\cdot}4$: at $x=4$, $y=24{\cdot}3$: at $x=9$, $y=36{\cdot}2$ and at $x=16$, $y=48{\cdot}1$—agreeing altogether very well with the ferricyanide-stopping curve of p. 439. We may mention that we have succeded in interesting some of our scientific friends in the Old Country in this research, and that we hope they will take it up and settle it, with the much better equipment for exact work that is available in the modern British Universities.

As for the practical application of the $FeSO_4$ process, we have had to content ourselves with finding out methods which succeed in the humble hands of the chemist. The Mines Trials Committee has apparently not had time to issue a report on the large-scale work it has done, but we are content to say that from the purely chemical point of view the use of $FeSO_4$ and magnesia gives a perfectly safe product, and we suspect, though not pretending to know anything about costs, that this is the cheapest available process. It should not pass the wit of engineers to devise a scheme for adding powdered calcined magnesia to the sands and sluice them with $FeSO_4$, and finally add lime on their way down. We find that Transvaal magnesia (from Malelane) acts excellently, whereas uncalcined magnesite and also the blue limes are useless. Even if the "conversion" were worse than it actually is, the use of alkali after the sluicing would render the liquor harmless except in the case of gross carelessness. We may mention that one of us took part in some large-scale experiments in the use of $FeSO_4$. The sand was sluiced with $FeSO_4$ solution into an empty boiler until it was half-full, when it was sealed up. After a few minutes the enclosed air reacted with the highly sensitive HCN test-papers; we put in some rats in a cage and they were alive 16 hours later although there was no ventilation. We tried again with small birds about the size of sparrows, which lived through the night but died after 16 hours; but as there happened to be a "cold snap" during the experiment, their fate may have been complicated by pneumonia.

As for Mr. Lofts, he is mistaken in supposing that we found very dilute cyanide solutions to be the most easily converted. Nevertheless we are inclined to agree with him that working cyanide strengths could be reduced still further, by using higher alkalinity and taking special measures to ensure precipitation of the dissolved gold. He thinks that salammoniac is an oxidiser—which it isn't. Nevertheless we happened to try it, because in its presence ferrocyanide combines with calcium to an insoluble precipitate CaK_2FeCy_6. It was found to be disadvantageous, as it spoilt the iron-magnesia conversion.

Finally, we are torn by conflicting emotions in regard to this question of destruction of cyanide, being glad that the problem has been solved, and at the same time really quite sorry that it didn't take five years to solve it, so that in the meantime some visible effect towards disappearance of those eyesores, the old dumps, might have been achieved. Being long-suffering residents of the town (and quite free from the trammels of working costs), we agree with Mr. Lofts that the mines ought to make an effort to shift those dumps "for their own benefit." Johannesburg, "with its seven hills," might then be quite a decent place to live in.

ADDENDA.

(*1*) *Sensitiveness of the Prussic Acid Tests.**—Experiments were made to find what proportion of hydrocyanic acid gas in air could be detected by the four tests depending on oxidisable diphenyl-derivatives. Small quantities of dilute cyanide solution were placed in a crucible at the bottom of a dry ten litre bottle, one drop 20% H_2SO_4 added, and the contents of the crucible boiled from the outside by means of a large burning-glass. Following are the results:

(*a*) With one part HCN in 30,000 of air, the reagents are instantaneous except benzidine which takes about 2 seconds.

(*b*) With 1 part HCN in 200,000 of air, hydrocoerulignone and tolidine are practically instantaneous, dianisidine requires about 20 seconds, and benzidine about 2 minutes.

(*c*) With 1 part HCN in 500,000 of air, the results are similar, but dianisidine takes about a minute, and benzidine does not react definitely.

(*d*) With 1 part HCN per million of air, hydrocoerulignone takes about 3 seconds, tolidine about 6 seconds, and the others are unreliable

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*See Proceedings Chemical Society (London), May 5, 1910, and this *Journal*, April, 1910, page [illegible]

(c) With 1 part HCN in two millions of air, hydrocœrulignone reacts in 10 seconds, and tolidine vaguely in 20 seconds.

(2) *Sensitiveness of a Mouse to HCN.*—The HCN gas from 2·2 mg. KCN was liberated in a 10 litre bottle and a mouse introduced. It became paralysed in 6 minutes and unconscious and practically dead in 14 minutes. The animal recovered however in fresh air in about 20 minutes. The amount of HCN in the air was 1 part in 10,000 originally, and the dose absorbed probably not more than half the total HCN present. viz., about ½ mgm.

It follows that the use of the test-papers underground gives a margin of at least ten times for people exposed to any fumes from stope-filling, because with 1 part HCN per 10,000 air, man could probably survive much longer than a mouse, although at this altitude the danger of fainting is probably pretty serious.

Mr. C. B. Saner (*Vice-President*): I think we must thank our President and Mr. Gray for their excellent and characteristic reply.

EXPERIMENTS ON THE PRECIPITATION OF GOLD FROM CYANIDE SOLUTION BY CARBON IN LIME.

(*Read at April Meeting, 1910.*)

By Edward H. Croghan (Member of Council).

REPLY TO DISCUSSION.

Mr. E. H. Croghan (*Member of Council*): There is little to be done by way of reply to the actual criticism on my paper. The discussion has been confined to narrow limits and has not been on the broad basis desired, namely some definite reply to the four questions tabulated by me, and originally emanating from our late esteemed member, Mr. Wager Bradford.

Mr. A. McA. Johnston, in his remarks as President, hoped to have contributed further detailed discussion which I feel assured would have been of great practical value.

Mr. Toombs contends that the way in which my experiments were conducted was unsatisfactory: in other words, that the manner in which I prepared my carbon negatived the value of the results. The conditions arranged by Mr. Toombs were quite different from mine. Mr. Toombs operated on a finely-ground artificial mixture of carbon and lime and dissolved out all the lime, thereby exposing the bare carbonaceous residue to the solvent action of the acid. In my case I maintained an excess of lime and picked out lumps of carbon which were still coated with carbonate of lime. This is indicative of the fact that no acid action had affected these lumps of carbon. I might also have avoided any misconception if I had qualified my remarks about keeping the mixture cool by stating that at no time did the temperature exceed 40° C.

Mr. Toombs mentions that during the course of some elutriation experiments on slime residue, he caught, on a 90 x 90 sieve, an assortment of carbonaceous matter which varied from 0·05% to 0·2% of the total dry slime and assayed from 10 dwts. to 20 dwts. per ton. These figures, he states, speak for themselves. I quite agree, but in reply would state that the lime must have been of peculiar quality, because these figures, assuming that 3 lbs. of lime were added per ton indicate a content of carbon in the lime varying from 1·6% to 3·2%. On the face of it this is impossible, and so there must have been other sources.

Mr. Toombs appears to be under the impression that I do not acknowledge carbon as a precipitant of gold, but this is quite incorrect. What I desired to convey was that under Witwatersrand conditions, *i.e.*, using 3 lbs. of lime per ton of ore, lime specifications were being made most unduly restrictive in the matter of carbon contents. A proof of this will be shown later.

Messrs. Clark & Sharwood's paper on "The precipitating effects of substances containing various forms of carbon and cellulose on cyanide solutions containing gold and silver" is very valuable, but to refer to it as Mr. Toombs does is overdoing matters a good deal. It is one thing to discuss the detrimental action of carbonaceous matter in experiments on solutions to which have been added 1·6% of coal, coke or charcoal; in another case of 6·6% of coal, and quite another matter to compare these to the subject under discussion. A reference to Table II. of my paper will show that the carbon percentage varies in the worst lime of 0 5% to the best lime of 0·01%, or an equivalent percentage when mixed with 1 ton of ore of from 0·0025% to 0·00005% carbon.

My second point is that the Rand ore itself contains carbon. This evening I have brought some specimens showing carbon in ore which have been kindly lent to me by Professor Young. Amongst these is a piece of banket from the Meyer & Charlton Gold Mining Company showing grains of *visible* carbon.

Professor Young in a recent paper entitled "The Replacement of Quartz by Carbon in Rand Banket," read before the Geological Society of South Africa, says: "The carbon in the Banket occurs chiefly in the matrix of the rock and only very rarely in the pebbles." In a previous paper on the "Auriferous Conglomerates of the Witwatersrand," he states that he has examined samples from various gold mines, such as the Meyer & Charlton, Crown Deep, Simmer & Jack, Van Ryn, North Randfontein, Rietfontein A., etc., etc., and "the form in which the carbon commonly occurs is that of small, black, opaque nodular grains, having a somewhat dull surface and varying in diameter from one millimeter downwards. Sometimes, as in the Buffelsdoorn specimens, the nodular grains, as they increase in number, blend together into one compact mass, which is anthracitic in appearance. Still another form is met with in Rietfontein, where it occurs not only in nodular grains, but also in veinlets with a columnar structure, the column being perpendicular to the walls of the veinlets."

In a paper by Mr. C. Baring Horwood, entitled "The Mode of Occurrence and Genesis of the Carbon in Rand Bankets," also recently read before the Geological Society of South Africa, some analytical data are given.

In typical samples of the Randfontein leader taken from the North Randfontein Mine carbon came out at 0·01 %, 0·03 %, and from the West Reef in the same mine one sees 0·02 %, 0·03 %, 0·08 %, and carbon in the West Reef Dyke 0·07 %, 0·12 %, 0·22 %, 0·08 %, 0·05 %, 0·06 %, trace 0·04 %, etc. These are almost all higher than the percentages of carbon permitted in limes, viz., 0·02 % or 0·01 %, which are specified as the figures not to be exceeded in lime contracts. As no data were available on *local* Rand Banket. I personally took a random bulk sample from the sorting tables of one of the mines not a mile from the General Post Office. The sample was typical Witwatersrand conglomerate consisting, as usual, of rounded quartz pebbles lying in a fine-grained silicious matrix mineralized with iron pyrites. The sample contained 0·003 % carbon. This carbon can be separated, and if added to a gold cyanide solution acts as a precipitant.

Professor Young states that it is not an easy matter to obtain samples showing visible carbon, as these samples occur only in patches, but there is nothing to show at present that carbon, invisible to the eye, is not very common all through the Banket. Mr. A. F. Crosse has often spoken to me of the gold in residues which, no matter how ideal are conditions, cannot be extracted by cyanide. Perhaps the presence of carbon in Banket to some extent explains this, and it may well be that until this question has been adequately dealt with it is unlikely that further improvements in extraction will be achieved. In my opinion there is now a distinct demand for a thorough research into the question. From the various papers on this question and the result quoted by me on a random sample showing no visible carbon a curious problem presents itself: Limes are specified here to contain not more than, say, 0·02 % or 0·01 % carbon. Thus on a maximum basis of 3 lbs. of lime per ton of ore, a bad lime containing 0·5 % carbon when mixed with a ton of ore gives in the total a percentage of 0·0025 %—a figure which is still exceeded by the carbon in my random sample of Banket. How extraordinary is the position when we come to limes containing 0·02 % or 0·01 %, equivalent when mixed with a ton of ore to 0·0001 % and 0·00005 %! Briefly, my sample shows 30 to 60 times more carbon in the natural banket ore than could be due to the lime last referred to.

Since others will not reply to the late Mr. Wager Bradford's questions, I beg to do so by stating that in my opinion he was correct in surmising that the theory that the amount of carbon in the limes used locally had any markedly adverse effect was far-fetched and based on insufficient evidence. Nor has the discussion of this paper brought any such evidence to light.

The President: I am sure we have to thank Mr. Croghan for his interesting reply, and especially for his valuable facts in regard to carbon in banket. I thought that the modern conclusion is that it is not the carbon itself that causes precipitation but the gases occluded in it. If that is the case, then I see no discrepancy between the author's results and the question under discussion.

THE CLASSIFICATION OF TAILING PULP PRIOR TO CYANIDING.

By Edward H. Johnson, M.I.M.M. (Past-President).

Of late the necessity of good classification for efficient tube milling has been generally realised and discussed,* and with the removal of amalgamation plates from the stamp mill the importance of this operation is enhanced, in order to ensure that all coarse gold passes into the tube mill circuit for amalgamation, and none overflows into the cyanide plant. As regards another equally important phase of classification, the methods of separation of sand from slime in tailing pulp, as distinguished from concentration of specifically heavier constituents of the ore, have undergone a number of developments since cyaniding first was practised on these fields. Commencing with the collection of sand in dams where products classified themselves with little artificial assistance, there followed the collection in rectangular pits, where by means of a gate overflow a fairly clean sand was obtainable, the slime being settled in dams at a lower level. As illustrating the natural concentration and classification which takes place in a shallow stream of low velocity the accompanying curve of values of sand collected in a pit 25 ft. × 50 ft. (for which I am indebted to Mr. J. H. Johnson) is instructive. (Fig. I.) An adaptation of this principle was used effectively as a means of concentration prior to the war.* The cost of rehandling dam and pit settled products led to the erection of special vats for collecting purposes with filter bottoms and side or bottom discharge doors. Primarily these were fitted with Butters' distributors and peripheral overflow of the slime.† The relation of pulp volume to the superficial area of the vat was not at first recognised, and where large diameter vats (in proportion to the volume of pulp) were installed, the distributor fell into disrepute on account of the amount of slime collected with the sand, and likewise because, being a moving device, it periodically required a little attention. This led to adjustable gate discharges at the periphery of the vats with hose distribution of pulp—the system most extensively used on the Rand to-day. This system involves in practice a number of miniature buddle concentrations, and may at different moments eliminate sand or retain slime according to the care exercised, whilst the amount of slimy (– 200 mesh) product retained in the sand charge may be very large* All these methods require that the sand so collected shall be transferred to other vats for treatment, either from density of the settled product or want of uniformity of distribution. The danger of contaminatng mill water acts also a a deterrent against direct treatment by cyanide in the collecting vats.

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* See this *Journal* [illegible], Feb., [illegible] Vol. [illegible], p. [illegible]

Three noteworthy attempts have been made to produce a cleanly classified product which might advantageously be treated in the condition collected. First, Mr. Hennen Jennings' system of hydraulic classification with inverted pyramidal classifiers at the Crown Reef.† As no slime treatment was practised at that date (about

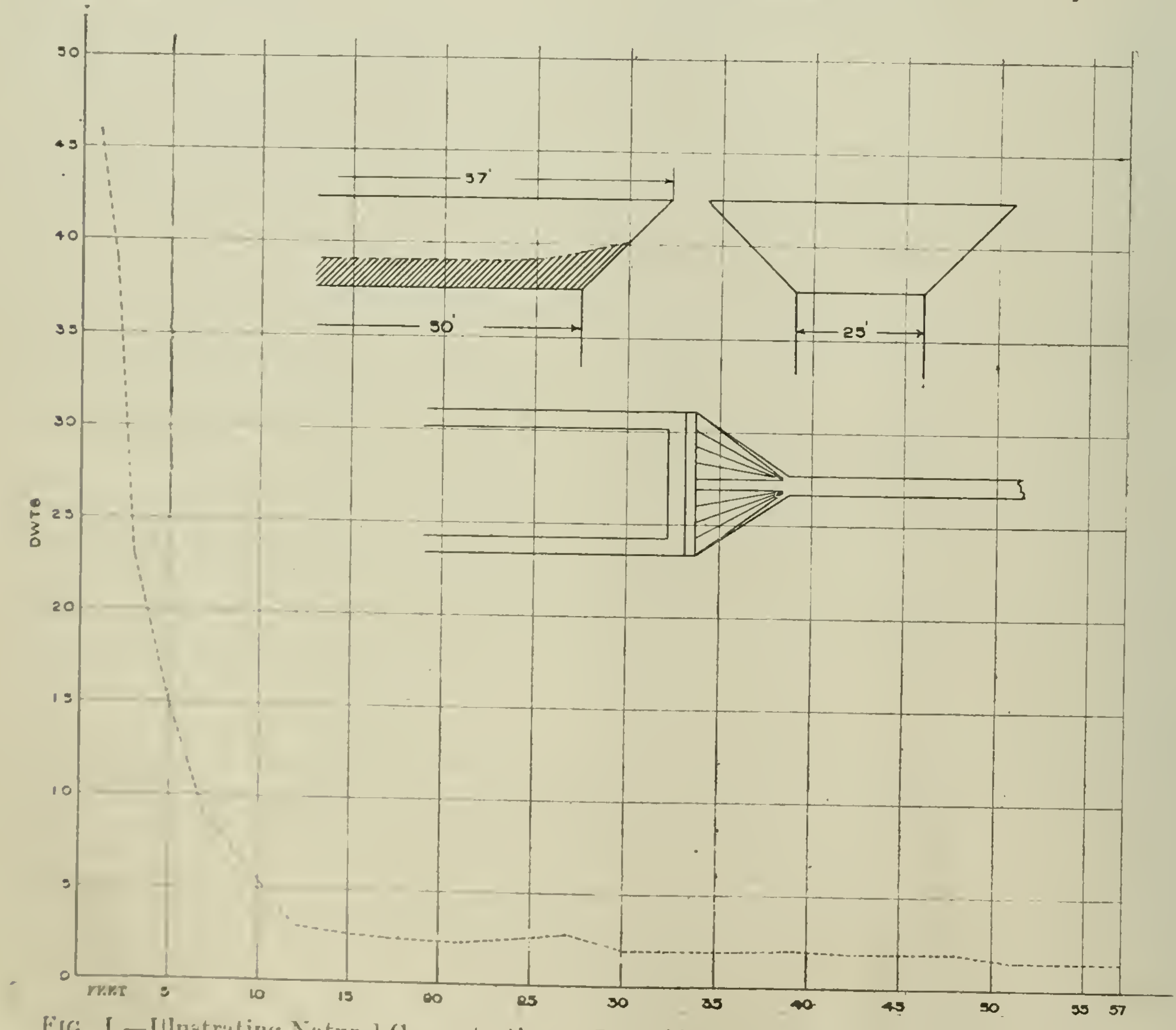

FIG. I.—Illustrating Natural Concentration. Inset Sketches showing Dimensions of Pit and Distribution of Pulp along the End Wall.

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* See this *Journal*, Vol. ii., p. 439.

† See this *Journal*, Vol. I, p. 15.

*See grading analysis of sand residue given in this *Journal*, May, 1910. Vol. x., p. 399.

†See this *Journal*, Vol. I., p. 4.

1894) the elimination of 45% of the crushed ore to the slime dams, where it lay untreated, rendered the system uneconomical at the time for the separation of sand and slime, though extensively used for the separation prior to lengthy cyanide treatment of a coarse pyritic product termed "spitzlutte concentrates." Second, Messrs. Hellmann & Crosse's separation by Wilfley tables. This was effective, but where concentration was uneconomical proved expensive. The third method is that devised by Dr. Caldecott, and described in a paper published in the August *Journal* of last year. As the East Rand Prop. Mines plants erected on the continuous sand collecting system were not completed until some months after the close of the discussion on his paper, I was unable to contribute from experience gained in operation, and I now wish only to deal with the classification aspect of this type of plant, the economic and mechanical features having already been fully dealt with by the inventor himself.

The combined East Rand plants of four 20 ft. sand filter tables, each with their complement of four primary cones (9 ft. × 8 ft.) and two washing cones (8ft. × 5 ft.) (flow sheet of which is given in Fig. II.) have for some months been handling the sand from approximately 4,000 tons

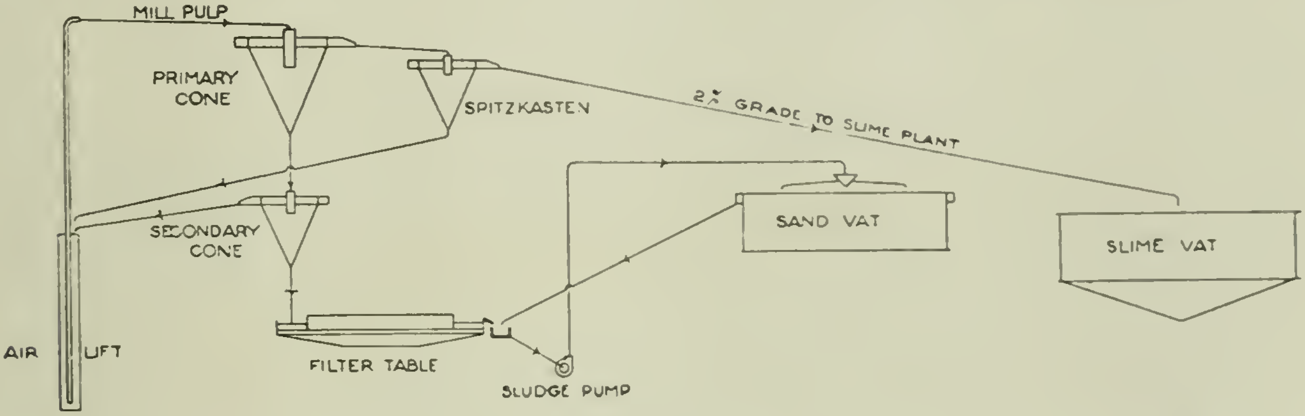

FIG. II.

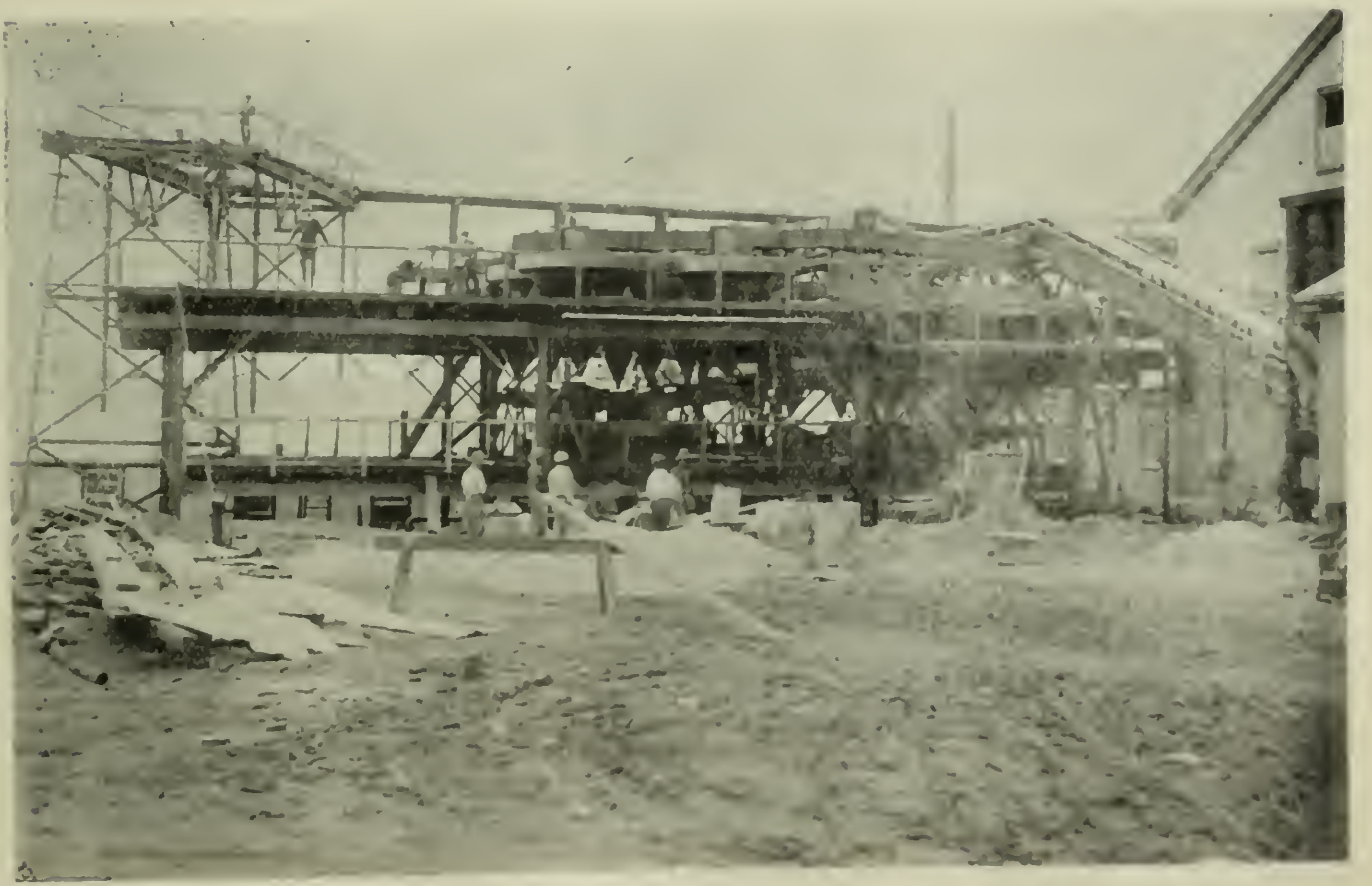

FIG. III. General View of Continuous Sand Plant (Cason Section).

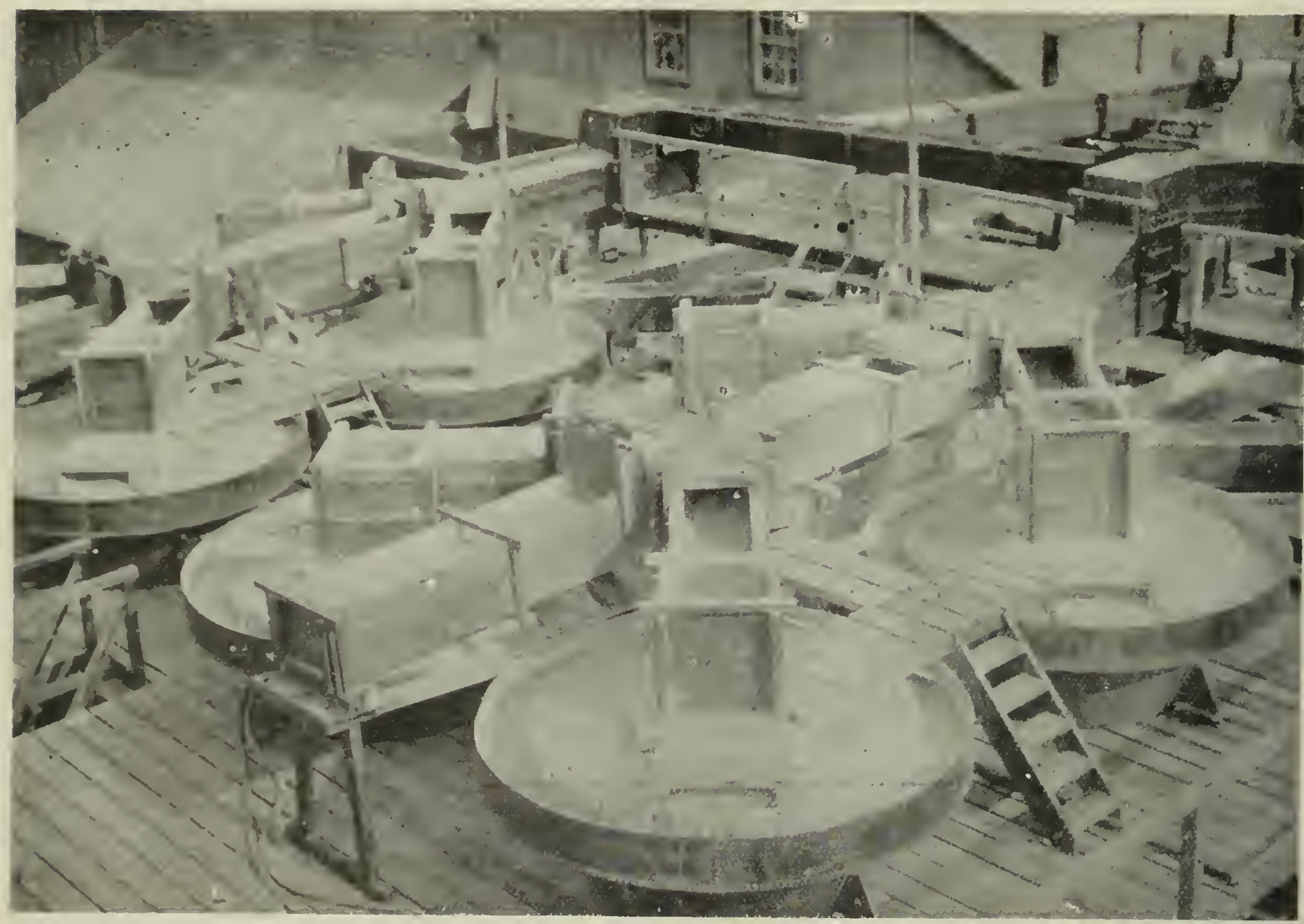

FIG. IV.—Distribution of Pulp to Primary Cones.

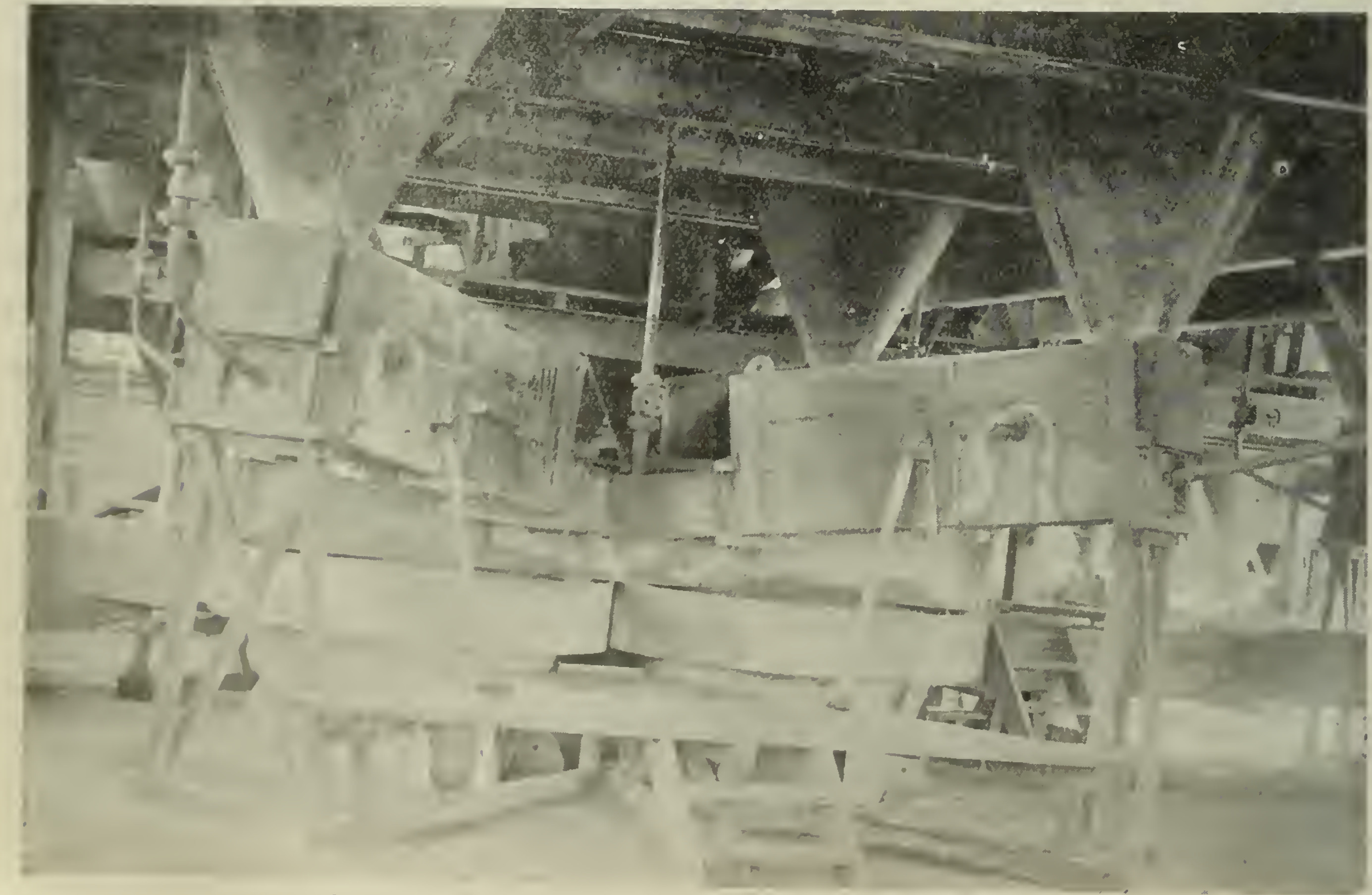

FIG. V.—Washing Cones, showing Water Introduction.

of tailing per day, equivalent to nearly one and a half million tons of ore per annum and effecting a separation of a little over 50% of the pulp as a slime containing on the average under 2% remaining on a 200 mesh screen. The slime content of the sand determined by elutriation is under 1%, giving a product of extreme leachability in the condition in which it is collected. The collection takes place in vats full of solution and the distribution is by Butters' distributors, the solution overflowing the periphery and returning to the launder carrying the sand from the filter tables. The accompanying graph (see Fig. VI., p. 162) of the values of solution leached during treatment of a collected charge, leaching commencing when the vat is half full of sand, illustrates the rapidity with which solution of the gold takes place without transfer or other means of disintegration or aeration of the deposited sand.

The efficiency of the classification is well illustrated in the accompaning grading and chemical analysis, which is typical of a considerable number that have been executed.*

It will be noted that whereas the – 200 sand collected with the sand product contains 8¼% to 9% of pyrite, the same product (– 200 sand) contained in the slime only contains 1·8 to 1·9% of pyrite, while the elutriated slime contains 1% to 1¼% pyrite. The data given shows that the diaphragm cone classification not only serves to effect the retention of the great bulk of the pyrite in the sand, but that at the same time the size separation is likewise efficient, nearly all the – 200 mesh product in the tailing pulp going to the slime plant whilst 88% of the sand is + 200 mesh. The relative percentages of pyrite in the various sand grades bears also upon the question of tube mill classification and crushing, showing the disposition of the friable and heavy pyrite to become comminuted in greater proportion than the silica.

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* Compare Proc *I.M.M.*, Vol. xiv., Oct., 1904, p. 53.

EAST RAND PROPRIETARY MINES, LIMITED.—SECTION, ANGELO.
SAND CHARGE No. 4,650.

	Original.	Residue.
FeS_2	2·65%	2·62%
Gold	7·15 dwt.	0·75 dwt.

Extraction = 89·51.

Screen.		Grading per cent. by weight.		Per cent. by weight of FeS_2.		Per cent. by weight of total FeS_2 present.		Gold dwt. per ton.		Per cent of total Gold present.		Dwt. Gold per 1 per cent FeS_2.		Extraction.
Mesh per lin. in.	Aperture.	Original	Residue	Original	Residue	Original	Residue	Original	Residue	Original	Residue	Original	Residue	Per cent
+ 60	·01 in.	8·4	8·1	0·88	0·80	2·8	2·2	5·2	1·72	6·2	17·8	5·9	2·15	66·9
+ 90	·006 in.	33·3	32·9	1·14	1·08	14·1	12·4	4·7	0·76	22·1	32·1	4·1	0·70	83·8
+ 200	·003 in.	46·3	45·0	2·68	2·65	46·3	41·5	5·5	0·64	36·0	36·9	2·1	0·24	88·4
– 200	·003 in.	12·0	14·0	8·22	9·00	36·8	43·9	21·1	0·72	35·7	13·2	2·6	0·08	96·6
Averages & Totals		100·0	100·0	2·68	2·87	100·0	100·0	7·08	0·78	100·0	100·0	2·64	0·27	88·98
Concentrate		2·7	2·25	65·97	71·0	71·5	63·5	109·2	9·8	43·9	28·0	1·6	0·13	91·0
Tailing - -		97·3	97·75	0·73	0·98	28·5	36·5	3·9	0·56	56·1	72·0	5·3	0·57	85·6
Averages & Totals		100·0	100·0	2·49	2·62	100·0	100·0	6·74	0·77	100·0	100·0	2·71	0·29	88·58

SLIME CHARGE No. 4,197.—From above sand.

Screen.		Grading per cent by weight.		Per cent by weight of FeS_2.		Gold dwt. per ton.		Extraction
Mesh per lin. in.	Aperture	Original.	Residue.	Original.	Residue.	Original	Residue	Per cent.
+ 200	·003 in.	2·21	2·41					
– 200	·003 in.	38·79	38·56					
Total Sand -		41·0	40·97	1·81	1·90	4·8	0·16	96·7
Slime by elutriation		59·0	59·03	1·19	1·25	2·0	0·06	97·0
Calculated assay		100·0	100·0	1·39	1·52	3·15	0·10	96·83

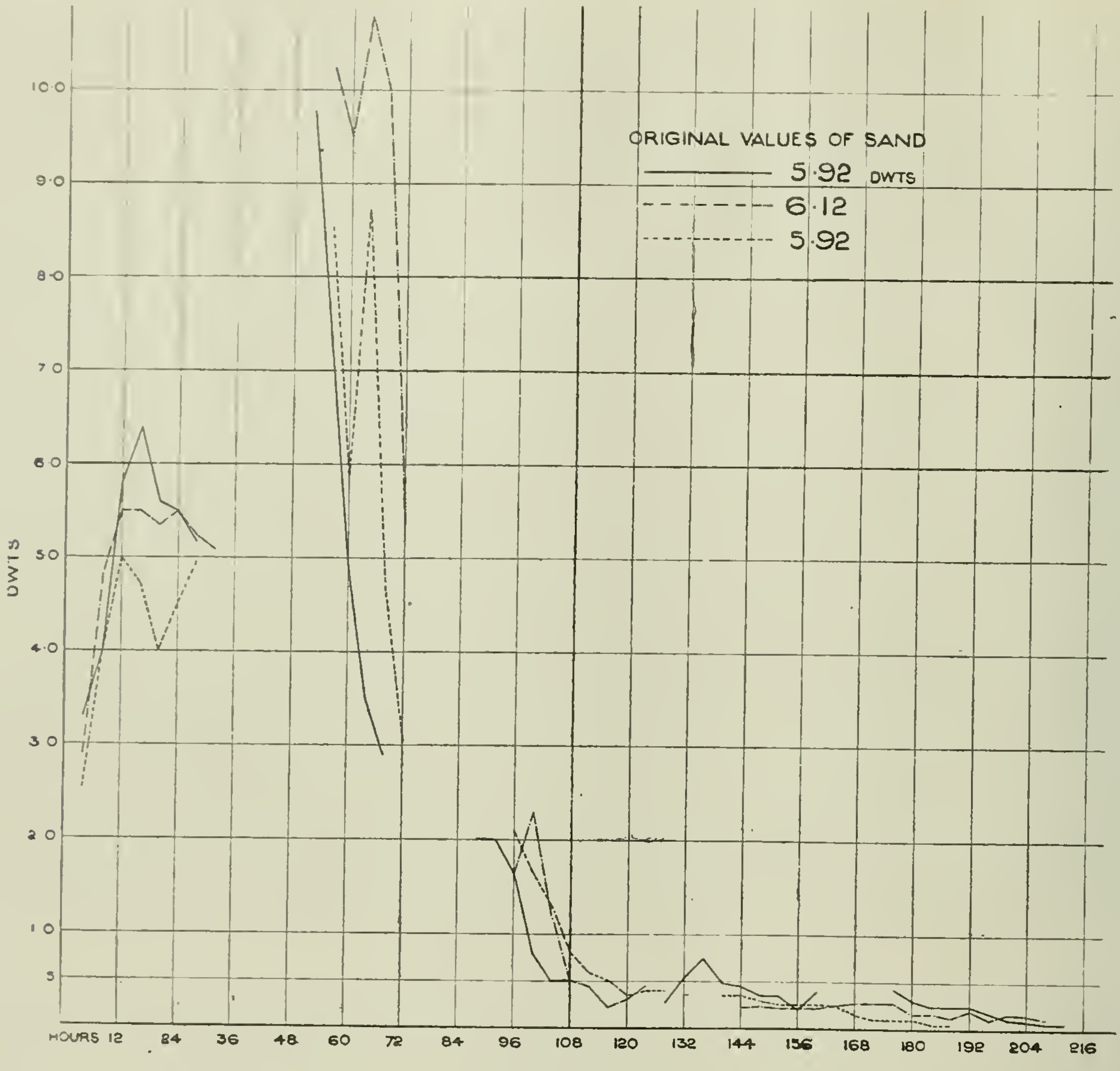

Fig. VI.

The highest extraction is obtained in the product having the greatest percentage of pyrite and the finest comminution, showing that it is the fineness of crushing which is of value in extraction, not the elimination of pyrite. The column giving the calculated gold value for each per cent. of pyrite indicates this very clearly, the + 60 product containing only 0·80% pyrite giving 2·15 dwt. for each per cent. of pyrite, as against the – 200 product containing 9·0% pyrite giving only 0·08 dwt. for each per cent. of pyrite. Also in the concentrate portion the extraction is 91·1% as against the sand 85·7%.

The essential difference between the system of diaphragm cone classification and all previous efforts in this direction, is the maintenance of a large steady underflow containing a minimum of moisture, as opposed to the usual thin fluid underflow, the slime elimination being proportionate to the reduction of moisture in the underflow. Classification by means of an uprising current of water is more suitable for the separation of coarse pyritic material in pulp. The effectiveness of the system of classification by the maintenance of minimum moisture content in the underflow is shown in the low slime content of the sand and the low pyritic content of the slime.

A somewhat novel feature of this system of classification is the fact that a ton of clean sand occupies a greater cubic space than under the older system. Conversely, a ton of settled slime containing much fine (– 200 mesh) sand occupies less space. The reason of this is that the voids (interstitial spaces) in the sand usually occupied in part by particles of slime are under this system true voids. Over 20,000 tons of the sand discharged from the vats have been weighed to determine this point, and has resulted in the certainty that the sand deposited under solution in this system of classification occupies 23·5 cub. ft. to the ton.

This approximates very closely to the condition of a transferred sand after application of solution. Coherent masses of sand or slime (lumps) are unknown, and the moisture on discharging averages 10·8%. Similarly, determination of the slime has shown that the inclusion of the finest product of crushing has resulted in settlements containing 36% moisture or 30 cub. ft. per ton, approximating somewhat closely to vacuum filtration in moisture content. The density of settlement will naturally vary with the amount of argillaceous matter contained in the ore. For the method of determination of the moisture content of slime I am indebted to Mr. R. E. Sawyer, who designed the apparatus used (Fig. VII.), and carried out the determinations. The following description of apparatus and method is by Mr. Sawyer :—

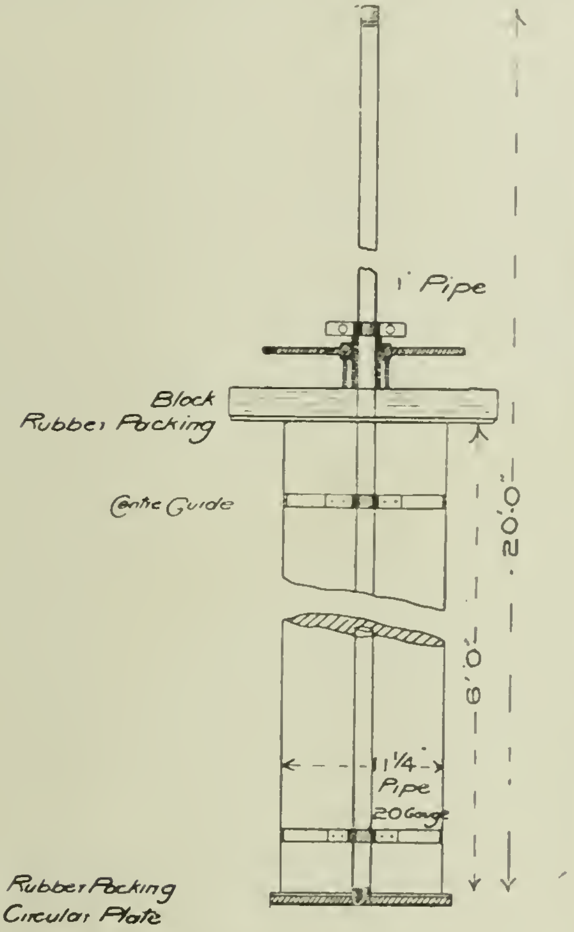

FIG. VII.—Slime Sampling Apparatus.

"The tonnage of slime is customarily determined by measurement of the depth of settled slime after decantation of the supernatant water or solution, the presence of surface blisters being the indicator of good settlement as due to solution escaping from the lower layers under pressure. The uncertainty of sampling the large volume delivered by a 12 in. pipe for specific gravity determination led to an effort to obtain a direct core sample for density determination, and after considerable experimental work the apparatus here described and illustrated proved most satisfactory.

Description of Apparatus and Method.—The apparatus consists of a long 1 in. pipe, into one end of which is screwed a circular disc 12 in. diameter, to which is glued a ¼ in. rubber disc. This is lowered into the tank when empty and securely fastened to the platform to the centre of the vat, the lower end resting flat on the bottom of the tank. The charge is then pumped in and allowed to settle and the solution decanted in the ordinary way. An 11 in. galvanised iron pipe, fitted with centre guides is then slipped over the 1 in. pipe and dropped into the tank, falling freely about 12 ft. The depth of the slime inside and outside the pipe is then accurately measured. A block of wood lined with rubber is slipped on to the top of the pipe and the whole thing clamped and tightened up with a big wing nut. The charge is then washed out of the vat and the contents of the pipe transferred to tared vessels, weighed and dried, after re-measurement inside the 11 in. pipe to ensure no leakage has taken place. The big pipe is calibrated by weighing when filled with water to various depths.

Method of Calculation.—We now have data for two independent methods of calculation.

(*a*) By volume. (*b*) By weight.

(*a*) The volume and weight of the slime sample, and weight of equal volume of water is known. From these the specific gravity of the slime pulp is calculated and the specific gravity of the dry slime being known, all other details, such as per cent. moisture and cubic feet per ton, are easily determined.* The specific gravity of the dry slime was determined in the laboratory from a number of samples taken at two minute intervals from the discharge pipe.

(*b*) The wet and dry weight of the slime sample being obtained by direct weighing, the per cent. moisture, etc., are at once known.

Form of Observations and Calculations.

	Depths			
Date 22.7.10.	First Section. ft. in.	Third Section. ft. in.	Outside Pipe. in.	Inside Pipe. in.
Charge No. 72 Tank No. 15	2 2	2 3	29	30

Corrections for internal and external dimensions.

Total.	Water lb.	Slime lb.
2·98	1·79	1·19

Tube.	Vessel.	Gross.	Tare.	Net.	Corrected.	Gross Dry.	Net Dry.	Corrected Dry.
A	1	98	23·5	15·9	161·98	119·0	102	103·19
	2	108	23·5					

	Per cent. moisture.	Cub. ft. per ton
By volume	35·6	29·3
By weight	36·3	29·6

The results obtained from these two methods should agree to within 1% of moisture. With reasonable care an even closer agreement can be obtained any difference between measure inside and outside pipe is taken to be due to loss of slime at 60% moisture. This figure was taken after many experiments, and is on the safe side. It makes, however, very little difference to the result."

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* See this *Journal*, Vol. II., pp. 87 and 474.

The President: I am sure we are unanimous in according a hearty vote of thanks to our esteemed Past-President for his present paper. I would only like to remark that the work done reflects the greatest credit on the East Rand Proprietary Mines and on the inventor of the scheme.

NOTES ON THE OCCURRENCE AND CONCENTRATION OF TIN ORES.

By C. Fred. Thomas (Member).

Cassiterite and the commercially important associated minerals.—Cassiterite is the mineral from which practically all tin is obtained. It occurs sometimes massive, sometimes granular and often so finely disseminated that its presence is difficult to detect.

Respecting its commercially important associated minerals:—*Wolfram* is a by-product of importance in Spain, Cornwall and the Straits Settlements. Its specific gravity is almost the same as that of cassiterite so that a separation by means of wet concentration is impossible. Previous to the use of magnetic separation, roasting with carbonate of soda was used, the wolfram being converted into tungstate of soda. The Wetherill magnetic separator was first used in Spain for the separation of wolfram from cassiterite. Germany is the largest buyer of wolfram, and there by electric smelting it is converted into tungsten and ferro-tungsten. Its value has fluctuated during recent years from 16s. to 57s. 6d. per unit of tungstic acid per long ton. Tungsten is used in the manufacture of high speed steel. At East Pool mine (Cornwall), over 20 years ago, the wolfram was separated from the tin ores and sold at a price ranging from £2 to £4 per ton. The present price for 65% tungstic acid is approximately £100 per ton.

Mispickel.—An extraction exceeding 1% arsenious anhydride (As_2O_3) from tin ores is not uncommon. After roasting selected mispickel, the residues often contain over 5% cassiterite. The value of arsenious anhydride has varied from £12 to £34 per long ton during the last six years. The extraction of mispickel is low by wet concentration, and selection previous to milling is advisable.

Chalcopyrite and other copper bearing minerals. Wet concentration for the extraction of these various sulphides is unsatisfactory. In the Calstock mines, East Cornwall, the ore averaged ·5 to ·7% of copper. The concentrate previous to roasting averaged 2·5 to 3% copper. The recovery by precipitation was 7d. per ton milled, with copper averaging £60 per ton. The calcination was forced, and waste burnt leavings contained 1 to 1·5% copper, probably due to conversion into the insoluble oxide by too great heat.

Silver.—Selected copper ores in East Cornwall averaged 1 oz. of silver in each unit of copper. Cross-courses have proved rich in silver near the contact with the tin bearing lodes.

Stannine, Tin Pyrite or Bell Metal Ore, which is a combined sulphide of tin, copper and iron, is a comparatively rare mineral found near the contact of slates with granite. It is probable that the extraction of tin from this mineral after roasting would be very low.

Production of Tin.—The world's production of tin is approximately 100,000 tons per year. The Straits Settlements, Bolivia and Banka Billiton account for 85% of the total, whilst Australasia Cornwall and Africa produce 12 to 13%. Bohemia and Saxony are working low grade cassiterite ores associated with wolfram and bismuth. Wood axles and lifters for the stamps were in use there in 1907. In the Province of Yunnan, Western China, there are several thousand Chinese working on low grade ores, but the bulk of the production does not reach the coast. Alluvial workings are responsible for the chief source of production, but the proportion of mined cassiterite is rapidly increasing.

Occurrence.—Cassiterite is mined from various deposits, which include: Lodes, fissures, beds, pipes, stockwerks, contacts and "gash veins," but in all the cases I am acquainted with, granite is found in the near vicinity.

In Cornwall, Dolcoath is mining from a depth of approximately 3,000 ft. The lodes strike nearly east and west, and dip south. In the slates (upper levels) these lodes were worked for copper, but on their entering granite at greater depth, the amount of copper decreased, whilst that of cassiterite increased. The ore from the bottom of the mine scarcely requires calcination, this being due to the fact that there the sulphides have almost disappeared. The ore is extremely hard and tough, and the cassiterite is very finely disseminated throughout. The concentrate from this ore is almost free from impurities, whose presence might have a hardening tendency on the metal, and, as sold, contains 7 to 9% moisture. The average assay value of the concentrate is 64% to 66% tin, and, because of the fluidity of the tin product in smelting, it realizes a comparatively high price.

The Carn Brea, East Pool and South Crofty mines are working on lodes striking approximately east and west, but some lodes dip north and others south. The north dipping lodes fault those dipping south. It is peculiar that the younger or north dipping lodes contain the most complex ores, wolfram being one of the contained minerals. Elvans in the granite often contain cassiterite on the faces only.

In the Transvaal the tin-bearing pipes are unique and are found in granite. In the felsites I have found tourmaline "eyes," measuring a few inches, containing cassiterite. Some tourmaline outcrops in felsite show traces of cassiterite. The shales where they have been intersected by the narrow fissure veins have been highly silicified and contain impregnations of finely disseminated cassiterite. (Dolcoath has mined granite containing 30 to 40 lb. of cassiterite per ton for great widths near the walls of the main lode.) Warm springs are found near the cassiterite deposits of the Transvaal, and are also encountered at depth in the Cornish mines: possibly an indication in favour of the infiltration theory. It is important to note that no tin mines in this country have yet attained a vertical depth of 400 ft.

Bolivian tin ores often contain a considerable proportion of antimony and wolfram. Their production has rapidly increased during the last few years.

Concentration.—The concentration of cassiterite is a widely discussed problem, and plants must vary according to the particular ores to be treated. I venture the opinion that the average extraction obtained from mined cassiterite does not exceed 80%. Previous to designing a plant I should strongly advise preliminary tests with the vanning shovel, with which much valuable information can be obtained Particular attention should be paid to any sulphides that may be present, to prove whether these sulphides contain cassiterite.

Slime accounts for the greatest proportion of loss in freed cassiterite. A mill dropping 90 times per minute will create more slime proportionately than with 100 drops. Pneumatic stamps dropping 135 to 140 times per minute will not slime to the same extent as a Californian mill with the same mesh screening. Possibly the greater agitation in the mortar box may account for this.

Sand from a modern plant with good classification should not be worth further treatment (or the tailing from the tables in front of the mill can be made too low in grade for further consideration). The object of wet concentration is to eliminate the maximum amount of the crushed ore as waste in the first concentration. In a particular case with a recovery of 29 lb. cassiterite and wolfram per ton, the rough sand assayed 2·5 to 3 lb. per ton; a concentration of this sand was attempted by various means but with practically no result, proving that the cassiterite was not free.

Chemical analysis of the ore and tailing is useful, but it only shows actual recovery. It does not give information regarding how to concentrate by means of water, as one is dependent on the specific gravities of the minerals. I am aware of no plant obtaining an extraction equal to that of an expert with the vanning shovel. The man in charge of all preliminary concentration must, to a very great extent, rely on vanning shovel tests. Chemical analysis is of vital importance in the final treatment of tin and wolfram concentrates.

Plant.—As descriptive of modern practice I append notes on a plant erected at South Crofty mine, Cornwall, in 1906-7 for treating complex ore containing cassiterite, wolfram, arsenic and copper, and also working data in connection therewith.

Headgear 80 ft. high, grizzlies $1\frac{1}{2}$ in. apart. Fine to bin and mill. Rough sorted and then to 18 in. × 10 in. jaw rock breaker, then to trommel with $1\frac{1}{4}$ in. holes. Fine to bin and mill. Rough to sorting belt and on to 15 in. × 9 in. jaw rock-breaker. Mine trucks hoisted in cage and arrangements made for sending through complex or simple ores separately.

Mill, 40 heads, 1,065 lb. Bremner mortar boxes with rounded corners. Drop $7\frac{3}{4}$ in. Discharge $1\frac{3}{4}$ in. to $2\frac{1}{4}$ in. Drops per min. 102. 20 mesh gun metal screening. (NOTE.—Gun metal screens costing 9d., as compared with steel $4\frac{1}{2}$d., per sq. ft., stood 5 days, as against 36 to 48 hours.) 30 b.h.p. motor to each 10 heads. Cam shaft to each 5 heads, Duty 2·8 to 2·9 long tons per 24 hours.

Classification into four divisions by Luhrig classifiers:—

No. 1 rough to buss tables—
yielding
- *a.* Concentrate to calciner.
- *b.* Middle head to tube mill.
- *c.* Tailing to waste,

No. 2 rough to frue vanners—
- *a.* Concentrate to calciner.
- *b.* Tailing to waste.

No. 3 rough to frue vanners—
- *a.* Concentrate to calciner.
- *b.* Tailing to waste.

No. 4 slime to settlers then to buss slimers, rag frames and round tables
- *a.* Concentrate to calciner.
- *b.* Tailing to waste.

Tube mills or pulverisers taking the middle head from buss tables, together with various products after calcination.

To frue vanners—
- *a.* Concentrate to calciner.
- *b.* Tailings classified: slime re-treated, sand to waste.

In pulverising great attention must be paid, so that a minimum quantity is converted into slime, the regulation of the tube mills depending on the grain of the cassiterite contained in the sand.

Strips.—All tailing passed through strips measuring 80 ft. to 100 ft. in length and 15 in. to 18 in. wide. Grade 1 in 8. The concentrate obtained treated according to quality.

Concentrate previous to roasting was obtained in four sizes. The average value was :

Cassiterite	180 to 190 lb. per ton
Wolfram	40 to 60 lb. ,,
Mispickel, equivalent to arsenious anhydride	200 to 250 lb. ,,

A small amount of copper.

Calcination by means of five Brunton calciners each with 15 ft. diameter revolving beds. The ironwork in each weighed 14 tons, and the inclusive erected cost for each calciner was approximately £500. Capacity per calciner per day of 24 hours was for a sweet roast 4 to 5 tons of concentrate, and the coal consumption averaged 10 to 12 cwt. The inclusive cost of roasting one ton of concentrate varied from 4s. to 5s. The tonnage calcined as compared with the tonnage milled varied from 10% to 15%.

Arsenic flues built of masonry, with plate iron doors, were provided for each chamber. These flues were cleaned out every four to eight weeks, and the quantity of arsenical soot, averaging 92 to 95% arsenious anhydride, As_2O_3, varied from 40 to 80 tons. This product was sold locally for refining, and its value per ton varied from £8 to £28 12s. 9d.

Magnetic Separation.—Two Wetherill separators were erected at an inclusive cost of nearly £2,000. A considerable amount of experimental work was necessary and it was found advisable to feed No. 1 rough concentrate direct to the separators from the calciners at a temperature of approximately 100° F. Nos. 2, 3 and 4 concentrates were brought by means of further concentration after calcination to 65 – 70% cassiterite and wolfram ; dried in a revolving cylinder and then passed through the separators.

No. 1 separator : 2 magnets were used for the strongly magnetic iron product, and 4 magnets were used for the weakly magnetic wolfram product.

No. 2 separator : 2 magnets were used for the strongly magnetic iron product, and 2 magnets were used for the weakly magnetic wolfram product.

The iron product contained 3% to 8% cassiterite, but very little in a free state. The current to these magnets varied from 1·5 to 3 amps. at 110 volts.

The wolfram product averaged 40% to 55% of tungstic acid. The current to these magnets varied from 10 to 14 amps. at 110 volts. This product was sent to lead tanks 20 ft. × 5 ft. × 1·25 ft., and a charge per tank was about 4 tons. Concentrated sulphuric acid was added to cover the charge, and the time of acid treatment varied from five to seven days. After a thorough washing and drying the residue was passed again through the separators. The amount of cassiterite liberated varied from 10% to 25%. Sulphuric acid cost £2 7s. per ton at the mine, and the consumption was 10 cwt. per ton of wolfram. The non-magnetic portion containing the cassiterite was easily cleaned and made marketable.

It is impossible to give complete details of treatment in a paper of this kind, and I have therefore omitted mention of kieving, buddling, etc., for final cleaning.

Copper.—The solution from the first concentration after roasting was sent to the precipitation tanks.

Electrical Power Costs.—Power was bought at 0·45d. per unit plus £6 per k.w. of maximum demand per annum, the peak having to remain constant for 20 minutes. The average cost was 0·7d. per unit. The price quoted for electrical power on winding engines was 1·5d. per unit, and steam winders were installed accordingly.

Analysis of Power Costs.	Pence per ton milled.
Magnetic separators	0·60
Vanners, tables, slimers, etc.	1·23
Calciners and small concentrators ...	0·33
Return water service	0·57
Mill	9·32
Lighting	0·53
Pulverisers or small tube mills	0·76
Rock-breakers, trommel and sorting belt...	1·85
Condensing plant...	0·23
	15·42

Marketable Products Recovered.—

Cassiterite 23 lb. per long ton containing	62 to 63% Sn.
Wolfram 6 lb. per long ton containing	60 to 65% WO_3.
Arsenic soot 1 to 1·3% containing	92 to 95% As_2O_3.

A small amount of copper.

The iron product from the separators was sold to the smelters direct and averaged 3% to 8% SnO_2. Smelters paid a high value for tin contents in order to secure the iron for smelting.

The Tailings after passing through strips, were settled in three large ponds, and the overflow water was returned to the mill by centrifugal pumps.

No. 1 pond contained chiefly sands averaging 5 lb. cassiterite and wolfram per ton.

No. 2 pond contained chiefly middles and slime averaging 12·3 lb. cassiterite and wolfram per ton.

No. 3 pond contained slime averaging 15·0 lb. cassiterite and wolfram per ton.

The fine sand and slime settled in Nos. 2 and 3 ponds were re-treated. A test made with the slime from No. 3 pond on a frue vanner, without the side shake, gave a recovery of 53%.

Notes on a Plant erected in South Africa for very fine grained cassiterite.—Grizzly bars, 1½ in. apart. Rock-breaker, 20 in. × 10 in. jaw to 1¼ in. Mill: 10 heads weighted to 1,400 lb.; drop, 7¾ in.; 102 drops per min.; discharge, 2 in. to 2½ in.; 16 mesh screen; duty, 3·1 to 3·4 short tons per head per 24 hours. Classification into four divisions: No. 1 rough to No. 5 Wilfley tables, elevator wheel for returning middle head discarded; No. 2 to 6 ft. frue vanners; No. 3 to 6 ft. frue vanners; No. 4 slime to rag frames and round tables. Calcination in a small reverberatory furnace for pyritic concentrate. Concentrates as shipped assayed 50% to 56% Sn and 90% would pass 150 mesh screening.

Reporting Values.—Unless the company is smelting its concentrates and producing metallic tin, values should be given in concentrates. With values given in metallic tin the values of the ore can not be calculated by referring to the cash price in London. One ton of metallic tin in one class of concentrate is not of the same value as a ton in another class of concentrate.

The following example will illustrate this:—Consider two shipments, each containing 7 tons of tin metal. We will assume cash price of tin in London £145, with three months delivery £143.

No. 1 Shipment.

	£	s.	d.
10 long tons of concentrate assaying 70% at £92 per ton	£920	0	0

No. 2 Shipment.

	£	s.	d.
14 long tons of concentrate assaying 50% at £56 per ton	784	0	0
Less additional transport in the extra 4 tons...	24	0	0
	£760	0	0

	£	s.	d.
The tin metal in No. 1 shipment realized per ton	£131	8	6
The tin metal in No. 2 shipment realized per ton	108	11	6
In each case per ton of metallic tin the approximate difference is ...	22	17	0

Due allowance has also to be made for such impurities as copper, wolfram, sulphur, arsenic, silica, etc., and the penalisation for these impurities.

Smelters' Charges.—The difference between the cash value of tin compared with that at three months is usually an important factor to consider, however, during the past one or two years there has been no very wide difference

In Liverpool and Cornwall the ticketings for or public auctions of tin concentrates from various parts of the world show very keen competition among the smelters. The Cornish ticketings are small as compared with those at Liverpool, but fortnightly sales often realise over £20,000. For medium grade concentrates exceeding 62% tin metal, one can estimate approximately the cash price per ton of concentrate. With cash and three months values within £3 to £5 per ton: (A.) Allow a margin of £2 to £3 below cash price. (B.) Deduct 6·25% from assay value. We will assume cash value of tin in London £145, and at three months £142, assay value of concentrate 66·25%. Make up price on £143 and 60%, all calculations being to the nearest 2s. 6d. Price = £85 15s. 0d. It is reasonable to expect a somewhat higher value per unit for 70% than 62% concentrate, but these figures will be found approximately correct.

If concentrates are sold by private treaty an arrangement is sometimes made with the smelter on a basis of, say, £8 per ton smelting charge on 70% concentrates and an extra 5s. to 10s. per unit below 70%. If the smelters charge per ton of concentrate on 66·25%, is approximately £10, then for 50% concentrate the charge may be from £15 to £17 per ton.

Future Prospects of Tin Mining in the Transvaal.—Origin of the Cassiterite.—"Granite is the mother of tin" is an old Cornish expression with a great amount of meaning. Concentrates obtained from granite not in the vicinity of known lodes have shown traces of tin. Felsites in the Transvaal contain cassiterite in small quantities widely distributed, and small fissures in this rock often contain rich impregnations.

Infiltration is a theory difficult to disprove. Taking the important tin mines of this country, the Zaaiplaats and Groenfontein pipes seem to indicate infiltration from beneath. They are possibly old geysers filled by means of infiltration. Rooiberg lodes show impregnations of the surrounding country rock with cassiterite along bedding planes. The deposition from solution will occur where the solution can most easily percolate. The South African Tin Mines show well defined fissures in shales, with impregnation of cassiterite in the shales near the fissures. Further exploration in depth is necessary to prove these deposits, but every fact seems to argue in favour of the source being beneath. If so, the future prospects are bright. It is reasonable to look for large deposits with depth, and although poor zones will probably be met with, depth is the important factor.

Tin Ores.—A close study of these ores is necessary, and they will probably prove an important national asset. Attention should be given to the sulphides in these deposits, and results may prove that cassiterite is closely associated with them. A pyritic ore requires more attention from a concentration standpoint but the additional cost of treatment by calcination, etc., should not exceed 1s. per ton milled. Some difficult ores for concentration have been encountered, but with depth such difficulties may not be met with. Complex ores are to be expected, particularly in deposits above the granitic rocks. Slime may not have had the necessary treatment, and great losses in extraction have occurred for this reason.

General Remarks.—Mining—The tin miner pays great attention to the amount of ore that can be mined after the blast, and he is careful to keep at hand the necessary tools for this work. The single hammer is not general in mining tin ores, because of the hardness and toughness of the rock. It is usual to see two hammers to each drill and often 20 drills are used per ton mined. The blast dislodges a proportion of ore but a considerable amount is fractured and can be dislodged by tools previous to starting drilling again.

It appears from the examples quoted the mill duty is low; generally tin ore is hard and tough and mill duties very low accordingly.

On Rand ore the tonnage per head per day with similar screening would probably be double those quoted.

Concentration.—Sizing for fine grained cassiterite is unsatisfactory, and one is dependent on hydraulic classification. It is essential to have clear water for the concentrating machinery, and the greatest care is necessary in regard to water supply, grade and speed of each concentrator. Regularity of speed is most important, and a separate drive for the concentrating plant is advisable.

The particulars given of values from the large settling ponds prove the difficulties encountered from a slime standpoint. The proportion of slime from 20 mesh screening with 102 drops per min. was nearly 35% on complex ores.

Magnetic separation on concentrated slime has given most unsatisfactory results. It is a question whether it would not be advisable to treat slime containing wolfram and cassiterite by roasting with carbonate of soda.

Elevators for pulp should be avoided if possible. The minimum height required from the top of the mill bin to the concentration floor is approximately 42 ft.

Extraction must vary according to grade and class of the ore to be treated. Taking cassiterite and the important associated minerals, the order of extraction according to percentage of recovery would be :—

1. Cassiterite.
2. Wolfram.
3. Mispickel.
4. Chalcopyrite.
5. Stannine.

Although wolfram is slightly higher in specific gravity than cassiterite, in crushing it creates rhombic flakes that are not so easily saved by wet concentration as the powdered cassiterite.

Classification after the first concentration on frue vanners for particular ores is advocated. It is claimed that an extraction on frue vanners without classification is satisfactory. Dolcoath sends the unclassified pulp direct to frue vanners.

Shipment of Tin Concentrates.—Dried slime concentrate is difficult to handle because of dust. If the slime concentrate is kept separate, it is advisable to sack it with from 3% to 5% of moisture present to prevent leakage through sacks.

The President : I have much pleasure in moving a hearty vote of thanks to Mr. Thomas for his interesting paper. The paper contains a number of practical hints which I think will be of value to our tin miners.

GRADING ANALYSES AND THEIR APPLICATION.

(Read at May Meeting, 1910.)

By H. Stadler (Associate).

DISCUSSION.

Mr. E. H. Johnson (*Past President*): When contributing, in July, to the discussion on this paper, I promised to give particulars of certain experimental work then being carried out, having for its object the increase of capacity of a crushing unit consisting of 10 stamps and tube mill. This object, as forecasted in my previous contribution, I hoped to attain by the separation of the fine ore at the feeder, and the direct delivery of this to the tube mill. I now wish to give results of this experiment.

The apparatus used is a specially designed feeder, which also acts as the separator, and governs the amount of fine ore fed to the tube mill, in proportion to the amount of coarse ore fed to the stamp. The result obtained was a duty of 26·1 tons per stamp with a final pulp containing 0·5% remaining on the 90 (·006 in. aperture) mesh screen. That is, 10 stamps and 1 tube mill crushed 261 tons in 24 hours. The following are the grading results and particulars of the trial :—

Coarse Crushing Experiment.—6/11/10.

Screen 9 mesh. Wire ·091. Aperture ·284.
10 stamps, and 1 tube mill 5 ft. 6 in. × 22 ft.

Running weight of stamp	...	1,634 lb.
Drop	...	$8\frac{1}{2}$ in.
Discharge	...	$2\frac{1}{2}$ in.
Duty	...	26·131 tons.

Gradings.

	Screens.	Launders. (Separated at Feeder)	Final pulp.
+3	—	24·3	—
+6	13·3	27·1	—
+10	9·0	9·7	
+20	19·2	13·7	—
+40	16·5	7·3	—
+60	7·0	2·7	—
+90	7·2	2·8	0·5
+200	7·8	2·9	15·5
−200	20·0	9·5	84·0

Comparing this result with single-stage crushing, where a duty of 5·88 tons per stamp with 1,600 mesh (·015 in aperture) and the following grading was obtained :—

	Per cent.
+60 (·01)	8·7
+90 (·006)	16·5
+200 (·003)	17·3
−200	24·9
Slime (elutriated)	32·1

a comparison of efficiency per h.p. may be made. 10 stamps (4·5 h.p. per stamp) and 1 tube mill (105 h.p.) consumed 150 h.p. in reducing 261 tons to the grading given. Single stage crushing to accomplish the same duty, but to the coarser final pulp given above, would have needed 44 stamps requiring 198 h.p. There is, therefore an increased efficiency per h.p. of 32%. The surface of fracture of the final pulp in each case works out as follows :—

Single Stage Crushing.—129,782 sq. in. surface of fracture.

Double Stage Crushing (with separation of fine).—169,967 sq in. surface of fracture. As compared with single stage crushing the double stage shows an increase of 32% in h.p. efficiency and 31% in grinding efficiency.

As a factor in gold ore reduction, I am of the opinion that we are only just beginning to realise the possibilities of the tube mill. It has been too tenderly treated in the size of ore particles fed to it up to the present, and its gluttony for work has not been properly appreciated. On the data available the author's condemnation of the tube mill as mechanically inefficient must, I think, be judged "not proven."

It would be of interest if the author would give the name of the publication in which an account appears of Prof. Kick's work upon the laws of ore crushing, and if he would likewise give us details of the confirmatory experimental work on which Kick's mathematical deductions were based. So eminent an authority as Prof. Richards,* who deals exhaustively with the work of Rittinger and Van Reytt, makes no reference to Kick, whilst Warwick† states that the bulk of recent work by German investigators supports Rittinger's views.

SAND-FILLING ON THE WITWATERSRAND.

(Read at June Meeting, 1910.)

By Edgar Pam, A.R.S.M. (Member).

DISCUSSION.

Mr. J. David Marquard (*Member*) : Thanks are due to Mr. Edgar Pam for his paper on Sand Filling on the Rand. The few remarks I am about to make are not for the purpose of criticising, for no one can afford to criticise as yet ; the subject is still in its infancy, and we mining men here must help to make it a success. I will say right away now that much of our future good and safe mining will depend greatly upon our successful sand filling.

I have just returned from a trip abroad, and spent some little time at Kattowitz in Silesia, where sand filling is largely in vogue. It will be just as well to remind you that sand filling has done much for Silesia. It was started there some six years ago by one of the cleverest engineers in the district. His ideas were largely criticised ; he, however, persevered under trying circumstances, for some of the difficulties were great. Most of these difficulties have now been practically overcome, with the result that to-day one mine is putting down 7,500 tons per day. The ultimate result now is that his mines are working cheaper and are safer than others in the neighbourhood ; his critics are convinced and are now sand filling also. Numerous advantages have been gained in Germany by this system, but that does not directly affect us. I will proceed to some of the problems that confront us here, but leave you with a diagram showing the different stages of the system as there carried out.

The problems facing us are more complicated and present more difficulties and dangers than do those of Silesia. True, they have learned much by hard experience, from which we can get a good start. There is much literature on this subject from theoretical points of view; this is largely a practical subject here and must be dealt with in a practical way.

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* A Text Book of Ore Dressing, p. 186, 1909.

† *The Mining World*, Dec. 26, 1908, p. 17.

Surface Transport.—This will vary on different mines, depending largely upon the respective elevations of cyanide tanks, dumps and shaft collars. The surface work on the whole will present no great difficulties. There is no reason why we cannot sluice the sand with monitor jets direct from our old dumps into launders and so transport it to the shaft tops, or we may lift our current sands (chemically treated against cyanide, etc.) with water from a sump near the cyanide tanks (by means of elevator buckets, centrifugal pumps, or in some cases start up our disused tailings wheels) to such a height as will give good drops for launders to the shafts. It is very essential that the slush should be as thick as possible: a good mixture is 60% sand and 40% water by volume. A launder with 20% dip will allow of this mixture flowing freely. On the Robinson these launders are lined with cement castings, a very good protection indeed. The manager there informed me that these castings are made on the property comparatively cheaply. The chemical destruction of cyanide must be left to our cyanide managers, whose duty it will be to see that the slush is in a safe state for the mines.

Comparison between Silesian and Rand Filling Material.—From the diagram it will be seen that the Silesian sand is sent down with various admixtures, such as clay, slags, stones, bricks, etc. These admixtures have beneficial and detrimental effects:—

Beneficial Effects.—(*a*) The clay lubricates the pipes, aiding transport.

(*b*) The clay with fine slags has a binding effect underground; after one year the sand is as solid and compact as sand stone, requiring a hammer to break a bit off.

(*c*) Slags and other materials give a certain amount of porosity to the mass, aiding drainage.

(*d*) The sand particles are rounded, unlike ours, which are angular.

(*e*) The ferruginous admixtures also give binding effect, for some little chemical action takes place.

Detrimental Effects. — (*a*) The clay causes trouble on the surface, sticking to the mechanical shovels, and choking the crushers, grizzlies, etc.,

(*b*) Slags, bricks, etc., scour the pipes.

For comparison take our material; we find it is nearly pure silica, with cyanide and cyanide compounds. It is not ferruginous, and we have no admixtures for binding or cohesion. On tunnelling into our dumps we find in the centre, although the sand is subjected to good pressure, absolutely no cohesion. This non-cohesive difficulty is one of the greatest we have to overcome. Our sand packs very well, as can be seen at the mines now filling. Whilst this sand is moist it appears to have some cohesive power; once it dries though there will be no cohesion, and a portion of the resultant pressure of the over lying strata will be transmitted to the supporting barriers; I refer to stopes with good dip. It certainly will be beneficial to get our sand as cohesive as possible behind our barriers. Around our mines there is a good thickness of red ferruginous clayey soil; this is often used for brickmaking. I have tried the following mixtures on the surface with fair success:—

90% sand 10% R.F.* soil plus water. Some cohesion and porosity.

80% sand 10% R.F. soil plus water plus 10% fine ash. Some cohesion and porosity.

80% sand 20% R.F. soil plus water. Good cohesion and porosity.

70% sand 20% R.F. soil plus water plus 10% rough ash. Good cohesion and good porosity.

Probably future experiments will find us a good cohesive mixture. It is worth some experiment. The R.F. soil can be shovelled with mechanical shovels on sites for future slimes dams, the excavations being finally used for damming slime.

Pipes.—The matter of suitable pipes for transport in the shafts and elsewhere is a very important matter. The wood-lined pipes described by the author are similar to those used in Silesia. In some mines there is a tendency to condemn these pipes already. It will be well to give them a good trial though, for they are quite successful abroad. In their pipes in Silesia they have:—

(*a*) Clay for lubrication.
(*b*) Good inside diameter, lessening velocity.
(*c*) Grains of sand rounded.
(*d*) Pure water (not acid).
(*e*) Slags, bricks, etc. Experience has shown that these remain somewhat in the centre of the pipes, whilst the sand is nearer the wood lining.

(*a*, *b*, *d*), we can comply with. (*e*) does not affect us. (*c*) we find our sand is angular, and it has a greater scouring effect on our pipes than their rounded sand would have. Porcelain lined pipes are used in the shafts in Silesia; they last a fair time. It may be our sand will not scour porcelain pipes too fast. These pipes, though

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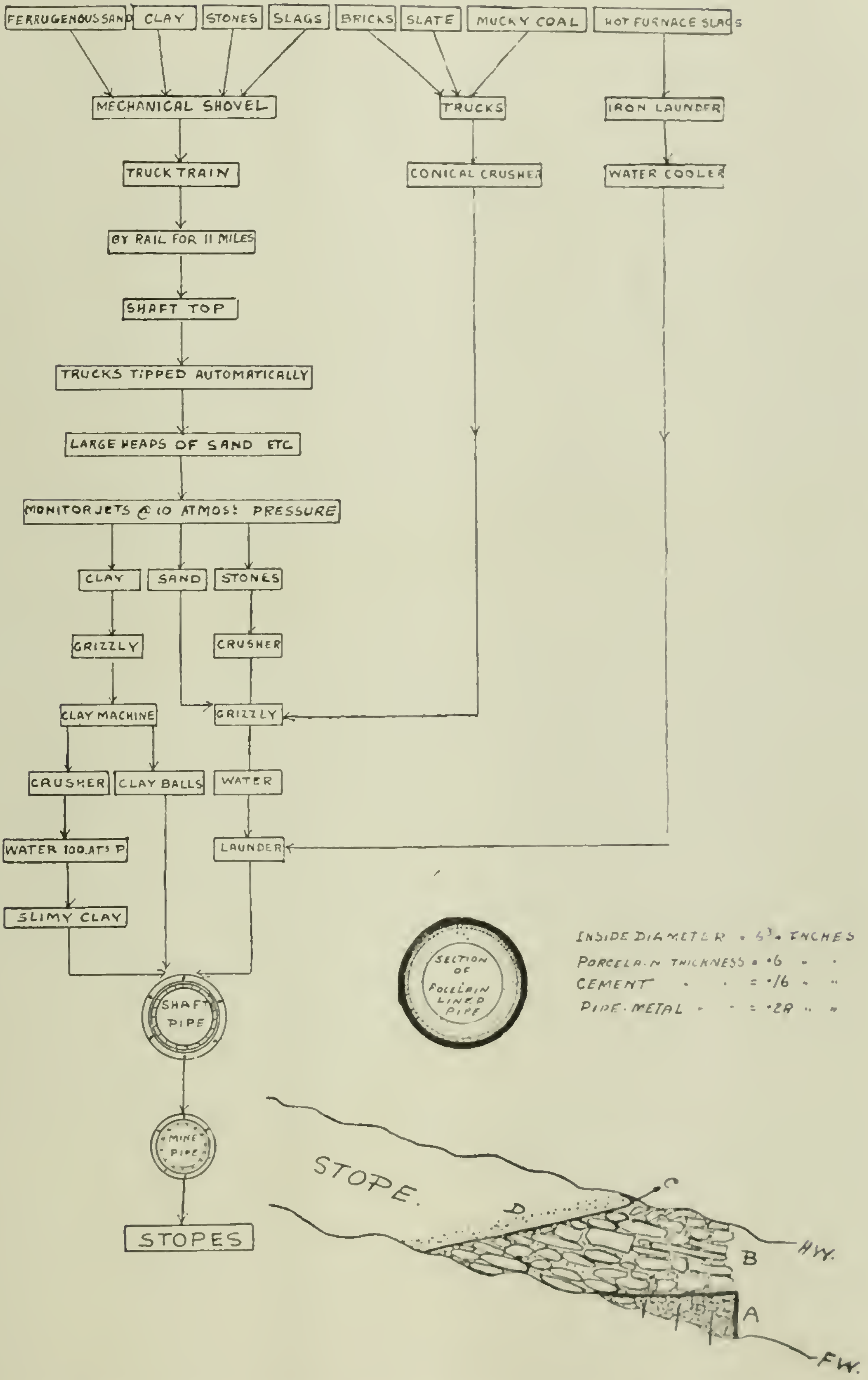
* R.F. = red ferruginous.



Silesian Material for Stope Filling. Diagram showing existing Methods in Operation.

they are expensive, should be given a trial. A good sized pipe in the shaft is an important matter, for then the velocity of flow will not be too high, obviating wear to some extent. An important point is that the pipes in shafts and elsewhere should remain reasonably full. The contents must be taken away below at the same rate as supplied at the top. For this purpose there must be a regulation valve in the mine pipe and a good sized supply hopper on top, to which the shaft pipe line is directly connected. The hopper must hold a fair amount of slush, such that the pipe will remain full if there is a temporary hitch in the surface supply. Telephonic communication allows of proper regulation by the operator below. Disregard of this precaution is one of the greatest causes of wear and tear in pipes, especially the shaft column. Often a sucking noise is heard at the top of the column, and at the delivery end into the stope the slush comes out with spurts under high pressure. This is largely due to air drawn in causing alternately cushioning and releasing, with resultant excessive velocities and jerks in the pipe line, hence wear of pipes and loosening of wood lining. The hopper above mentioned will prevent the sucking in of air. A great point to remember in connection with this is to flush the pipes well with water before and after use, and never to fully close the regulation valve.

The pipes down shafts in Germany are always in duplicate. The system of sending the sand down in a paste through 8 in. boreholes, as at the Simmer and Jack, has many advantages, for it saves piping. Unless this bore hole is used as a pressure column there will be some difficulty in getting the material to the far east and west portions of the mine. Very deep boreholes will cost a good deal to put down, the expense increasing very much after the first 400 ft. There is also the disadvantage of passing through faulted ground or dykes which may cause future blocking of the borehole. It may be that the pipes, etc., will be so costly eventually that we shall have to try boreholes or other means of getting the material down. In the mine itself wooden launders should be resorted to wherever practicable.

Barriers.—Until we are certain that our sand coheres, or have a proper cohesive mixture for filling purposes, I certainly would not recommend wooden barriers: they serve in Silesia, but there the conditions are different. Their barriers are not nearly so extensive as ours will be: their sand coheres and hardens long before their barriers have time to rot: their stopes are flat, causing the resultant pressure on the barriers to be small. Our sand has to remain indefinitely: will the wooden barriers and grass remain, especially if the lagging is wet on one side and dry on the other, two conditions which will rot timber soon. Think of the overlying rock crushing with its resultant pressure on the barrier, and think of this wooden barrier swelling and getting impervious to stope dripping water: much of this water may collect behind the barrier; another cause for anxiety.

As a suggestion the barrier sketched might do well:—

(*a*) Stones masoned with binding jumpers into the footwall; this will have a wedging effect on the overlying waste pack B.

(*b*) B is a waste pack packed well, the cavities being filled with rough ash.

(*c*) Cocoanut matting (if ash has good filtering effect then matting is not necessary).

(*d*) Two feet layer of rough ash; it is essential that all future water must be drained, therefore the barrier must be porous.

Stope Filling and Dewatering.—Here we will learn much by future experience: on the mines which are now sand-filling good progress has already been made. It seems, however, that the more places there are to fill, the better the filtering and de-watering of the sand can be regulated. There must be at least two stopes to work consecutively day by day, each stope having a full day in which to prepare it for the next supply. It is an advantage to have a period between each filling, for then the sand can settle properly, leaving the water to percolate through the sand, or, clear above the sand.

It is best and cheapest to have no side barriers, but to keep boys heaping the free end up and letting it lie at its natural angle of repose, or less. The clear water is removed through old disused 3 in. pipes. These pipes have 1 in. holes drilled every 9 in. along their lengths. The pipes are laid on the footwall from bottom to top of stope; the holes have wooden plugs in them, which are pulled out to let the clear water off.

The following are some of the points to watch in stope filling and dewatering of sand:—

(*a*) Fill at the furthest point away from the drainage pipe.

(*b*) Have no side barriers, the sand will lie at its natural angle of repose at the free end.

(*c*) Where wooden side barriers are preferred it would be best to have the filling pipe nearest to, and the discharge pipe furthest away from, the barrier, because all the coarse sand remains nearer the former pipe; this would make a better packing against the barrier. The other way will leave slimy sand against the barrier which often forces its way through during filling operations, generally slushing the drives, etc.

(*d*) The sand must be so arranged in the stope that the water flows off at the junction of the footwall and the sand, and by no means against the hanging.

(*e*) The supply launder should drop the slush from the hanging wall or as high as possible, and must be led right up to the sand; the slush must not be allowed to run in from the top of the stope on the footwall.

(*f*) When starting a stope see that the first sands are well packed against the barrier; this is an important point, and will prevent much slime, etc., getting into the drives at the commencement.

(*g*) The length of the filling face must be as long as possible; the longer the better.

(*h*) The discharge or drainage pipe can have more than one hole every 9 in., say, 3; this will allow of quick dewatering; after the clear water is off, the holes are again plugged and filling can be restarted.

(*i*) Between the supply and discharge pipe arrange small paddocks with bits of board, this will retard the flow, causing the sand to settle and the water to arrive reasonably clear at the discharge pipe.

In Australia stopes are being filled successfully with dry sand at a fairly low cost. Distribution takes place underground with chutes, travelling belts, etc. It will certainly do us no harm to study their system. With the electrification of many of our mines in the near future power can be got cheaply and easily underground; some of this power could be used for transporting appliances for sand. It would not be a bad idea to get our sand down the vertical, dry; it can be fed by means of a screw conveyor if near the dump, or otherwise, into a strong square wooden launder, 12 in. × 12 in. The launder must be fixed in the corner of the pump compartment, then dry sand is shot down into a funnel-shaped bin into which jets of water are playing all round. The funnel shaped bin can be placed in such a position in the shaft as would give some head for the mine distributing pipes. The mine pipes are directly fixed to the bin. This arrangement will save:—

(*a*) Extra length of piping in shaft which is so subject to wear.

(*b*) Pumping of sand filling water to the surface; the actual amount coming to the surface being what the mine makes.

Most of the remarks I have made are suggestions only, and I trust more will be heard in the Society on sand filling, for I am sure we all want to learn. The subject here is still partly in the experimental stage: it has to get out of that stage soon, for it is important that our overlying strata should be kept up. It is also important to fulfil all the advantages mentioned by the author in his paper. There are many other advantages as well which we shall see come to the fore as soon as practical sand filling is an accomplished fact on all our mines. We are gaining our experience, and practical knowledge is the outcome of experience: when we have that practical knowledge sand filling will be successful, and with that success we shall be able to alter many of our present mining arrangements underground for the benefit of the Rand generally.

Mine.	Sand.	Clay.	Stones.	Slag.	Bricks.	Slate.	Murky Coal.	Red Hot Slag.
Gluckauf	70	16	14	–	–	–	–	–
Ferdinand Ludwig ..			–	95	5	–	–	
Mysolowitz ..	80	10	10	–	–	–	–	–
Gieschi ...	90	5	5	–	–	–	–	–
Laura Hulle Richter ...	50	45	5				–	
Florentine ...	30	20	–	–	10	7	8	25

Mr. A. R. Hughes (*Member*): As the filling of worked out stopes with sand has provided quite a new experience for underground men on the Simmer and Jack mine, it is with considerable diffidence that I venture to give my experience and a short description of the methods employed in connection with the work.

The stope selected for sand filling is admirably situated for this purpose, as it is enclosed by a dyke on the east and south sides, and by a block of ground on the west, which is being worked towards the east. From the main drive on the south side of the dyke a cross-cut extends north, from this a box-hole rise has been put up about 18 ft. entering the bottom of the stope, and on the west side of the dyke there is an intermediate drive leading through to No. 1 shaft (see Fig. 1.).

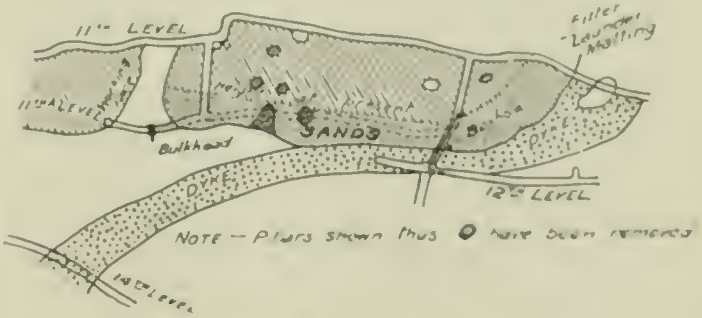


Fig. 1.

A bulkhead made of 3 in. × 9 in. deal, supported by 8 in. × 8 in. upright timbers, was placed across the intermediate drive to the east of the working face, the 3 in. × 9 in. deals being perforated and covered with cocoanut matting on the face next the sand, and thus forming an outlet for drainage on the west side of the stope. It will now be apparent that an enclosed space has been provided with the exception of the box hole entering the stope on the south or bottom side. This box-hole is used as the main outlet for drainage; a dry wall was built across it through which a 6 in. pipe projects into the stope. A filter frame 12 in. square covered with cocoanut matting and extending up the stope above the level of the sand was placed over the 6 in. pipe, thus forming an efficient filter through which all the drainage must percolate before escaping into the level below (Fig. II.). The square filter is made in 12 ft. 6 in. lengths, one end being made 14 in. square for about 2 ft. to fit over the length previously fixed, so that when the sand rises to about 2 ft. from the top of the filter the next length can be placed over it, and by this means the filter is always kept above the level of the sand (Fig. III.). Inside this frame a covered wooden launder is placed to maintain an opening should the filter frame collapse. The drainage from the sand is a small stream of about one-third of a ton of clear water free from cyanide to every ton of sand deposited, and although about 200 tons of current residue is lowered daily, men are still at work in the stope taking out pillars, etc., and suffer no inconvenience from cyanide or any of its products.

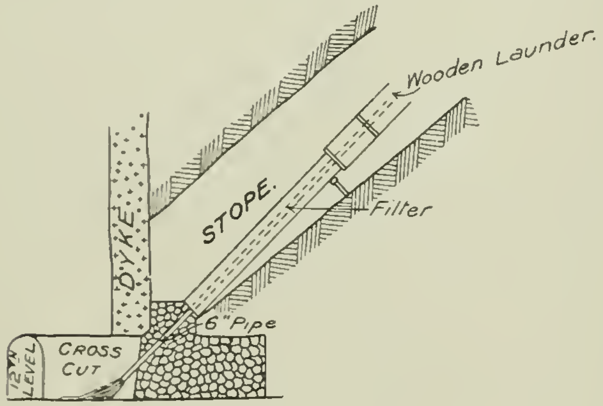


Fig. II.

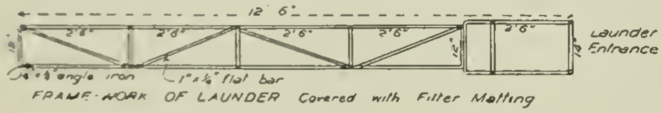


Fig. III.

The sand is conveyed from the bottom of the bore-hole in wooden launders 9 in. wide and 7 in. deep with a grade of 15° on which the pulp flows freely when diluted with a small spray of mine water. The sand forms a compact mass on which men can walk comfortably in from two to four hours after filling has stopped. The cleaner and more freely leachable the sand the better, as slime tends to retard drainage and pockets water between the points where the launders deliver sand.

In conclusion, I should like to point out that there should be several stopes ready for filling when it is proposed that the whole of the current sand residue be sent underground, as the removal of pillars, etc., takes time and when dealing with the daily production of sand from the cyanide works filling must go on without interruption. For a stope less favourably situated for sand filling than that described I would suggest that stulls be made of stout mine poles hitched at top and bottom, lagged over with old pipes and rails and packed with waste rock to a thickness of about 4 ft., the inner face being covered with cocoanut matting. This would afford perfect drainage and the sand being in a dry state would exert very little lateral pressure against the stulls, and there would be no uncontrollable mass of sand and slime saturated with water to become a real source of danger in the mine.

I wish to express my thanks to the General Manager of the Simmer and Jack Proprietary Mines, Ltd., for being allowed to give these details of a new kind of work.

NOTES ON PRECIPITATION.

(*Read at July Meeting, 1910.*)

By F. D. Phillips (Member).

DISCUSSION.

Mr. J. Hayward Johnson (*Member*): The author's remarks suggest, as did Mr. Mather-Smith's paper under the same title (Vol. ix., March, 1909, p. 300), that an unnecessary amount of zinc is in use. This may be so where conditions are ideal, but in many plants such conditions are not obtainable, the flow of solution being irregular. In many cases boxes have been built with a nominal capacity of, say, 300 tons solution per 24 hours, but very often during some period of the 24 hours the rate of flow is equivalent to 400 or 500 tons per 24 hours and then decreases again to under an equivalent of 100 tons, though the total tonnage will not be more than 300 tons. Again, there is the deleterious effect of organic matter, etc , coming into solution which will often cause the first three or four compartments to act as cleaners and no precipitation occurs until after this cleansing has taken place. This is especially noticeable during the dry season, when dams become polluted with compound refuse, etc., to say nothing of the old bug-bear, "stope sweeping." Some fifteen months ago I had trouble with my precipitation boxes, and on making enquiries the mine captain informed me he had been sweeping a stope containing some 12 in. to 18 in. saw-dust, etc. Rubbish of this kind coming to the cyanide plant not only affects precipitation but also the dissolving and extraction of gold, the saw-dust acting as a precipitant in the vats (see "Notes on Precipitating Effects of Substances containing various Forms of Carbon, etc., by Messrs. Clark and Sharwood, Vol. ix., p. 234), and organic matter absorbing the oxygen that is required for dissolving the gold. During the discussion of the paper no mention has been made of any of these bad effects, one of which is to cause the zinc to be covered with an apparently white metallic deposit giving the zinc the appearance of being newly cut or cleaned of all other deposits, whereas on examination it is found to be a coating, and all action ceases and an equivalent of a reversal of current in a Siemens-Halske box takes place and gold is redissolved. Unfortunately I have been unable to have any analysis made of this deposit. I find sal ammoniac dissolved at the head of the boxes to be the best antidote for this disease.

NOTES ON BATTERY PRACTICE.

(*Read at September Meeting, 1910.*)

By A. R. Stacpoole (Member).

DISCUSSION.

Mr. J. Hayward Johnson (*Member*): The author evidently in writing his paper had in view battery practice as carried out in the outside districts and in *some* of the mills on the Rand. In his first paragraph he speaks of the usurpation of the mill managers' responsibilities by the reduction officer, and in his second and third of the extraction by amalgamation

One reason for this usurpation is no doubt due to the fact that too little attention has been paid to the values of screen and pulp tailing assays, doubtless it is very difficult to watch the extraction by amalgamation owing to the difficulty of obtaining a true screen sample, especially where mercury is added, as I believe it still is in many places in the mortar boxes: and again, owing to the plates being close to the boxes it is very difficult to take an accurate sample. If attention is called to the theoretical recovery as per screen and tail sample the excuse always comes forward that the former is not to be relied upon, or "any how the scrape is up." Scrape increase may be due to several things other than catch of gold during period of sampling. More mercury left in amalgam, plates being softer and giving up amalgam more freely, and lastly a greater tonnage having been put through during the period under review. Cyanide men have to work and recover gold in accordance with the results returned by the assayer, and until the clean-up is through and result of bullion assay is returned they are more or less in the dark. Hence they watch their extraction returns, assays of residues and solutions leaving boxes very closely, and all likely sources of loss. The reduction works manager carries this principle out right through the works, and is very careful that nothing escapes from the one department with the idea that it will be caught in the next.

The name of amalgamator, as applied to many men in mills, is a misnomer: they are more like millwrights, and the amalgamation has to take care of itself. If (I am speaking of some batteries and those in outside districts), mill extraction or recovery had to be according to samples taken at head of plates and after amalgamation, more care would be taken that amalgam was not rubbed off by rough handling of tools, etc., around plates, the recovery of black sand would have more attention, and amalgam would not be so noticeable in the launders leading to cyanide works. Speaking of increase in mill tonnage, this is often obtained by increasing the amount of water used, which naturally means a coarser product, a bigger flow over plates, and a less percentage of recovery by amalgamation as well as total extraction.

Mr. F. W. Cindel (*Member*): In criticising Mr. Stacpoole's paper it is not my intention to discuss it in detail nor to defend the mill men, for I believe that if they take a keener interest in cyaniding the time will come when they will benefit by it, not only in a material way but educationally, thereby increasing their knowledge of the process of gold extraction after the pulp leaves the battery.

I can hardly agree with the author when he says a mill man thinks it his first and only duty to crush and put through a big tonnage, paying but little attention to extraction. Personally, I take the opposite view, and have always worked with the intention of getting the highest extraction possible. In fact I take a great pride in my work if my extraction is 1 %, or even ½ % higher than my neighbour's. My first object is to take out by amalgamation all of the gold As we can get only from 74 % to 76 % in the mill we try to recover the rest by cyanide. I wish to impress upon the author that on this particular mine, if we do not get the gold called for, both in the mill and cyanide works, my immediate chief wants to know the reason. Of course we try to crush as much as possible, but extraction should be the principal thing to strive for. As is often the case, the manager orders the screening in the battery to be reduced from, say, 400 mesh to 100 mesh, perhaps to increase the tonnage and surpass the previous month's figures. What is the result? He lowers his working costs slightly, lowers the extraction in the mill and sends a few more ounces of gold to the dump in residues. The reduction plant the capacity of which remains the same, is required to treat a greater or less tonnage. When the screening is lowered in the mill the cyanide manager is suddenly called upon to make room for the extra amount of pulp, consequently some tanks are discharged before they have had their proper treatment with the result as stated above.

The author is making a sweeping remark in the second paragraph of his paper. He condemns mill men wholesale without taking into consideration any of the many little things that affect extraction. His idea of catching black sands is a good one, and is being used in a number of mills to-day (without the overflow pipe). If I remember rightly, it was some fifteen years ago I saw this launder being used, and I am sure that was not the first one invented. In the mill of which I speak, the amalgamator has an 18 in. bucket into which he puts his brooms, brushes, mercury bottle, scrapers, etc., while dressing plates. This he carries about with him from plate to plate. I consider it a good receptacle and not so cumbersome as the author's trolley. If an amalgamator cannot put his brushes, etc., into an 18 in. bucket I fail to see how a cyanide tank would be of any use to him. Of course, if the author's trolley were nicely painted in bright colours it would be pleasing to the eye, and perhaps be an object of cleanliness to the men; if drawn by a horse, as has been suggested, the trolley would be all out of proportion. I might suggest a small white boy be used, but this again would remind one of one's kindergarten days. But why draw the trolley in such a primeval manner as with man or horse when we have advanced methods of motive power, such as electricity or the gas-engine; surely the noise of the latter would not be objectionable in the battery.

The author is quite right when he says care should be used in changing screens, shoes, dies, etc. If he will take the trouble to go along the reef I am sure he will find methods employed in our mills that are quite as efficient as those set forth in his paper. A person not conversant with modern Rand mill practice may be led to believe that the methods employed here are far from up to date, and are carried on in a careless manner.

To know how to change screens, carry brooms, brushes, tools, empty launders and buckets is quite a simple matter, and until the late Gold Law came into force this work was mostly done by Kafirs. In mills where this work is still to be learned I admit they are a long way behind the times. No doubt the information would be of great advantage to learners.

The author says by taking the plates out of the battery, free gold would concentrate in the launders. I should like to ask him what would become of this concentrated gold? Would it keep on concentrating and gradually fill up the launder, or would it eventually find its way to the tube mill plates? For many reasons too numerous to mention the plates should not be next to the mortar box and it is only those people who still adhere to obsolete ideas who insist upon it.

Referring to extraction, the author strikes the right note in the last paragraph of his paper where he advises mill men to run their mills as though there were no cyanide plants behind them.

Mr. C. L. Dewar (*Associate*): I think the author is rather hard on the mill man, whom he accuses of concerning himself more about tonnage than extraction. He says, the first duty of the mill man is to see that no free gold, amalgam, or mercury goes over the end of the plate. Speaking generally—Is the quantity escaping unduly high? I do not think it is, otherwise the mill man would soon be made aware of the fact, and we would probably hear more from him about extraction and less about tonnage. It might be pertinent to remark that until recently, those who controlled our industry were, or seemed to be, more concerned about cost per ton than they were about profit per ton, with the result that the mill manager was often called on to do his utmost to get a big tonnage. I think this explains why the mill man has so much to say about duty per stamp. Tonnage is to the mill man what footage is to the shaft sinker.

I do not think any mill manager would complain if he had an amalgam room built according to the author's idea, but if the ore bins were full the three or five stamp battery would soon have more to do than crush the plate sand and by-products from the amalgam barrel. Does the author mean to allow the black sand to pass into the main tailings launder after treatment in the amalgam barrel?

I do not think the mill men on the Rand can learn much from the author's paper. In reading it before this Society, I venture to say that he was actuated chiefly with the desire to make the mill men who belong to this Society speak out and make themselves heard, and (as our Chairman remarked) join in our discussions. If the paper succeeds in doing this, it will have served a useful purpose.

Mr. Andrew King (*Associate*): When writing his paper Mr. Stacpoole seems to have forgotten that the mill manager has nothing to do with the amount of ore delivered to him, and this to a very large extent governs the rate of crushing. Screens with larger apertures have to be used in order that the increased tonnages now being dealt with on many mines may be crushed. Under these conditions, which are vastly different from those of a few years ago, it cannot be expected that the same extraction should be obtained on the battery plates as formerly. Screening with 200 holes and less per square inch is in regular use in many mills, and with such screening it is impossible to prevent a certain amount of amalgam being carried into the tube mill circuit. Even with a much finer screen than that mentioned above, some amalgam, although it may be small in quantity, will escape from the battery in spite of the good condition of the plates and use of amalgam traps. If, however, the tube mill circuit is properly arranged and is provided with the requisite number of cone classifiers with diaphragms, there is no chance of any amalgam overflowing with the cyanide pulp. Very little amalgam is found in the launders carrying the pulp away from the shaking plates, so that the amalgam found in the other launders in the tube mill circuit must come from the battery plates. This is, of course, due to the much finer condition of the pulp leaving the tube mills and the slower rate of flow of that pulp over the shaking tables. As well as the size of the screen aperture and the condition of the plates there is another factor on which the extraction to a large extent depends. Before condemning the results obtained at one mine, or comparing them with those obtained elsewhere, one must take into consideration whether the ore is from an outcrop, from a "deep" or "deep-deep" mine, from the main reef or from the "bastard reef."

By having a reduction works manager the actual responsibility of the mill manager may be somewhat lessened, but by the great changes which have taken place in battery practice his duties have been greatly increased. He now has to crush and amalgamate a much larger tonnage, and at the same time crush it much finer, producing a larger percentage of slime before sending the final product to the cyanide plant. He deserves every credit for showing himself able to cope with these new conditions more or less forced on him.

The author, while urging mill men to take more part in discussions on current battery practice and in referring specially to the removal of the plates from the battery, himself only raises one point against such a scheme. He says "free gold will be sure to concentrate in the cement launders and in every corner it meets on its way to the amalgam table." If the launder is of "cement" it will be perfectly smooth and if properly constructed will have no corners. The author also seems to have forgotten that, as the result of several independent trials which have been carried out, it is now generally recognised that in increasing an existing plant or in building a new one it is more economical to decrease the ratio of stamps to tube mills, and that by crushing coarse in the battery and by having the proper number of tube mills to reduce the pulp to the requisite fineness before sending it to the cyanide plant, more − 90 mesh product (− ·006 in. aperture) is produced per horse power consumed.

I hope the author is wrong in saying that the method for collecting black sand, to which he so strongly and rightly objects is in use in "many" mills. While admitting that there is room for greater care by the average amalgamator yet there seems no necessity for the cumbersome apparatus the author advises. I should like to know what objection he has to the following method. Instead of shutting off the water at the same time as hanging up the stamps allow it to run until the plate is clear of pulp. This will take only a few seconds and will not wash away the black sand previously adhering to the plate. Now brush the black sand up to the top of the plate and put it into the amalgam bucket. After dressing do the same again to collect the loosened sand. I agree with the author that brushes should not be carried around in the hand. If they are properly dumped on the plate then all the amalgam and black sand will be removed and can be put into the bucket. Surely there is no objection to having another bucket for brushes and tools.

If the amount of black sand from the amalgam room is large, instead of having a battery of light stamps for recrushing it and sending the still very rich tailing from the plate with the main launder, it is much better to recrush it in a small tube mill, pass the outflow over a small shaking plate and cyanide the tailing separately in a Brown air-lift vat. If the amount of ore being fed to the stamps has been properly regulated behorehand there will be comparatively little sand in the box when it is opened, and very little of this need fall on the plate when the screen is raised. A portable platform made of wood with a covering of sheet iron (commonly called a shoe and die board) should be placed in front when doing any heavy work in the box. If this is properly constructed it will fit close against the mortar box so that very little gravel or coarse sand can get on to the plate. There is no excuse for tools coming in contact with the plate. For these reasons the rubber aprons suggested by the author seem unnecessary. Of course it is advisable to have a collecting trough at the bottom of the plate to catch whatever gravel or coarse sand does come over. This need not be sent to the amalgam room. Return all such material to one box, the pulp from which should not be sampled if there is any fear of it containing amalgam.

The meeting then closed.

Obituary.

The deaths of the following members is recorded with much regret :—

Mr. **Charlie Edgar Rusden**, who died on September 10th, 1910, from the effects of having, by accident, fallen down the shaft of the Poverty Mine, Essexvale, S. Rhodesia. The deceased was trained at the Redruth School of Mines, and came to Johannesburg in 1897, being appointed assayer to the Geldenhuis Estate and G. M. Co , Ltd. This position he retained until 1905, when he was promoted to the cyanide managership of the same company. In the early part of 1909 Mr. Rusden resigned his connection with the Geldenhuis Estate in order to go to Rhodesia, where he ultimately acquired one-half interest in the mine where he met his death at the early age of 33 years.

During the late war Mr. Rusden saw a great deal of active service, being attached to the South African Light Horse, and for which he received the Queen's and King's medals and three bars.

The deceased was elected a member of the Society in October, 1904, and leaves a widow to mourn his loss.

Mr. **Frank Ogle** died on Saturday evening, October 8, at the Simmer and Jack Hospital after only a week's illness. The cause of death, at the age of 47 years, was enteric fever complicated by pneumonia, the fatal results of which devoted nursing and medical skill failed to avert.

Mr. F. B. Ogle came to Natal from England in the eighties to take up farming pursuits, but moved to Barberton about 1889, where he engaged in mining and metallurgical work at the Big Buffalo, Sheba and Woodbine mines until the outbreak of war in 1899, when he went to England. Returning to the Transvaal about six years ago, he shortly afterwards entered the service of the Simmer and Jack Proprietary Mines. Upon the starting of the Simmer Deep-Jupiter reduction plant two years ago he became the chief smelter upon this property, which responsible position he held at the time of his death.

During his long record in the service of South African mining companies he became universally respected as a man of the most scrupulous integrity and devotion to duty, whilst his kindly unassuming disposition endeared him to those with whom he was associated, as was evidenced by the large number of mourners attending his funeral at the Germiston Cemetery on Sunday afternoon as a last tribute of respect.

He joined this Society in July, 1898, as an Associate, and was elected a Member in January, 1907. He was likewise a Fellow of the Royal Colonial Institute, and belonged to the Sons of England, representatives of which last body acted as pall-bearers at the funeral.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Portland Cement Calculations.—"Portland cement is a finely ground slag containing definite proportions of silica SiO_2, alumina Al_2O_3, iron oxide Fe_2O_3, lime CaO, magnesia MgO, potash K_2O, soda Na_2O, and small quantities of other ingredients as impurities.

Various proportions of limestone and clay are finely ground and fused to a clinker, but the mixture must be such that the lime formed shall have a definite ratio to the silica, alumina, and ferric oxide in the slag, or if magnesia be present in small proportions the same conditions must apply as for lime. The slag should have the following ratios for lime.

$$\frac{3CaO}{SiO_2} = \frac{3 \times 40 + 3 \times 16}{28 + 32} = 2{\cdot}8$$

$$\frac{2CaO}{Al_2O_3} = \frac{2 \times 40 + 2 \times 16}{2 \times 27 + 3 \times 16} = 1{\cdot}1$$

$$\frac{2CaO}{Fe_2O_3} = \frac{2 \times 40 + 16}{2 \times 56 + 3 \times 16} = {\cdot}7$$

The results show that 2·8 times as much lime as silica, 1·1 times as much lime as alumina, and ·7 as much lime as ferric oxide must be used to form a proper cement slag. The figures used are the atomic weights of the various elements.

Magnesia should not be present in the mixture beyond 5%, and what is present should be proportioned to form the following combinations :—

$$\frac{3MgO}{SiO_2} = \frac{3 \times 24 + 3 \times 16}{28 + 32} = 2$$

$$\frac{2MgO}{Al_2O_3} = \frac{2 \times 24 + 2 \times 16}{2 \times 27 + 3 \times 16} = {\cdot}784$$

$$\frac{2MgO}{Fe_2O_3} = \frac{2 \times 24 + 2 \times 16}{2 \times 56 + 3 \times 16} = {\cdot}5$$

The results show that there must be 2 parts of magnesia to 1 of silica ; ·784 part to 1 of alumina ; and ·5 part to 1 of ferric oxide. It is usual, however, to reckon magnesia in terms of lime, thus, $\frac{CaO}{MgO} = \frac{56}{40} = 1{\cdot}4$ parts of lime are equivalent to 1 part of magnesia.

The lime is obtained from the limestone used and contains some impurities, but these are ascertained by analysis. The clay is also analysed so that the product to be derived from the fused mixture can be accurately foretold. Special rotary furnaces are used at present in preference to the old stack furnaces, and these are fired by gas or pulverised coal. The clayey material which is practically aluminium silicate, with small percentages of impurities is said to commence to fuse at 2,912° F. Pure limestone does not fuse but is converted into lime and then uniting with the materials in the clay forms the cement clinker. The lumps of clinker are separated from the unfused material and pulverised to form Portland cement. It has been ascertained that the finer the clinker is pulverised the better will be the cement since in extremely fine condition the cement forms a silicate with the sand used in practical construction work. The majority of limestones contain some clay and clays often contain limestone, and in making a mixture of raw materials previous to slagging these must be balanced. Magnesia may replace the lime up to 5% without bad results, and ferric oxide may replace alumina. A certain amount of iron acts as a flux, lowers the fusion point of the mixture, and promotes the combination of lime and silica. The replacement of practically all the alumina by iron has been advocated for cement that is to be used in sea water. Limestone containing approximately 75% calcium carbonate $CaCO_3$ and 20% of silica, alumina, and iron oxide is termed natural cement rock because it contains as impurities the quantity of clayey material that must be added to pure limestone in making cement. It is seldom that the right proportions are found as impurities in limestone to make a good natural cement, and to produce the right porportions clay or limestone must be added.

Other things being equal, raw material approaching natural cement rock can be a little more cheaply prepared than a mixture of pure limestone and clay.

Assuming a clay and a limestone to contain the following ingredients, it is desired to calculate a cement mixture:—

	Clay.	Limestone
Silica SiO_2	60·17	·38
Alumina Al_2O_3	22·65	·18
Ferric oxide Fe_2O_3	4·66	
Calcium oxide CaO	·31	55·62
Magnesia MgO	1·85	
Alkalies K_2O and Na_2O ...		
Loss on ignition	6·35	43·50
Water H_2O below 220° F. ...		·17

Clay.—

60·17 × 2·8 = 168·47 parts CaO for 100 parts SiO_2
22·65 × 1·1 = 24·92 parts CaO for 100 parts Al_2O_3
4·66 × 0·7 = 3·26 parts CaO for 100 parts Fe_2O_3

196·65 parts CaO for 100 parts clay
·31 + (1·85 × 1·4) = 2·90 parts CaO in 100 parts clay

193·75 parts CaO to every 100 parts clay

Limestone.—

$$\left.\begin{array}{l}\cdot38\times2\cdot8=1\cdot064\\ \cdot18\times1\cdot1=\cdot198\end{array}\right\}1\cdot262$$ parts CaO in 100 parts limestone which is not available, hence 55·62 − 1·26 = 54·36 parts calcium oxide available in 100 parts of limestone.

Cement Mixture.—

$\frac{193\cdot75}{54\cdot36}$ = 3·54 parts limestone required for 1 part clay, or 354 lb. of limestone for every 100 lb. clay. The quantity of volatile matter in the limestone is 3·54 × (43·5 + ·17) = 154·59; the volatile matter in the clay 6·35; hence the cement when burned will weigh 454 − 154·59 = 299·41 lb.

The percentage of lime should be from 60 to 63·5% in good Portland cement, while in the above mixture it is 65·9. The ratio for the silica-alumina must be within 3 : 1 to 4 : 1 or 2·5 : 1 to 3·5 : 1. A richer clay will have a lower burning temperature and carry less lime, which makes a poorer cement."—*Mines and Minerals*, Aug., 1910, p. 25. (A. R.)

Parts of Material by Weight.

		Clay.	Limestone.	Burned
Total	...	100·00	354·00	299·41
Silica ...	...	60·17	1·27	20·50
Alumina	...	22·65	·60	7·76
Ferric oxide	...	4·66		1·55
Lime ...	...	·31	196·89	65·90
Magnesia	...	1·85		·61
Volatile	...	6·35	155·99	

METALLURGY.

Method for Testing Slime.—"The vanning plaque or shovel is ordinarily used to make a test upon the tailing from vanners. Where the tailing consists principally of sand, the test can be made rapidly and satisfactorily; but with slime more care and time are required. For this purpose I have found it convenient to use the small laboratory 'centrifuge' in place of the plaque, and the results are sufficiently satisfactory to warrant my bringing the method before your readers. The 'centrifuge' consists of two cups, fastened to the extremities of a horizontal bar. The bar is fastened at its centre to a vertical shaft. The slime to be tested is mixed with water and placed in two tubes, resembling test-tubes; these are placed in the cups, and the vertical shaft is rotated at from 2,000 to 3,000 r.p.m. for several minutes. The result is to force all the concentrate and any coarse sand to the end of the tube, where they may be readily seen and examined. By using a graduated tube and calibrating the same with known weights of pyrite (or other heavy minerals), approximate quantitative results may be obtained. The presence of much fine sand or of any coarse sand renders the results unsatisfactory. The distribution of the materials in the tubes is illustrated by the accompanying sketches."—G. J. Young.—*The Mining Magazine*, August, 1910. (J. E. T.)

Fig 1

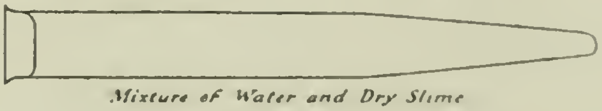


Fig. 2

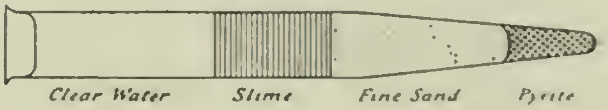


Fig. 3

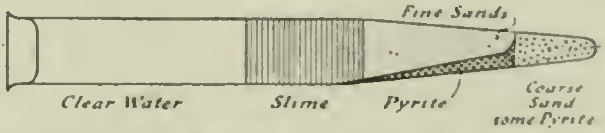


Fig 4

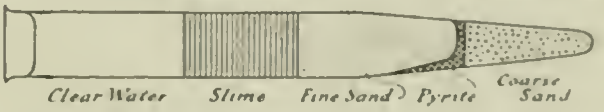
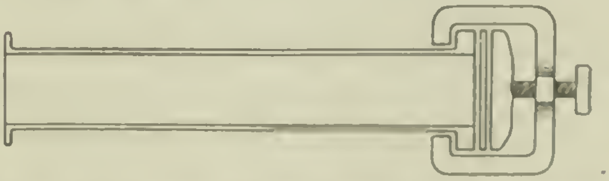


Fig 5



THE EFFICIENCIES OF CRUSHERS.—"When making comparisons of the efficiencies of different crushing machines it is desirable to be able to estimate the work actually done in crushing the ore from a given size of feed to a given size of product, the screen analysis of both feed and product being determined. The method proposed for such calculations is based upon the principle that if a quantity Q of material that will just pass through a screen of m meshes to the inch, then the work done is proportional to $Q(n-m)$. This statement is based on experiments and the following reasons and assumptions:

Suppose a quantity Q of material is made up of particles of an average diameter x, then number of particles is proportional to quantity divided by the volume of each particle; that is to $\frac{Q}{x^3}$. Now, let the particles be crushed to some smaller diameter y. If one of the original particles is a cube, whose side is x, and this is divided up into cubes each of side y, the division may be effected by shearing along three sets of parallel planes at right angles as shown in Fig. 1. The number of such shearing planes parallel to one side is $\frac{x}{y}-1$, and these are each of area x^2. In order to divide up the original cube, then, along one set of these parallel planes the work required to be done is proportional to $\left(\frac{x}{y}-1\right)x^2$, and since the shearing has to be effected along three sets of planes, the total work done is measured by $3\left(\frac{x}{y}-1\right)x^2$. But there are $\frac{Q}{x^3}$ particles, and therefore the total work done in reducing a quantity Q from average diameter x to diameter y is measured by $\frac{Q}{x^3}\times 3\left(\frac{x}{y}-1\right)x^2$, or leaving out the constant, $Q\left(\frac{1}{y}-\frac{1}{x}\right)$. According to this, if we now further crush down to diameter z the additional work done is $Q\left(\frac{1}{z}-\frac{1}{y}\right)$, and the total work done in crushing from diameter x to diameter $z = Q\left(\frac{1}{y}-\frac{1}{x}\right)+Q\left(\frac{1}{z}-\frac{1}{y}\right)$, which $= Q\left(\frac{1}{z}-\frac{1}{x}\right)$, the same as if the crushing were all done in one operation.

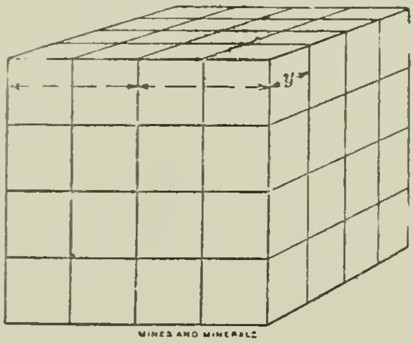


FIG. 1.

To make the computation, then, it is necessary to determine the reciprocal of the diameter of the particle. But this is very nearly double the number of meshes per linear inch of the screen through which the particle will just pass, as is seen from Table 1 for the I.M.M. standard laboratory screens.

It consequently follows that the crushing of a quantity Q of ore, consisting of particles of uniform size that will just pass through a screen of m meshes per linear inch down to another uniform size, such that they will just pass through a screen of n meshes per linear inch, requires an amount of work proportional to $Q(n-m)$.

If, then, we have material fed into a machine consisting of: a per cent. of material just passing through p meshes to the inch; b per cent. or material just passing through q meshes to the inch; c per cent. of material just passing through r meshes to the inch; and this be ground down to a product consisting of: d per cent. of material just passing through s meshes to the inch; e per cent. of material just passing through t meshes to the inch, the work done in grinding may be found by first of all computing the work required to reduce the feed down to some very small uniform size of, say, m meshes to the inch, next computing the work required to reduce the product of the machine down to the same size of m mesh, and then subtracting the two. That is to say, the total work done in the machine is proportional to

$$a(m-p)+b(m-q)+c(m-r)-d(m-s)-c(m-t)$$
$$=d\,s+c\,t-a\,p-b\,q-c\,r$$

because $a+b+c=d+e=100$

TABLE 1.

Number of Meshes per linear inch.	Diameter of Aperture inch.	Reciprocal of Diameter.
5	·1000	10·00
8	·0620	16·13
10	·0500	20·00
12	·0416	24·04
16	·0312	32·05
20	·0250	40·00
25	·0200	50·00
30	·0166	60·24
35	·0142	70·42
40	·0125	80·00
50	·0100	100·00
60	·0083	120·48
70	·0071	140·84
80	·0062	161·29
100	·0050	200·00
150	·0033	300·00
200	·0025	400·00

TABLE 2.

	Machine A. Output. 20 tons per day.		Machine B. Output. 17 tons per day.	
	Feed	Product	Feed	Product
	%			
Through 20 and on 40	10·3		·5	
Through 40 and on 60	18·7		5·8	
Through 60 and on 80	23·5		29·6	
Through 80 and on 100	12·2	4·3	31·3	
Through 100 and on 120	18·6	19·7	14·7	6·2
Through 120 and on 140	10·4	28·6	8·2	12·5
Through 140 and on 160	4·8	21·7	7·9	47·2
Through 160	1·5	25·7	2·0	34·1

This gives us then a very simple rule for finding the work done in a grinding machine, which we may best describe by a numerical example. Suppose that we wish to compare the efficiencies of the grinding effected by two machines, the feed and resultant products of which grade as in Table 3.

TABLE 3.

Multiplier	Machine A. Result of Multiplying Percentages by Multiplier.		Machine B. Result of Multiplying Percentages by Multiplier.	
	Feed	Product	Feed	Product
30	309		15	
50	935		290	
70	1,645		2,072	
90	1,098	387	2,817	
110	2,046	2,167	1,617	682
130	1,352	3,718	1,066	1,625
150	720	3,255	1,185	7,080
180	240	4,626	360	6,138
	8,345	14,153	9,422	15,525
		8,345		9,422
Units of grinding per ton ...		5,808	Units of grinding per ton	6,103

We have then to multiply each percentage by the number of meshes to which it corresponds. Thus, the material passing through 20 mesh, and caught on 40, will be considered to correspond to an average size of 30 mesh, and the percentage of this class of material is accordingly multiplied by 30. Regarding the material that passes through 160 mesh as equivalent to an average size of 180 mesh, we may arrange the computation as in Table 3.

Thus the total work done in crushing by machine A is proportional to 5,808 × 20 = 116,160, and that by machine B to 6,103 × 17 = 103,751, and the crushing efficiencies of the two machines are in the proportion of 116 : 104, nearly.

The theory underlying this method is in general accordance with the tests of Von Reytt, quoted in Richards' 'Ore Dressing,' Vol. i., page 305." – Prof. R. W. CHAPMAN.—*Australasian Inst. of Mining Engineers, Mines and Minerals*, Feb., 1910, p. 413. (A. R.)

CYANIDATION OF SILVER ORES.—"The application of cyanidation to silver ores is most largely practiced in Mexico, where the system employed is generally well known. In striking contrast to the usual method is the treatment used at the Zambona mine in the Alamos district of Sonora. The unusual feature is in the extreme rapidity of the solution of part of the silver, and the peculiar advantage of concentrating the ore prior to cyaniding. The ore is an orthoclase felspar, brecciated and filled with calcite. The mineral is argentite, distributed throughout the gangue in all sizes down to minute particles. This mineral is easily concentrated on tables, so that 50% of the silver content of the ore can be removed in this way. The tailing is then cyanided. A solution carrying 0·075% free KCN is mixed with the ore in the ratio of 6 : 1, the pulp being treated in Dorr classifiers. The sand goes to tube mills, the discharge from which is returned to the classifiers which in turn discharge their overflow to Dorr slime thickeners. The underflow is stored for delivery to filter presses as needed. In this manner the silver is extracted from the ore by merely concentrating and grinding in solution without agitation of any kind. The silver is reduced from 16 oz. to 1·3 oz. per ton.

As illustrating the sensiblity of the ore to cyanide treatment the following table is given showing the reduced silver content at various stages of the treatment:

Ore after concentration	...	8·30	Silver, oz.
After mixing with solution	...	4·48	,,
Tube mill heads ...	...	5·52	,,
Tube mill tailing ...	...	5·28	,,
Discharge from Dorr thickener		2·72	,,
After standing 6 hours	...	2·28	,,
After a 6-hour agitation	...	2·18	,,
After a 9-hour agitation	...	1·96	,,
After an 18-hour agitation	...	1·82	,,

For a short time the mill was operated without concentration and by cyanidation only. It was found necessary to use a solution carrying 0·15% free KCN and to supplement the other treatment by a 30-hour treatment in Pachuca tanks." – B. B. KNIFFEN.—*Mining and Scientific Press*, Feb. 26; *Metallurgical and Chemical Engineering*, viii., 4, April, 1910, p. 204. (J. A. W.)

SLIME TREATMENT.—"During the last 18 months a series of interesting experiments have been made by the technical staff of the Great Fingall mine at Day Dawn, from which many valuable data have been obtained. When the idea of modernising the slime treatment at the mine was first mooted, the writer had a small agitator fitted up, in which the slime (cyanide-solution pulp) was agitated and re-circulated by means of low-pressure compressed air, in the presence of lime. Experiments conducted on these lines showed excellent extractions of the gold. On the erection of the present vacuum plant, the treatment of the slime—which was carried out as nearly as possible to the experimental method—presented numerous initial difficulties, and it was apparent, during the process of overcoming same, that many points required investigating. To this end, a small experimental agitator was erected, and many experiments were made. The technical work connected therewith was entrusted to Mr. W. B. Blyth, assisted by Mr. W. B. Chomley, the chemist and cyanide plant foreman respectively.

From laboratory experiments, Mr. Blyth found that good extractions could be obtained in an air-tight vessel with the formation of little or no ferrocyanide, whereas the slime that was treated in an open vessel showed poor extractions: ferrocyanide was formed in large quantities, and a heavy consumption of cyanide resulted.

This experiment was next made in the model agitator. But as it was found that the agitator could not be made sufficiently air-tight to produce a similar result, further work on this basis was discontinued, and experiments were undertaken with a view to effecting the complete oxidation of the ferrous salts by agitation with air in the presence of lime.

It was proved that, in the ore treatment, it is essential to prevent the oxidation of the comminuted iron caused by abrasion in the process of crushing the ore, and the experiments demonstrate that this is effectively accomplished by crushing in alkaline solutions. In the case of slime that has been exposed to the action of the atmosphere, it is necessary to decompose the ferrous salt completely by aeration in the presence of lime.

These data were followed up in practice, and our metallurgical operations have been based on them, with the result that the difficulties hitherto encountered have been entirely removed. From the data shown it is apparent that lime plays quite an important part in preventing the solution of cyanide-forming salts in the one case, and of their decomposition in the other instance.

Dr. W. H. Walker states (*Times Engineering Supplement*, March 3, 1909) that iron will not rust in a solution of soda ash, lime water, or soda phosphate. This appears to be due to the fact that iron will not rust readily except in the presence of carbonic acid; lime-water precipitates the latter, and thereby prevents the solution of the iron. Heyn and Burr state (*Electro-Chemical and Metallurgical Industry*, January, 1909) that carbonic acid acts only as any other acid in dissolving iron with the evolution of hydrogen. The presence of hydrogen would have a decidedly reducing effect on the iron salts in solution, and would, on the addition of sodium or potassium cyanide, tend to produce ferrocyanide. In our slime practice we look upon the presence of an appreciable amount of ferrocyanide in solution as being indicative of high residue values. Although ferrocyanide is itself a solvent of gold, its presence indicates insufficient oxidation, and a probable excess of ferrous hydrate, which would act as a precipitant of the gold. This is explained by the fact that when metallic iron particles are oxidised in a neutral solution, to the soluble ferrous state, the addition of lime precipitates ferrous hydrate. Ferrous hydrate, however, takes a considerable time to become oxidised to the ferric state under the conditions of cyanide practice, and it is, therefore, reasonable to assume that this explanation accounts for the indication mentioned above.

II.

Several improved methods of agitation and aeration have been recently introduced, which are a considerable improvement on the ordinary method of agitation by stirring.

The system introduced by the writer and employed at the Great Fingall mine, consists—in addition to agitation by stirring—of re-circulation of the pulp through a tube 2ft. in diameter, by means of low-pressure air on the air-lift principle. The tube is connected to the ordinary agitation vat. In addition to the absorption of air by the direct contact of low-pressure air with the pulp there is also a certain amount of aeration caused by the stream of pulp falling from the delivery launder of the re-circulator into the vat. A series of trials extending over a period of several months showed that this method produced an increased extraction of gold ranging from 5% to 8% more than that obtained by the ordinary method. Another interesting and novel method of aeration, introduced by Messrs. Wright and Jaentsch of Kalgoorlie is that of taking advantage of the centrifugal force set up in the process of agitation. This is effected by means of four or more pipes of about 8 in. in diameter, fitted to the ordinary agitator-gear, and set at an angle of approximately 45°. In the process of agitation, the force set up by the rotary action of the agitator-gear to which the pipes are fixed, causes the pulp to flow through the pipes and in falling from the outlets of pipes through the atmosphere, it is brought into direct contact with the air. This, in addition to increasing the agitation, facilitates the aeration of the pulp. In order to compare the efficiency of these two systems, a number of trials were made by Mr. C. C. Freeman, metallurgist at the Great Fingall mine. The results obtained from approximately 1,500 tons of slime, showed that the Fingall system yielded an extraction of 82·3% and the Wright-Jaentsch method 80·9% of the gold contents of the slime, which averaged 9·4s. per short ton. The increased horse-power over that absorbed by ordinary agitation was in the former 4·6 and the latter 1·05. Allowing for the cost of power at 2s. 6d. per horse-power day, the difference in favour of the Wright-Jaentsch system is equal to 8·875s. per charge of 75 tons. Against this however, the Fingall system shows an increased extraction of gold of 1·4%, which, on the value of 9·4s. per ton, is equal to 0·1316s. per ton, or 9·75s. per charge of 75 tons. Consequently a comparison of power and extraction shows a difference in favour of the Fingall system of only 10·5d. per charge which can be regarded as negligible and the two systems may therefore be considered equal in efficiency and economy."—*Journal of Chamber of Mines of Western Australia*, June 30, 1910, p. 169. (H. A. W.)

MINING.

PER SQUARE FATHOM.—"I have followed American mining practice for several years, and even wandered as far from the Brown Palace hotel as Central City, where the sturdy Cornishmen still measures ore by the 'cord.' In other districts I have been accustomed to hear ore expressed in the unit of the ton of 2,000 lb. There has never been, so far as I am aware, any serious objection to the use of this unit in the metal-mining operations of the Western States, whether the ore comes from vertical veins or from inclined and flat deposits of greatly varying width.

It is true there is no deposit of ore in the United States exactly resembling the auriferous 'banket' of the Rand. But there are deposits where the mining and stoping methods pursued are comparable. In such districts as Leadville, Colorado; Joplin, Missouri; Mercur, Utah; the Cœur d'Alene, there have never been any especial difficulties in computing the cost of mining and treatment on the basis of the ton of 2,000 lb. weight. Yet in any of these bedded deposits, the variations of stoping width are as great as on the Witwatersrand.

Referring to the suggestion of Mr. H. F. Marriott, on which you comment in your May issue, that the value and cost of mining the ore of the Rand shall be computed 'per square fathom,' I should like to ask, as a layman who has never seen the Rand, how this basis of computation is going to simplify the already somewhat complicated Rand cost-sheets?

What is a fathom? Except among sea-faring men, its use is confined to a few mining districts in England. Originally the word 'faden' was imported to Cornwall from Germany, and 'fathom' was the nearest the local population could attain toward pronouncing it. The original length was probably about seven feet. It was gradually adopted by the Land's Enders to denote the depth to which the average man can dive below the surface of the sea. Thus gradually it has come to mean in England six feet, and is in universal use as a nautical measure. There is no reason why the word should be forced into metal-mining practice in the Australian and South African gold-fields, in which the entire civilized world takes an interest. I remember to have read, two years ago, an interesting and carefully prepared paper on the deep leads of Australia. I read parts of it twice, but at the end it meant nothing to me. The value of the paper was vitiated by the writer's repeated use of 'fathom,' 'square fathom,' etc. To my mind we may as well refer to the 'port' and 'starboard' walls of a vein.

There are numerous French, German and American shareholders in the gold mines of the Rand. Is it fair to these gentlemen, who must take an extremely live interest in the reports of the various companies, to introduce at this date an entirely new system of valuation, essentially British and limited in acceptance? To hope for the general adoption of the metric ton is impossible. Failing this, the ton of 2 000 lb. comes the nearest to an universal weight possible for use in American or English mining communities.

To me it does not seem an impossible thing for a mine manager to figure out how much a ton of ore costs to mine and treat, however large the operations, and however complicated the system of accounting. The same may be said of its assay-value and the amount recoverable by the system of treatment in use. Without appearing to criticise too harshly from a distance, it nevertheless appears to me that the type of mining science which seeks to impose on the most productive gold industry of the world a system of accounting entirely at variance with standard practice as recognised elsewhere is on a par, for example, with that pseudo-science of certain modern geologists who have invented a system of rock nomenclature intelligible only to themselves.—C.W.P." Honolulu, June 6.

"Although I have never been on the Rand yet as a mining engineer and as a shareholder in some of the Rand mines I think I am entitled to record my protest against the adoption of Mr. H. F. Marriott's 'square fathom' system of quarterly returns. My knowledge of the subject is entirely derived from a pamphlet 'A New Policy in Mining,' containing extracts from Mr. Marriott's speech on the proposed system of returns and newspaper comments.

I find in the Century Dictionary:

'*Square fathom*, in mining, 36 sq. ft. of the vein, measured on one of the walls, and including its whole thickness. The available amount of ore in a mine worked on a regular fissure-vein is usually reckoned by the square fathom.'

This definition is a good illustration of the fallibility of dictionaries, for with an acquaintance of many of the mining districts of the world and an extensive perusal of mining reports I only know of two districts where the 'fathom'—whether it be square or cubic I do not know—is used: Lake Superior and Cornwall. And in my mind there is no question that the above definition is wrong for ore in fissure-veins is *not* usually reckoned in square fathoms, but in tons.

It ought to be axiomatic that any change made in mining reports should be in the direction of uniformity and not of diversity, and if Mr. Marriott had suggested the use of the square metre and gramme for making his returns, I should have felt a certain amount of sympathy with him. I am quite sure he will hear a strong protest from the French and German shareholders when he attempts to inflict this ancient measure upon them. And in my own case I find it so troublesome to contend with the different sorts of tons and ounces in English-speaking countries and with cargas, arrobas, marcs, zolotniks per 100 poods and dolis per cubic sagene in foreign lands that I cannot welcome the introduction of this obsolete unit of measurement—the fathom—into modern mining. Consequently I sincerely hope that many other engineers will join me in protesting against this anachronism.

It is consoling to see that Mr. Marriott does not have implicit faith in his new system, for I notice that he proposes to make his quarterly returns at present both on the basis of tons milled and the square fathom. Thus the reader of these reports will have the blessed privilege of ignoring the square fathom returns if he so desires. Mr. Marriott also states (p. 1): 'though in discussing these principles I leave out mention of the word 'ton' as far as it is possible to do so, it is of course understood that for the internal operations of the mine under the control of the manager the measurement by this method will always be used to a certain extent in keeping track of his operations.' But in spite of this use of the ton-milled costs in the mine and in his quarterly returns, Mr. Marriott calmly announces (p. 10): 'Working costs per ton milled, in so far as they concern anyone beyond the local interest of the manager in his details of working, are as dead as the proverbial door-nail.' This is a statement that seems to me strangely at variance with the facts, for, as far as I know, the use of the ton as a basis for estimating costs is still world-wide. Moreover, as regards the Rand, I find that in the statement of the results for the six months ending June 30, 1910 (published in the *Financial Times* for July 23, 1910) nothing is used in estimating costs except tons of ore milled. Hence I fail to see what authority Mr. Marriott has for saying that working-costs per ton milled are dead.

I have not seen a clear account of what Mr. Marriott means by his square fathom system, but I gather from his various remarks that he intends to divide the ore reserves into blocks whose gold content can be accurately estimated; this, I take it, is what he means by his cryptic utterance (p. 13) 'the ore reserves are, or should be, blocked out into solid units of such a size that their exterior information will give a fairly accurate estimation of their interior contents.' Mr. Marriott further states that against each of these blocks, on a stoping plan the value of the gold contents as estimated for the total value of the reserves, should be written in pounds sterling and the square fathomage noted alongside.' A separate ledger account is to be kept with each block, and all expenses incurred in extracting the gold—including the due proportion of outside expenses—are to be charged up against the gold obtained. The balance remaining to the credit of this block, after the ore has all been extracted and treated, will be the profit available for dividends. Mr. Marriott gives a much fuller statement, but I think the above fairly represents his views. I fail to see any reason why the square fathomage should be used to get at the value of his blocks. To determine their values the ton unit *must* be used, and as this is the case why drag in the square fathom?

Mr. Marriott says that the square fathom has the advantage of giving the manager 'the fullest scope possible—scope far beyond that which he has enjoyed in the past—to show what he is worth as a responsible individual.' Mr. Marriott does not explain exactly what he means by this phrase, and I decline to guess at his meaning except to assume that it has something to do with working the blocks into which he has divided the ore reserves. Now the extraction of gold from these blocks of ore combines a number of physical operations in the mine and on the surface and I cannot see why there is any occult power in writing the words "square fathomage" on a map that enables a manager to make more profit than if he wrote the words "tons of ore.'

A second advantage mentioned is (p. 10) that the 'value of gold contents of all your' (Mr. Marriott is speaking to mining managers in Johannesburg) 'payable ore reserves is an item on which greater accuracy must be insisted upon than has been the practice in the past.'

It is plain enough that the accurate work in the past had nothing to do with the system by which the results of such work were tabulated, and that the results of accurate sampling and assaying can be as well expressed in values per ton as in £ s d per square fathom, and hence the square fathom system, from this point of view, has no advantage over the tonnage system.

Still another advantage claimed for the square fathom system is that comparison can be more readily made. Mr. Marriott's words are: (p. 24) 'the ore reserves are recorded in terms of the area within which they are included. That area has a distinct relation to the remaining claim area of the Company's property, and a direct comparison with this remaining ground can be made on the same basis. The profit now being obtained and to be expected from these reserves is reduced to terms of profit of gold contents per unit of area, and the quarterly returns of all other necessary details are reduced to the same unit. There is thus made a straightforward and progressive means of comparison between the valuation of the mine, the profit being obtained from the current working, the estimated profit from the developed ore reserves, and the relation of the extent of the current operations and development to the prospective life of the mine. These records are thus reduced to one simple, absolute, and easily-comparable basis, which, when it is once understood, will be found to present a much clearer statement of the current operations and future prospects of the Company than has in the past been possible with the more complicated system hitherto adopted.'

I understand that this statement (made by Mr. Marriott at the annual meeting of the Village Main Reef, on June 15, 1910) means that it is easier to compare areas in square fathoms than tonnages, but I cannot see in what respect areas are the better unit for making comparisons.

Perhaps it will be plainer to take a concrete case. Let us assume that on January 1, 1909, Block No. 1 measuring 2,000 square fathoms containing £40,000 in gold; or stated in the usual manner, Block No. 1 contained 20,000 tons of £2 ore, or £40,000. If on December 31, 1909, this Block No. 1 measured 500 fathoms and contained £10,000 in gold we should know that ¾ of the block had been worked, but we should know it exactly as well as if we were told that the Block contained 5,000 tons of ore worth £10,000. And for my part I cannot see that the statement in square fathoms gives any more information than does the statement in tons. What is true of one block will be true of all the blocks and hence true of the whole mine. It does not follow, however, that because there is no need of altering the present system of keeping accounts on the Rand, as regards the ton-milled unit, that Rand mining methods and Rand accounts are perfect, and, in fact, Mr. Marriott has pointed out a number of places where improvements can be made.

One improvement that is needed not only on the Rand but in other mining districts is the adoption of some system by which the capable and hard-working manager shall get credit for his good work. As mines are now run, 'a good mine makes a good manager' and conversely the best manager living will gain but little reputation if he is unfortunate enough to have charge of a poor mine. At present the criterion of good management is apt to be low working-costs per ton when in reality maximum profit-making should be the sole test.

Every engineer knows that it is possible to have two adjoining mines, working on similar ore deposits, mine A with high working-costs per ton making a profit, and mine B with low working-costs making a loss. Mr. Marriott concludes from this well-known fact that a mere comparison of working-costs per ton gives no information as to technical skill and business judgment of the management, and seems to think that a comparison of the profits made per square fathom would be fairer to the managers than a comparison made on tonnage costs; but I feel quite sure that if Manager A made £10 profit per square fathom worked, while Manager B only made £2 the average shareholder would immediately conclude that A was the better man, although a careful study of all the conditions might show B to be A's superior in all respects.

It is certainly discouraging for an ambitious manager to be always tied to poor mines and thus never have proper credit for his careful work. In an ideally perfect community it would be possible to classify the mine managers according to the excellence of their work and without reference to profits made by the mine over which they had charge, but I doubt if this idea can be put into practice at present.

Another improvement that ought to be made is the furnishing of the shareholder with an official valuation of his shares and a statement of how this valuation is arrived at. The present practice of throwing an undigested mass of statistics at the shareholder's head is not instructive, and the introduction of the square fathom would give no additional information of value.

In my opinion every annual report should contain an official valuation of the shares of the company concerned. This could surely be done by some independent committee of course unconnected with financial interests of any kind.

A report such as the following: "We value the shares of the So and So Mining Company at £3½" would be worth more to the shareholder than libraries of printed statistics and acres of mining maps. I am so confident of the value of such a system of independent estimates that I take this opportunity of suggesting that such a course should be taken."—C. W. PURINGTON, W. H. SHOCKLEY, *The Mining Magazine*, July, 1910, p. 115. (J. E. T.)

BAGGERIDGE COLLIERY EQUIPMENT.—"Great interest centred in the pit bottom, which is being arranged for the simultaneous loading of four-deck cages, requiring the removal of a large area of the strata to form the insets, and extensive robbing of natural support on opposite sides of the shaft. To compensate for the weakness caused by such removal arches of special construction have been constructed. These extend for a distance of 15 yards on either side of the shaft and are roughly elliptical in section. The greatest diameter in the clear of each ellipse is vertically from floor to roof 54 ft., while the greatest diameter at right angles thereto is 27 ft. These arched galleries starting at the shaft, 14 yards below the floor of the Staffordshire 10 yard seam, which is here about 20 ft. in thickness, are formed with concrete. The appearance presented to view when making an inspection of them is that of a smooth surface, the bricked portion being merely the surface lining next the roadway, behind which the specially reinforced concrete has been first placed. At intervals, however, the bricked surface is hidden from view by large steel channel frames which encircle the chamber and connected cross girders, one at each decking level, to strengthen the resistance to the immense side pressure already in evidence.

At the end of the inset galleries the continuation of the roads is of reduced section, more particularly in height, merging into barrel arches of 21 ft. diameter, the whole being carried at a pre-determined regular gradient, irrespective of the dip of the seam.

The actual lining of one of these roads now in progress with reinforced concrete, together with the preliminary timbering for supporting the large excavations necessary, were seen, and were objects of great interest to the visitors. In these roads there is 14 inches facing of brickwork behind which is a 3 ft. thickness of concrete reinforced by Kahn bars. Experience has rendered some modification necessary in the shape of the bars which, when first used of arched shape in the roof, were unable to resist the strains thrust upon them, resulting in breakage of the brickwork and downward protuberance of the bars along the centre of the roadways. Minor roads of 15 ft. and of 8 ft. widths are also in course of construction or intended to be, and in this way will be carried a considerable distance and hereafter be used as main haulage roads.

The Ransome ver Meyer machine for mechanically mixing the concrete, consisting of crushed slag, sand and Portland cement, with a regulated water supply to the mixer, was also inspected. This is driven by a small engine from a belt on the surface, and the concrete is discharged into tubs ready to be sent down the pit. The pit frame is of steel lattice girders 75 ft. in height.

The main winding engines are 4-cylinder tandem compound with Corliss valves, built by Messrs. Fraser & Chalmers, Ltd. The drum is slightly conical at each end and parallel in the centre, 18 ft. in diameter. The engines are fitted with Whitmore's steam reverser and steam brake, and are capable of raising 400 tons per hour from a depth of 643 yards. The boilers are of the Lancashire type, 30 ft. by 8 ft. 6 in., carry 150 lb. pressure, and are fitted with Sugden's downtake superheaters and Green's economiser."—*Iron and Coal Trades Review*, July 1, 1910, p. 14. (A. R.)

MISCELLANEOUS.

AUTOMATIC FILTER.—"The firm Albert Kahlert, Berlin, have brought out a filter, which is suitable for filtering tinctures, wines, liqueurs, photographic solutions, etc. It consists of a ribbed and perforated porcelain cylinder, round which the filter paper is rolled and kept in place with rubber bands. The firm maintains that 50 litres of wine or tincture can be filtered in an hour, without any waiting, and by using only a quarter of a sheet of filter paper.—*Chemiker Zeitung*, No. 74, 1910. (T. D.)

SCHOOP'S "SPRAY" METHOD OF METAL DEPOSITION.—This method seems to be superior in many respects to electro plating. The metal to be deposited is forced through a suitable nozzle by means of highly compressed and heated gases and thrown in a fine spray or mist against the object to be plated.

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The process does got require the same constant supervision demanded by the electro plating process, and by it on account of the absence of heat, bodies with little heat and fire resisting qualities such as wax, paper and wood, can be coated.—*Zeitschrift fur Angewandte Chemie*, July 1, 1910, p. 1214. (T. D.)

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Abstracts of Patent Applications.

(C.) 520/09. Thomas Henry Bradbury. Improvements in electric resistance furnaces. 12.11.09.

This invention relates to an electric furnace in which a metallic wire is wound in the form of a solenoid embedded in slagwool around a muffle of suitable refractory material such as clay, etc. The furnace is intended for heating drills, assaying, smelting bullion, etc.

(C.) 461/09. Hans Charles Behr. Improvements in ore storage and handling devices. 9.10.09.

The object of this invention is to allow the most economical ore bin in a given space and to provide feeding means specially adapted for handling large quantities of ore.

The ore which is discharged from chutes under the storage bins is transported to the mortar boxes by a travelling belt of constant speed in a constant stream less in quantity than the average consumption of the stamps, and the excess feed is furnished by a well known automatic feeder actuated by the stamps, or the transporting belt itself is arranged as an automatic feeder in such a way that when the mortar box is filling up an electric resistance is put before the C.C. motor which drives the belt and thus reduces the speed of the latter.

(C.) 411/09. Gustavius Adolphus Sheeley. Improvements in linings for tube mills and the like. 8.9.09.

This application relates to improvements in tube mill liners, and consists in providing renewable pegs or posts which are wedged in grooves formed in segmental plates, and turned to fit the interior of the mill. Various forms of grooves and fittings are shown in the drawings and are covered by the claims.

(C.) 566/09. Robert Allen. Improvements in apparatus for filtering or clarifying liquids. 8.12.09.

This application is for a clarifying filter, in which submerged filter-leaves of the type of the Moore or Butters are used. The filtering action is due to the pressure exerted by a low head of liquid. The apparatus is designed for the clarification of liquids cloudy with suspended matter.

(C.) 322/10. Nobels Explosive Co., Ltd. (1), William Tuxford Evans (2). Improvements in electric igniting apparatus for fuses and the like. 7.7.10.

This application relates to a fuse firing apparatus in which a series wound generator is manually driven through the agency of a quick threaded screw and nut. An arrangement is included for diverting the current during the process of operation from a circuit internal to the apparatus, to the external circuit in which are placed the fuses to be fired. There is also a magnetically actuated contact which prevents the external circuit from being closed unless the current generated exceeds a predetermined value.

(C.) 527/09. Sidney Rickman Adams. Improvements in means for regulating or shutting off the underflow from de-watering and classifying cones and the like. 19.11.09.

The application refers to a cut-off gate for thickening cones in which an inclined surface or cam in the rotating member causes the gate to close tightly over the orifice of the cone, rendering it a water-tight joint. The surface of the gate is covered with leather or other resilient material.

(C.) 203/10. Thomas McCraw. Improvements in tube mill linings. 30.4.10.

This application relates to improvements in tube mill linings and consists in the insertion of bars of steel, or other hard metal, longitudinally between and flush with courses of flint or other hard stone, to prevent wear.

(C.) 336/10. Hugh McMenamin (1), Francis Allen (2). Improvements in ore feeding means for the stamp of stamp mills. 14.7.10.

The apparatus consists of two end rollers provided with flanges, the space between the flanges being about the same width as the feed chute. These rollers from the end drums for a conveyor belt surface, and motion is imparted to the belt by means of any suitable friction drive such as is commonly in use already on other forms of ore feeder, the driving power being applied through the drum nearest the mortar box, or alternately, through both ends of drums. The rock from the ore bins slides down the feed chute on to the conveyor belt, and in order that the weight of the ore may not distort the surface of the belt, an intermediate roller (or rollers) is introduced below that part of the belt on which most of the ore rests. The applicants claim that the absence of this supporting roller or rollers in any previous similar device caused the endless belt to sag, and so rendered effective driving very difficult, and also, that the result of the irregular feeding of the ore causes excessive wear and tear, and frequent breakages of the stamps, the feeder itself, and its parts. These intermediate rollers are not positively driven, and the whole apparatus is simply an ordinary conveyor belt with intermediate idlers. No return idlers are used to support the return surface of the belt, this being unnecessary owing to the short length of the conveyor belt.

Provision is made for taking up the stretch of the belt in the usual way, *i.e.*, by means of adjusting screws by which the distance between the end rollers can be increased.

Provision is also made to apply a scraper to the underside of the front drum so as to prevent any mud or stones adhering to the belt after it has passed the feed opening in the mortar box.

The claims made are three in number. The first two cover an endless belt to be used for the purpose of feeding ore into stamp mills, in combination with intermediate rollers for the purpose described, while the third claim covers the combinations with a scraper under the front drum.

(C.) 328/10. Otto Liebknecht. Improvements in the manufacture of sodium cyanide. 9.7.10.

This application describes a method of preparing sodium cyanide by passing the vapours of trymethylamin into molten metallic sodium.

(C.) 511/09. Sidney Sherard Osborn. Improvements in tube mill linings. 10.11.09.

This specification describes a tube mill liner with roughened or corrugated pegs or ribs to facilitate retention of pebbles in the channels.

Selected Transvaal Patent Applications.

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Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

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(C.) 460/10. Arthur Frederick Gunstone. Improvements in or relating to detachable wheel rims. 16.9.10.

(C.) 462/10. Isaac Francis Taylor (1), Samuel Pick (2). Improvements in cranes. 17.9.10.

(C.) 463/10. Peter Petersen. Improvements relating to granolithic paving slabs and the like. 17.9.10.

(C.) 464/10. Kurb Wilhelm Otto Schweder. Improvements relating to centrifugal pumps. 17.9.10.

(P.) 465/10. Lars Pedersen. Improvements in arresting dust produced in rock drilling. 20.9.10.

(C.) 466/10. Thomas Taylor. Improvements in woven driving belting. 20.9.10.

(P.) 467/10. William Henry Hutchinson (1), Malcolm Canmore Inglis (2). Improvements in gripping and handling appliances. 21.9.10.

(C.) 468/10. Arthur Wilson Ferns. Improvements in acetylene gas generators and lamps. 21.9.10.

(C.) 469/10. Frederick Arthur Maximillian Schiechel. Apparatus for separating minerals and like materials. 22.9.10.

(C.) 470/10. Lewis Condict Bayles. Improvements in oilers. 23.9.10.

(C.) 471/10. Lewis Condict Bayles (1), Albert Hibbs Taylor. Improvement in pressure fluid tools. 23.9.10.

(P.) 473/10. Ernest Holroyd Martin. Improvements in discharge means for stamp mills and other crushing machinery. 26.9.10.

(C.) 475/10. Norman Frederick White (1), Thomas Robert Dibbin (2), Percy James Drain (3). An improved safety catch gear for mine skips, or cages lifts, elevators and like vehicles. 27.9.10.

(P.) 474/10. John Webb. A self supplying water power pumping machine. 26.9.10.

(P.) 476/10. Daniel Edward Hockly (1), Herbert Caughan (2) Improvements in machines for husking shelling and grading maize. 27.9.10.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

ANGUS, THOS., *to* Ariston Mine, via Klerksdorp.
ARDERNE, R. A. S., *l/o* Greylingstad ; c/o Mr. D. R. Colman, P. O. Pilgrims Rest.
BALL, H. S., *l/o* London ; c/o Bank of Montreal, Montreal, Canada.
BAYLDON, H. C., *l/o* London : Kotchkar, Orenburg Government, Russia.
BLYTH, W. B., *l/o* Lawlers : c/o Messrs. Bewick, Moreing and Co., Kalgoorlie, W. Australia.
CHAPPLE, A. J., *l/o* Stawell ; Philip Street, Parramatta, New South Wales.
COTTER, A. J., *l/o* Johannesburg ; Cinderella Consolidated G. M. Co., Ltd., P. O. Box 75, Boksburg.
CRONIN, J., *l/o* Schweizer Reneke ; P. O. Box 4800, Johannesburg.
DANIEL JAMES, *l/o* Langlaagte ; New Primrose G. M. Co., Ltd., P. O. Box 193, Germiston.
DORR, J. V. N., *to* 728, Equitable Building, Denver, Colo., U.S.A.
EVANS, SAMUEL, *to* 79, Nugget Street, Johannesburg.
FRASER, A., *l/o* Benoni ; P. O. Box 14, Brakpan.
GERLINGS, H., *to* P. O. Box 3786, Johannesburg.
GIBBINGS, G. W., *to* P. O. Box 390, Salisbury, Rhodesia.
GIBSON, J. R., *to* P. O. Box 587, Bulawayo.
GOSNEY, P., *l/o* England ; Old Nic Mine, Bulawayo, Rhodesia.
HOLDEN, W. C., *to* North Randfontein G. M. Co., Ltd., P. O. Box 31, Randfontein.
ISAACS, R. McI., *l/o* London ; Don Proprietary Mines, Ltd., Gwelo, Rhodesia.
JOHNSON, TOM, *to* Ferreira Deep, Ltd., P. O. Box 1056, Johannesburg.
LEE, H. J., *l/o* Randfontein ; New Goch Gold Mines, Ltd., P. O. Box 1096, Johannesburg.
LEYSON, WM., *l/o* Rhodesia ; Roberts Victor Diamonds, Ltd., Victor, O.F.S.
MACKENZIE, A. WILKIE, *l/o* Sabie ; Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg.
MELVILL, GEO., *l/o* Florida ; Geldenhuis Deep, Ltd., Cleveland.
NICOLAUS, R. C., *l/o* Johannesburg ; Mining and Metallurgical Club, St. Ermines, Westminster, London, S.W.
PARSEA, A. E., *l/o* Eureka ; Lonely G. M. Co., Inyati District, Rhodesia.
PERKIN, D. W., *l/o* Knights ; New Unified Main Reef G. M. Co., Ltd., P. O. Box 5, Maraisburg.
ROBERTS, S., *l/o* Krugersdorp ; P. O. Box 361, Salisbury, Rhodesia.
ROSS, A. W., *l/o* Day Dawn ; Vivien G. M. Co., via Lawlers, W. Australia.
SCHWABE, S. B., *l/o* Krugersdorp ; Staff Quarters Crown Mines, Ltd., Fordsburg.
SHARP, E. MILES, *l/o* Ariston ; New Modderfontein G. M. Co., Ltd., P. O. Box 25, Benoni.
SMITH, ROWLAND, *l/o* Umsweza : Harvey's Chance, Golden Valley, Rhodesia.
STACPOOLE, A. R., *l/o* Rhodesia ; French Bobs Mine, Louws Creek, Barberton.
SUMMERS, EDWARD, *l/o* Klerksdorp ; Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights.
THOMAS, J. FRED, *l/o* Knights ; "Miramar," Perranporth, Cornwall.
TUCKER, P. A., *l/o* Germiston ; Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg.
VAUGHAN, J. E., *l/o* Boksburg ; Inspector of Mines Office, Winchester House, P. O. Box 1132, Johannesburg.
WAGNER, Dr. P. A., *l/o* G.S.W. Africa ; P. O. Box 2083, Johannesburg.
WHELER, A. S., *l/o* London ; P. O. Box 48, Salisbury, Rhodesia.
WHITE, C. K. *l/o* Johannesburg ; Crown Mines, P. O. Box 156, Fordsburg.
WHITE, H. A., *l/o* Germiston ; Roodepoort Central Deep, Ltd., P. O. Box 114, Roodepoort.
YATES, JOHN *l/o* Johannesburg ; P. O. Box 401, Salisbury, Rhodesia.

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OF SOUTH AFRICA.

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Vol. XI. NOVEMBER, 1910. No. 5.

Proceedings
AT
Ordinary General Meeting, November 19, 1910.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the South African School of Mines, on Saturday evening, November 19th, Dr. James Moir (President) in the chair. There were also present :—

43 Members : Prof. G. H. Stanley, Messrs. C. B. Saner, W. R. Dowling, R. Allen, K. L. Graham, Tom Johnson, E. J. Laschinger, H. A. White, Prof. J. A. Wilkinson, S. Beaton, W. Beaver, A. J. Bowness, G. L. Burnett, J. Chilton, M. H. Coombe, T. Donaldson, R. Gascoyne, W. W. Hesom, A. B. Inglis, A. E. Irwin, J. H. Johnson, A. J. Johnson, G. A. Lawson, J. Lea, W. D'Arcy Lloyd, H. S. Macgregor, F. A. McCoy, P. T. Morrisby, E. Pam, J. F. Pyles, A. Redfern, E. Roberts, G. A. Robertson, A. H. Searf, G. H. Smith, S. H. Steels, Ralph Stokes, W. A. C. Tayler, A. Thomas, C. Toombs, C. F. Webb, and E. M. Weston.

20 Associates and Students : Messrs. M. S. Archibald, W. E. Burrows, F. C. Carbis, G. J. V. Clarence, J. Cronin, C. N. Davies, C. L. Dewar, W. J. N. Dunnachie, J. Gibson, O. Harrison, B. W. Holman, A. King, L. T. Leyson, S. H. Olivier, F. J. Pooler, H. B. Powter, T. L. Thorne, W. Waters, F. Wartenweiler and E. J. Wiseman.

18 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as printed in the October *Journal*, were confirmed.

NEW MEMBERS.

Messrs. W. Beaver and E. J. Laschinger were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows :—

Creasey, Walter John, P. O. Box 1112, Johannesburg. Cyanider.
Giddey, Claude Leslie, East Rand Proprietary Mines, Ltd., P. O. Box 66, East Rand. Cyanider.
Thomas, Arthur, Durban Roodepoort Deep, Ltd., P. O. Box 110, Roodepoort. Shift Boss.
Wells, Eric Francis Vesey, Village Deep, Ltd., P. O. Box 1145, Johannesburg. Mine Captain.

The Secretary : Since the last meeting o the Society the following have been admitted by the Council :—

As Associates.—

Chester, William, P. O. Box 463, Germiston. Amalgamator.
Filmer, Jr., Harry Joseph, P. O. Box 1409, Johannesburg. Assistant Amalgamator.
Freeman, Charles Cuthbert, Great Fingall Consolidated G. M. Co., Ltd., Day Dawn, Western Australia. Metallurgist.
McDivett, David, Geldenhuis Deep, Ltd., Cleveland. Cyanider.

As Students.—

Cowles, Eugene Pomeroy, Simmer Deep, Ltd., P. O. Box 178, Germiston. Reduction Works Learner.
McLean, Calvin Stowe, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Tube Mill Learner.

General Business.

The President : I have a letter of apology from Mr. Stuart Martin, who regrets he is unable to attend this meeting through illness.

I have also to state that our Committee on Mining and Metallurgical terms has extended the date for suggested alterations to the end of this month. In the meantime we have received a letter from the Institution of Mining and Metallurgy stating that it is going to reopen the question. I think the subject may as well be closed at this end, and that Institution, with its greater opportunities, be allowed to complete the matter finally.

A third matter which I have to mention is the address to the Duke of Connaught from the ten local Societies, a photograph of which you may see here. The original has been finished and signed, but unfortunately it is not visible, as it has to be bound in leather and will only be ready at the last moment.

CYANIDE AS AN INSECTICIDE.

Prof. J. A. Wilkinson (*Member of Council*): In order to prevent any misconception which might otherwise possibly arise in connection with an abstract recently inserted in the *Journal*, and mentioned by Mr Thomas at the last meeting, I beg to state that the procedure adopted by the Council in such matters deprives abstractors of any responsibility in publication. Further, one cannot be held responsible for the experiments and opinion of others. As cyanides are used so largely here, I thought that the note would prove of interest to members. Unfortunately I came in somewhat late at the last meeting and did not know that this matter had been mentioned until the printed copy of the *Journal* came to hand.

CHEESA* STICKS.

Mr. T. Donaldson read the following contribution on behalf of the author:—

Mr. W. Cullen (*Past President*): At the last meeting of the Society I mentioned that Messrs. Brock, of London, had introduced a cheesa stick which seemed to do very well in practice. Since then further experience has been gained with it, and as a result some small modifications were made which are embodied in the samples which are exhibited. A quantitative analysis of the ingredients has not been made yet, but they consist of chlorate of potash, barium nitrate, magnesium and shellac. In order to make the ignition certain and easy, a little touch paper is wound round the working end. The stump, or handle, is composed of about two inches of wood, which enables the stick to be burned right down to the bottom.

In order to ascertain whether the gases were noxious or not, ·4 gm. of the cheesa stick mixture was ignited by electric means in a gas tight bottle of air of 3,300 c.c. The resulting gases had the following composition:—

	Per cent.
Oxygen	19·27
Nitrogen, etc.... ...	78·62
Carbon monoxide ...	·20
Carbon dioxide ...	1·91
Nitrogen, oxides of ...	Nil

As will be seen, they are very low in carbon monoxide. In order to ascertain the probable vitiation of the mine air underground, taking the composition of pure air to be:—

	Per cent.
Oxygen	20·90
Nitrogen	79·06
Carbon dioxide ...	·04

and working with the analysis given above as a basis, the following has been calculated. If one of these cheesa sticks, containing 18 gm. of combustible mixture be completely burned in an enclosed space filled with air and measuring 6 ft. × 6 ft × 8 ft., *i.e.* with a capacity of 288 cub. ft., the air afterwards would have the following composition:—

	Per cent
Oxygen	20·87
Nitrogen	79·05
Carbon monoxide ...	·0035
Carbon dioxide ..	·07

One of the advantages of this cheesa stick is that it can be burned right to the end. Supposing only one inch of it is necessary to set off a round, the remainder can be kept for further use. If used in this way it comes out quite as cheap, if not cheaper than the spit fuse, and the advantages over the blasting gelatine stick are obvious.

Once ignited, the stick is extremely difficult to put out, and it even withstands the concussion of the underground blast. It burns for 6 minutes 40 seconds to 6 minutes 45 seconds, but of course there are slight variations as between stick and stick. The flame, as you can see, is bluish, and shows up very well underground. There is only one further improvement which has been suggested, and it will be given effect to, *viz.* the waterproofing of the paper wrapping, which would make it better able to withstand the underground conditions.

The President: I am sure we all thank Mr. Cullen for his interesting contribution.

Mr. E. J. Laschinger (*Member of Council*): I was talking to one of our Past-Presidents, Mr. S. H. Pearce, and it occurred to both of us that it might be a very good idea to try the lighting of fuses with an ordinary blow lamp as used by plumbers. If a special lamp were constructed of a convenient size, we thought it would be a good thing to try. We know, of course, that the composition of the gases from such a light would be free of noxious fumes or any really dangerous constituents.

The President: It has a blue flame, has it not?

Mr. E. J. Laschinger: Yes. I bring it up as a suggestion as to whether it would not be advisable to try it, or if it has been tried the results might be ascertained.

Mr. H. A. White (*Member of Council*): I have to point out that that kind of lamp is on the same principle as the Primus stove, which was so strongly condemned by the Society a little while ago.

Mr. E. J. Laschinger: It is very much more convenient and gives a strong heating flame. I am certain from my own experience of lighting fuses, although I have not tried this lamp myself, that a blow lamp would ignite the fuses as well as any cheesa stick ever invented.

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* Cheesa is phonetic spelling of *tyisa* or *tshisa*, Zulu for hot or burning; cheesa stick literally in this instance firing stick—a substitute for spit fuse, commonly in use in South Africa.

Mr. H. A. White: Personally I have no objection to using a Primus lamp, but this Society has practically condemned the use of them.

Mr. E. J. Laschinger: In the hands of the inexperienced ordinary public.

Mr. H. A. White: Or miners!

Mr. M. H. Coombe (*Member*): I noticed that the moment water was applied to the cheesa stick by Mr. Donaldson it was extinguished. That certainly renders it impracticable for use underground, in wet winzes, shafts or faces generally. It could only be used in a very dry face. Another important factor which takes away from its efficiency is its lighting power. It gives a dull bluish light, which would be very difficult to find fuses by, and in the endeavour to evolve a substitute for the ordinary gelatine cheesa stick this most important point must be borne in mind.

Mr. J. Chilton (*Member*): We tried these new cheesa sticks at the Village Main Reef last Thursday, and there was only one complaint in the whole mine. Otherwise they were a perfect success.

Mr. T. Donaldson (*Member*): Although these cheesa sticks are put out by water, let them dry a little and they are as good as ever again.

Mr. M. H. Coombe: Referring to Mr. Laschinger's suggestion that a modification of the plumber's blue flame would be an efficient lighting medium, I consider it most impracticable. The miner is burdened with sufficient paraphernalia now, and hedged round with regulations and restrictions which sufficiently hamper him in his work without having to carry a plumber's blue flame lamp around with him, to say nothing of tins of paraffin or petrol. In a dry face no better medium can be used for lighting than a "spit fuse." If "nicked" in a proper manner it is absolutely efficient and quick, and when a 3 ft. "spitting fuse" is burnt out when lighting a round of 5 ft. fuses it is a warning to the miner to "get out" whether his fuses are all lit or not, but in wet faces nothing as efficient as the gelatine cheesa stick has as yet been devised, and the members of this Society will have a hard battle to fight before they chase it out of our mines.

Mr. J. H. Johnson (*Member*): I notice the word cheesa stick is used very continuously. Speaking of metallurgical and mining terms I believe the sub committee decided that the word cheesa stick should not be used. It is a Kafir word, and to spell it as we spell it is wrong also.

The President: I believe, Mr. Johnson, the Zulu dictionaries have special ways of spelling the word, viz., *tshisa* and *tyisa*, but we have apparently bilingualized the word for the *Journal*, and the result is not very pretty.

Mr. E. J. Laschinger: I should like to know the objection to the blow lamp. As for increasing miner's paraphernalia, I would very much rather enjoy an innocuous atmosphere with a blow lamp in a confined space than the noxious gases usually prevailing. What we have to do in the elimination of the cheesa stick is to get rid of the noxious gases arising from that stick, which are the most harmful gases the miner has ever to fear. I do not think we can exaggerate the danger these gases mean to the miner's health and the sooner something is adopted which will do away with the dangerous fumes and give off harmless gases so much the better. I think every miner ought to thank those who try to introduce something of that kind to prolong his life. Therefore I do not see any weighty argument in the suggestion that it would be extra paraphernalia. The cheesa stick takes some time to manipulate. A blow lamp might require a few minutes more to start it going, but in the actual operation of lighting the fuses I think it would take no more time and we have the argument in favour of using such a harmless flame.

Mr. T. Donaldson: This cheesa stick I have described has no harmful gases whatever.

Mr. M. H. Coombe (*Member*): I trust the members of this Society recognize the importance of this matter. One of the most important and hazardous details of the miner's work is the lighting of his fuses. The success of his round of holes depends on the correct lighting sequence of them. In addition to this the element of danger is of the greatest. Despite the advance made in the art of fuse manufacture the best brands will sometimes "run," and the miner knowing this "lights up" as quickly as possible that he may get away from the face in a minimum of time. Herein lies the efficiency of, and consequent liking by the miners for, the gelatine cheesa stick. First, it gives out a fierce heat which cannot fail to ignite the powder in the fuse; second, it burns slowly; third, it gives a strong light which enables him to see clearly and quickly what he has to do, and lights him away after he has lit up; fourth, it is easy of ignition; fifth, it is not extinguished by water falling on it; sixth, it is always at hand and simple of preparation.

I have heard of shift bosses going through the mine with a policeman's bullseye lantern hung up to their belts, but I fail to imagine the miner with a plumber's blue flame lamp hung around his neck, and the boy in the rear with a tin of paraffin or petrol.

Mr. E. J. Laschinger (*Member of Council*): I have had some experience of lighting fuses. I have been underground in a very wet shaft and lit 20 fuses one after the other after everyone else had cleared out. I am not speaking from ignorance of the subject. Although I have not had a chance of doing that sort of work this last few years, I should be quite prepared to do it at any time. I think a plumber's blow lamp would be a good thing to try and see whether it is adapted or could be adapted to the purpose. I do not wish to lay down the law that a blow lamp should be used all along the reef but because of what I know of lighting fuses and of what I know of blow lamps, I think the experiment is worth making. If the gentlemen who criticise me would try a blow lamp and let us know their experience we should know something more about it.

Mr. A. Redfern (*Member*): I think that for stopes and drives there can be nothing better than the "spitting fuse" for lighting holes. If a large number of holes has to be fired, then a fuse of equal length to those in the holes charged, should be "nicked" every half inch, the "nick" made by cutting out a small wedge shaped portion of the fuse, always reaching the core. To ensure a good light which will not be extinguished by concussion, a torch should be used, as described by Mr. E. M. Weston, or a similar one made, without using a candle, from the wax coated paper in which some firms supply explosives. When firing charges in a wet sinking shaft, or in case of falling water in stope or drive, the miner carries both knife and "cheesa stick" in one hand, takes up and holds the fuse with the other hand, while he nicks it with the knife and lights it from the cheesa stick. Therefore a blow lamp or other device, to efficiently replace the "cheesa stick," must be capable of being held and used in the same hand with the knife.

Mr. G. Hildick Smith (*Member*): I should like to draw your attention to a lamp, recently invented locally, with which experiments are being conducted at present, I believe by the Mines Trials Committee. The lamp contains a mixture of calcium carbide and calcium phosphide, and a gas is given off, which ignites spontaneously. I have personally used this lamp underground for lighting up fuses and it worked quite all right. It was impossible to blow it out. I believe experiments were also satisfactorily conducted with it on the Village Deep.

The President: Does it not get too hot to hold after using it a few minutes?

Mr. G. Hildick Smith: No, it is protected with asbestos packing.

Prof. J. A. Wilkinson (*Member of Council*): I should like to ask Mr. Saner if electric firing has been used here to any large extent. That undoubtedly is a method which would do away with the cheesa or any other kind of firing stick.

Mr. C. B. Saner (*Vice-President*): I was very keen to try electric blasting. Mr. Weiskopf, from the Dynamite Factory, offered to try it underground. On arrival at the mine I informed him we were going to blast 38 holes, but that before going below I would like to see a demonstration of it on the surface. I sketched out the actual plan of the shaft, full size. The mine captain marked out the position of the 38 holes; the fuses were connected up from outside the area of the shaft; the electric battery was pressed down: final result, eleven misfires!!! If Prof. Wilkinson would come down into a shaft bottom in two or more feet of water in a dimly lighted humid atmosphere standing amongst the fuses he would realize the difficulties of electrical firing in this particular case.

Mr. M. H. Coombe: Electric firing is efficient under certain conditions. The prime condition is that each hole must be free to carry its own burden. In other words, there must be no "easers." In the Kimberley open face workings battery firing was largely used and with success, but only on large benches where each hole had a free "burden." Electric firing can be used in shaft work, and indeed in any class of mine work, but it would be necessary to fire in "rounds," and as this is a loss of time, and a loss of time is a loss of footage, present day practice on the Rand cannot allow of electric firing. Time fuses must be used, as often the results of twelve to fifteen holes depend on the successful breaking of three or four in front of them, and these must be timed to go in front of the back holes.

Mr. W. Waters (*Associate*): With regard to Mr. Saner's remarks about misfires with electric blasting, I should like to mention that I was present with Mr. Weiskopf when the experiments were made. The misfires were not the fault of electric blasting, but must be attributed to long storage of the fuses at the Dynamite Factory. Careful examination of the fuse heads proved that the flashing mixture around the platinum bridges had become affected by dampness. I have tried electric blasting with success on various mines of the Rand, both in stopes and drives, and this method of firing holes can be made effective and efficient if miners will give it the necessary time. On the Rand every miner is in a great hurry to blast his holes and get to the surface as soon as possible. If electric blasting were insisted upon by the Government and by mine managers, there should be no difficulty in making it successful in every way. We have electric fuses of two kinds: (1) instantaneous (2) with delay action. It is possible to fire a whole face at one time, and to arrange for the shots to go off in the required rotation. I shall be pleased at any time to prove to Mr. Siner that electric blasting can be made a success of, both above and below ground.

Mr. G. Hildick Smith: I was engaged in sinking shafts in a colliery district at home where all the blasting was done by electricity, using high tension detonators fired from the mine power service, a round of sump holes being first fired simultaneously; if it was found that these had fired all right, the remaining holes making up a complete round were finally fired simultaneously also. Electric firing by means of a magneto-exploder was also successfully resorted to in driving the cross measure drifts.

THE SHRINKAGE METHOD OF STOPING AT THE FERREIRA GOLD MINE.

By G. Hildick Smith, B.Sc., F.G.S. (Member).

Generally speaking, overhand methods of stoping on the Rand are the exception rather than the general rule, owing chiefly to the average dip of the reef—which may be taken at about 30°—being too flat to admit of the adoption of these methods, and it is for this reason that the "back filling" and "shrinkage" methods of overhand stoping are perhaps not so clearly understood on this field as they should be.

The author is fortunate enough to be in charge of that section of the Ferreira Gold Mine in which a block of ground, dipping at an angle of about 70°, is being worked out by means of what may perhaps be said to be the first application of the "shrinkage" method of overhand stoping on the Rand, not taking into account, of course, the many examples of the so-called "back" stoping, which are rendered necessary on so many mines due chiefly to dykes and faults. In view of this fact the author has gone to a certain extent into the question of "shrinkage" stoping as practised in various parts of the world, and also into the "back filling" method which is closely allied to the "shrinkage" method and about which a few remarks are included.

The following paper has been written in the hope that it may perhaps be the means of bringing these methods, where practicable, into a more general use in this country, chiefly on account of the increased safety afforded in working as compared with the, at present, too common and dangerous underhand methods employed on the highly inclined areas of the reef.

The subject has been divided into the following heads:—

(1) "Shrinkage" and "back filling" methods of overhand stoping as practised in various parts of the world.

(2) A detailed description of the "shrinkage" method as applied at the Ferreira Gold Mining Co., Ltd.

(3) Advantages of the "shrinkage" method.

(4) Disadvantages of the "shrinkage" method.

(5) Conclusions and possible further applicability on the Rand and in South Africa generally.

(6) Bibliography.

(1) *Shrinkage and Back Filling Methods of Overhand Stoping.*—By the "shrinkage" and back filling methods of overhand stoping are understood those methods of breaking ground by means of which ore bodies having a dip of about 50° or over are worked out from the various levels upwards as opposed to the various underhand methods, which necessitate the working out being done from the levels downwards.

(*a*) *The "Back Filling" Method of Overhand Stoping.*—This method differs from the "shrinkage" method inasmuch as the stoped areas as they are formed are filled with waste rock, debris, etc., instead of being filled with part of the ore as it is broken. This system, as already mentioned, is chiefly applicable to highly inclined ore bodies, and hence is the system of stoping *par excellence* for sand filling; as by the adoption of the method of timbering the chutes described in detail under head 2, either by filling directly off the drive pillars, or where no drive pillars are left, off stulls, practically all the reef can be taken out. When the mine is worked out, the result is that the whole of the worked out portions can be closely filled with sand or waste, eliminating all danger to the surface, etc., due to caving ground or what not. This in theory, therefore, is the ideal method of working out a highly inclined ore body and is employed, *e.g.*, in the Butte District, Montana, and in Western Australia, etc.

(*b*) In the "Shrinkage" method of overhand stoping the ore broken is used temporarily for the filling, and therefore during what may be termed the first working, only about 33% of the actual ground broken can be directly run off for further treatment. The remaining 67% has to be left in the stopes as filling and remain there until the block being stoped is worked out, when the whole can be run off through the boxes and trammed to the shaft. This method is employed at the Alaska Treadwell Mine, Alaska, Colorado etc., and it is this method that has been employed at the Ferreira Gold Mine, described in greater detail under head 2.

To summarise therefore we have the "back filling" method, in which all the broken ore is directly available for treatment as required, suitable for moderately wide deposits requiring a minimum of timber and a minimum amount of ore to be left as safety pillars, etc., and also sufficient suitable waste material to fill up all stoped out areas.

The "shrinkage" method is employed where insufficient waste for filling is available, and it requires only a minimum of timber and a minimum of ore to be left as safety pillars, etc. : it is suitable for any width of ore body, but it necessitates a greater expense for actual mining than the "back filling" method, other things being equal, on account of the loss of interest on the money expended to break the ground until it is finally run off.

(2) *A Detailed Description of the Shrinkage Method as Applied at the Ferreira Gold Mine.*—A block of ground A, Fig. I., having been left standing between the No. 1 and No. 2 shafts, and also west of No. 1 shaft, extending from the 100 ft. to 300 ft. level, giving on the dip of the reef about a 200 ft. back and 240 ft. along the strike had to be worked out. Owing to the length of the back and the trouble experienced in the old underhand stopes working on other portions of the reef having the same dip at this mine, and also chiefly to the increased safety obtainable, it was decided to work out this block on the "shrinkage" system. By referring to Fig. I. the general condition of affairs before work was commenced on the block can be seen. The No. 1 shaft being here above the reef, it was necessary to leave a shaft pillar *b*, 40 ft. on each side of the shaft. The two remaining portions of the block A, could therefore be worked when required.

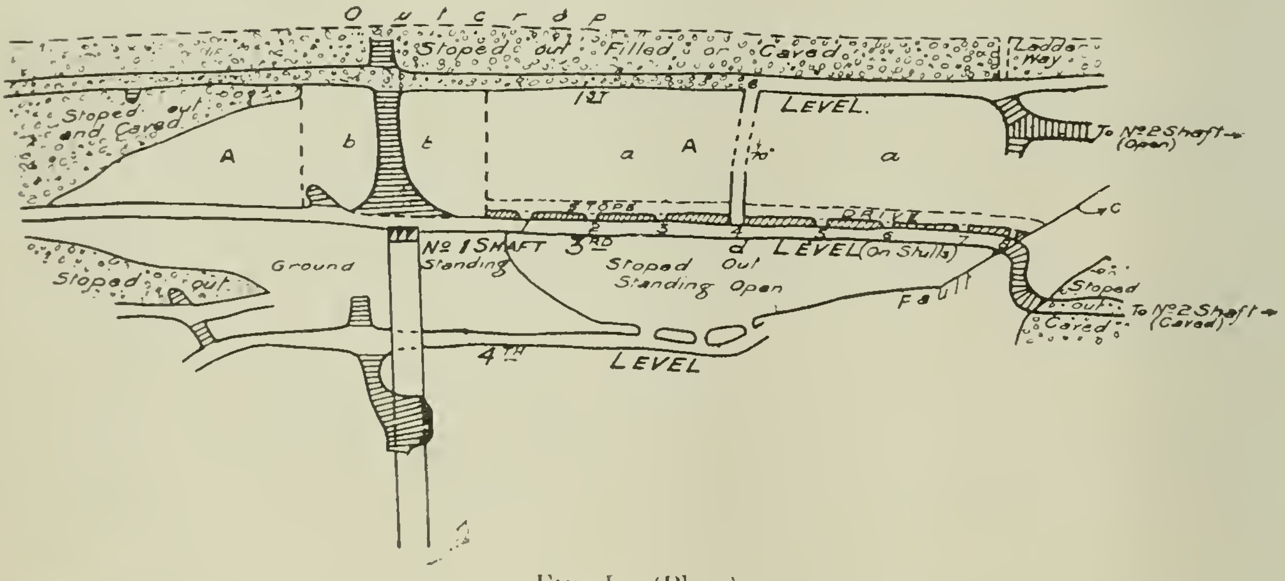

Fig. I.—(Plan.)

Work was commenced on the eastern portion *a*, as follows :—

The distance along the 300 ft. drive between the shaft pillar line and the fault *c* was roughly bisected at *d* and a rise commenced to connect to a winze *e* sunk down from the 100 ft. level for about 50 ft. The rise was put up by means of Waugh drills, and at the same time the box holes about 30 ft. apart along the drive were also put up to a height of 15 ft. As will be seen from Fig. I. the ground below the block A had previously been stoped away as far as possible by the underhand method causing the 300 ft. level itself to be carried on stulls (as shown in Fig. VI.). The hanging being good, this old stope was still standing open as far as the 400 ft. level. On this account it was thought advisable to leave drive pillars above the 300 ft. level instead of, as might have been possible, taking all ground out from immediately above the level and replacing it by a row of stulls with box holes at convenient intervals. A stope drive was carried east and west from a point 15 ft. up the raise to hole the box holes and form the drive pillars. Starting with the idea that we are going to stope and fill with broken ore, it will be seen that it is necessary to have some definite and safe way of getting to the face as it proceeds upwards, and also to have an easy method of handling any broken ore in excess of that required for filling. It was therefore decided to carry up three pass ways in all, these to be put in and reared up as stoping proceeded, from the box holes Nos. 2, 4, and 7, Fig. I., and in the remaining box holes, Nos. 1, 3, 5, 6, 8, to put in ordinary boxes of the type shown in Fig. II.

For this kind of box all that is required is that the box holes should be squared up by "pop

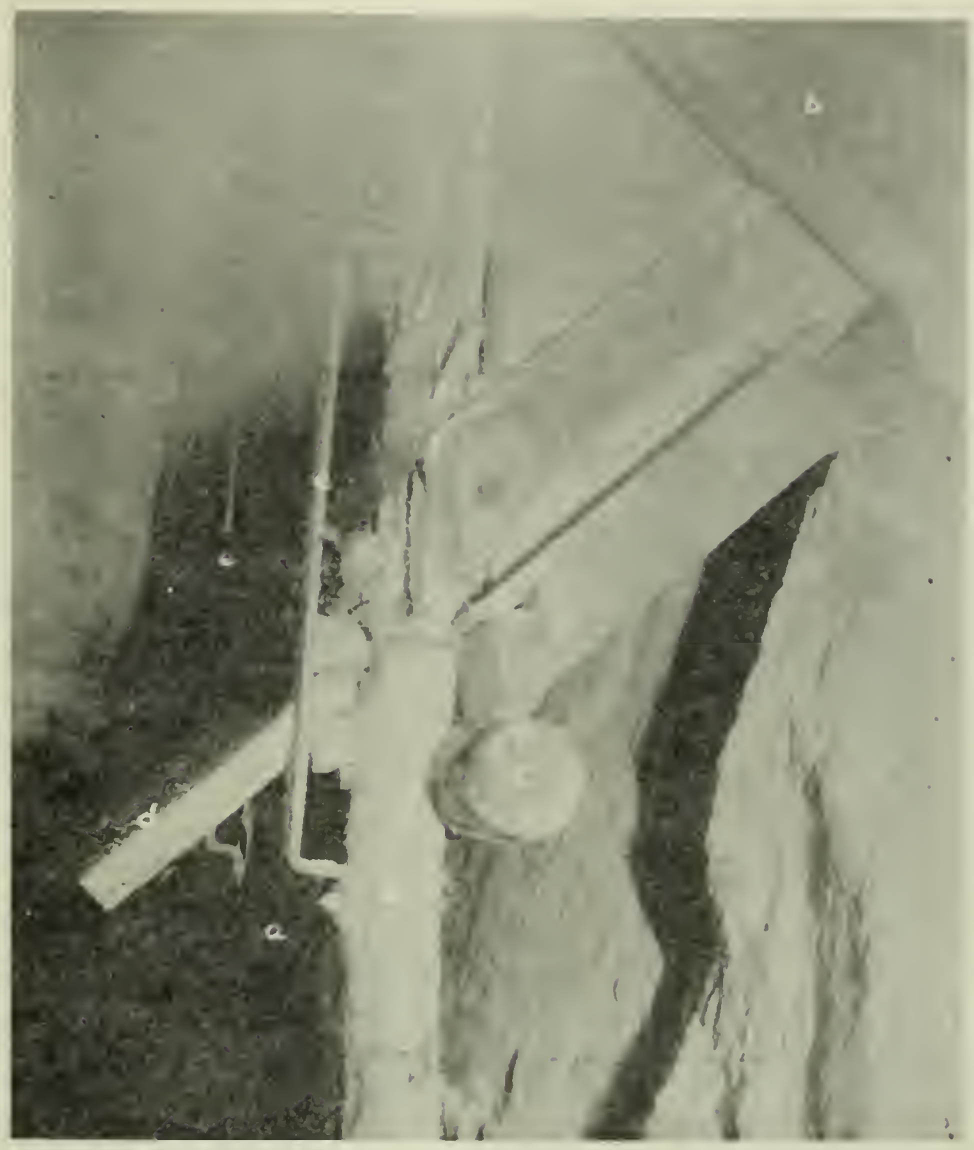

Fig. II.

holes if necessary, to allow for a good head and hitch for each of the two back legs *bb*, Fig. II. These legs should be about 12 in. diameter, and well put in. To these are bolted two strong lagging pieces *aa* to form the front legs of the the box, across which is bolted another lagging piece *c* as a bearer for the bottom planks *d*. The box is closed in at the sides and across the back legs for a sufficient distance down from the hanging with 2½ in. planks, and ½ in. × 2½ in. iron straps *ee* are bolted on the front legs to hold the front boards, which can be taken out when required to run ore from the box. A box of this description can be put in in one shift by a white man and five boys at a cost of 35s. for labour and 37s. for timber, bolts, etc., or a total cost of 72s. Bearers for the setts used to form the ladderways and ore passes in the box holes Nos. 2, 4 and 7, Fig. I. were then put in. The bearers were of 9 in. × 6 in. timber and were three in number, the outside two to carry the end pieces, and the centre one to carry the dividing piece of the first sett. The setts were cut from 3 in. × 9 in. gum planks, and were put together as shown in Fig. III., *aa* being the wall plates, *bb* end pieces, and *c* the divider, between the ladderway and ore pass. It is important that the setts should be made of gum planks or some equally hard wood, in preference to deals, which are too soft and would be broken up before stoping was finished. Each complete sett costs 17s.

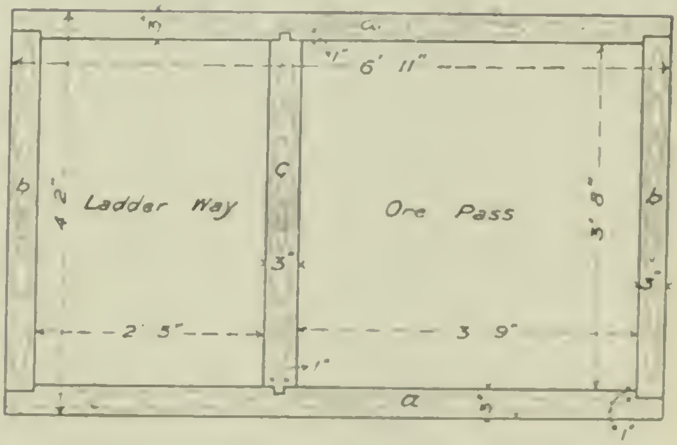

Fig. III.

for timber, and one carpenter could cut sufficient setts to keep the work going underground by cutting out 10 setts per day, giving a cost for labour on the surface of 2s. per sett. One white man and 5 boys, on single shift, put in on an average 12 setts per shift underground, which were sufficient to keep pace with the stope, (when stoping was being done on double shift) giving a cost for labour underground of 3s. per sett or a total cost per sett, when in position, of 22s. making a cost per foot for each chute of 29s. 4d. This gives a cost for timbering of 1s. per ton broken for the particular block of ground under consideration. Another important step in the building up of the setts is the proper placing of the three bearers already mentioned. These should be placed as nearly as possible at right angles to the dip of the reef—and hitched in the foot and hanging—so that when the setts are built up from them, little or no curve is required at the start to bring the setts into the line necessary to carry them straight up through the stope. This is shown in *a*, Fig. IV., not as at *b*, Fig. IV., which necessitates a curve, an easy thing to make if only a slight curve is required, but if too sharp a curve is necessary with only an inch joggle, the dividing pieces will be out of their joggles at either their foot or hanging wall ends, depending on which way the curve is necessary and a wide gap will consequently be left between the setts.

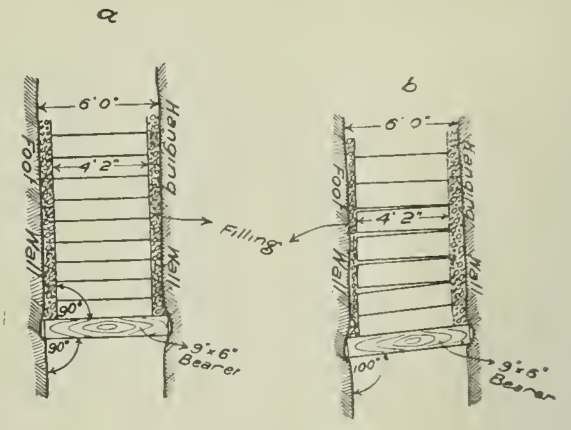

FIG. IV.

For the sizes of the setts used the bearers were placed 3 ft. 7 in. apart for the ore-pass and 2 ft. 3 in. apart for the ladderway. The bearers having been carefully fixed into position, the first sett can be laid directly on them and secured in

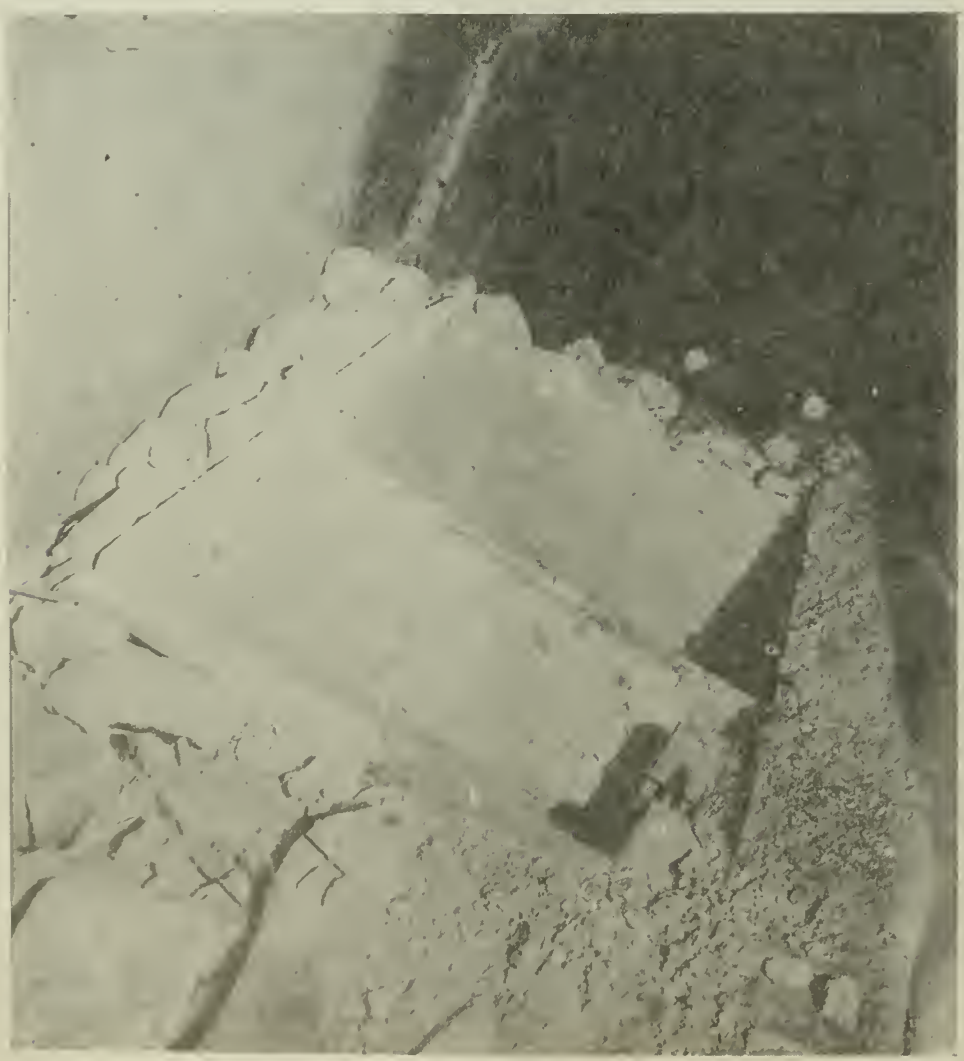

FIG. IV*a*.

any position required by filling in and wedging up around the wall plates and end pieces; no bolts, nails, etc., being used throughout, each sett simply lying directly on the one below, the whole being packed in and wedged tight, as may be seen from Fig. IV*a*., p. 196. The setts should be carried up to within about 6 ft. of the stope face. Due to the fact that as the stope face proceeds upwards any broken ore shovelled into the ore-pass would have some distance to fall and would therefore smash up the type of box shown in Fig. II., it was necessary to put in a different kind of box at the bottom of each ore pass. The ore-pass boxes were made as shown in Fig. V.

The front legs *aa* were of 9 in. × 6 in. timber morticed and bolted on to the bearers *bb* at the top, and at their footwall ends placed on a 9 in. × 6 in. baulk packed into the stull filling. A cross-bearer *c* was placed between the front legs and held in position by morticing and the bolt *d*; this bearer carries the front ends of the bottom planks of the box, the back ends of which are supported by a cross lagging piece *e* carried on holding-down bolts. The planks forming the bottom of the box were finally covered by $\frac{1}{2}$ in. iron plate *f* cut to fit behind the front legs to prevent it slipping forward. The sides were closed in with gum planks and the front fitted with a $\frac{1}{2}$ in. iron door *g*

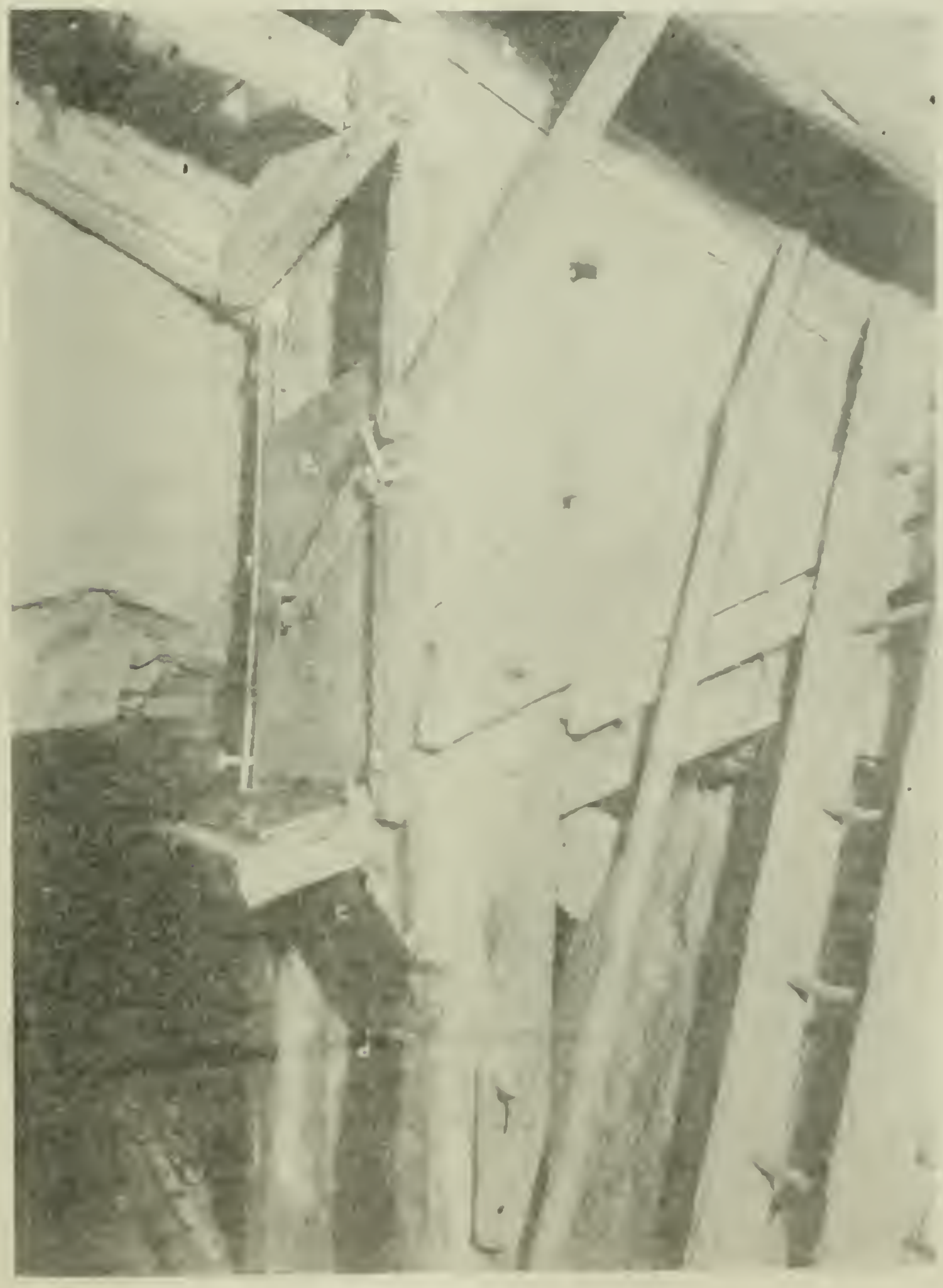

FIG. V.

moving in suitable slides and operated by means of the lever *h*.

A box of this kind can be put in in 1½ shifts by a white man and five boys at a cost of 53s. for labour and 85s. for timber, plates, etc., or a total cost of £6 18s. All air pipes were at first carried up the ladder-ways (as shown in Fig. V.), and as stoping proceeded fresh lengths of piping were added, all taps being kept below the level of the top sett.

The ladders were made of 2½ in. × 4½ in. planks, placed 10 in. apart, with ¾ in. diameter rungs, having a 10 in. tread. Ladders of this description (as seen in Fig. V.) would cost, when in position, about 1s. per ft.

A general idea of the condition of affairs after the stope had been got into good order, may be gathered by reference to Fig. VI. It was found best to have a box at the extreme east and west boundaries of the stope. On the west, therefore, the pillar, as originally cut, was stoped away by hand labour and replaced by stulls and the box put in (as shown in Fig. VI.) This allowed the broken ore to be run off more easily, and hence the cutting of the shaft pillar was facilitated, the last bench being always carried out dead in line with the preceding benches. This gave some trouble at first as the face was carried too steeply and the broken ore became blocked up at the points of the stope, the portions of the face east and west of the chutes were eventually carried at a flatter angle than the rest of the face—the best angle for which was found to be about 35° (as shown in Fig. VI.)—and no further difficulty in this way was experienced.

The main reef and leader run together giving a total width of reef about 12 ft. The main reef portion being of poor value, the leader bands only were stoped on to the so-called interbedded dyke, which is here slightly decomposed giving a narrow soapy band and making an excellent footwall to break to. The hanging also is very regular and forms a good breaking plane, the average stope width being about 6 ft. All stoping was done on contract, double shift, at a price of 40s. per fathom, and the average results obtained were as follows :—

Fathoms per machine per shift = 0·582.
Tons per machine per shift = 10·48.
Cost of explosives per fathom = 12s. 1d.
Cost of explosives per measured ton = 8·06d.
Pounds of explosives used per ton = 0·57.

Giving a total cost per measured ton broken of 3s. 1d : this includes contractor's earnings, cost of upkeep of machines, drill sharpening, etc., but exclusive of timbering.

Usually four holes were drilled to a bench, in which case the two top holes were drilled so as to just carry water, and the front holes were drilled as nearly parallel to the face of the bench as possible. All holes were drilled nearest together at the collar and leading outwards in order to reach the foot and hanging at the back ends as nearly as possible. Sometimes a bench could be carried

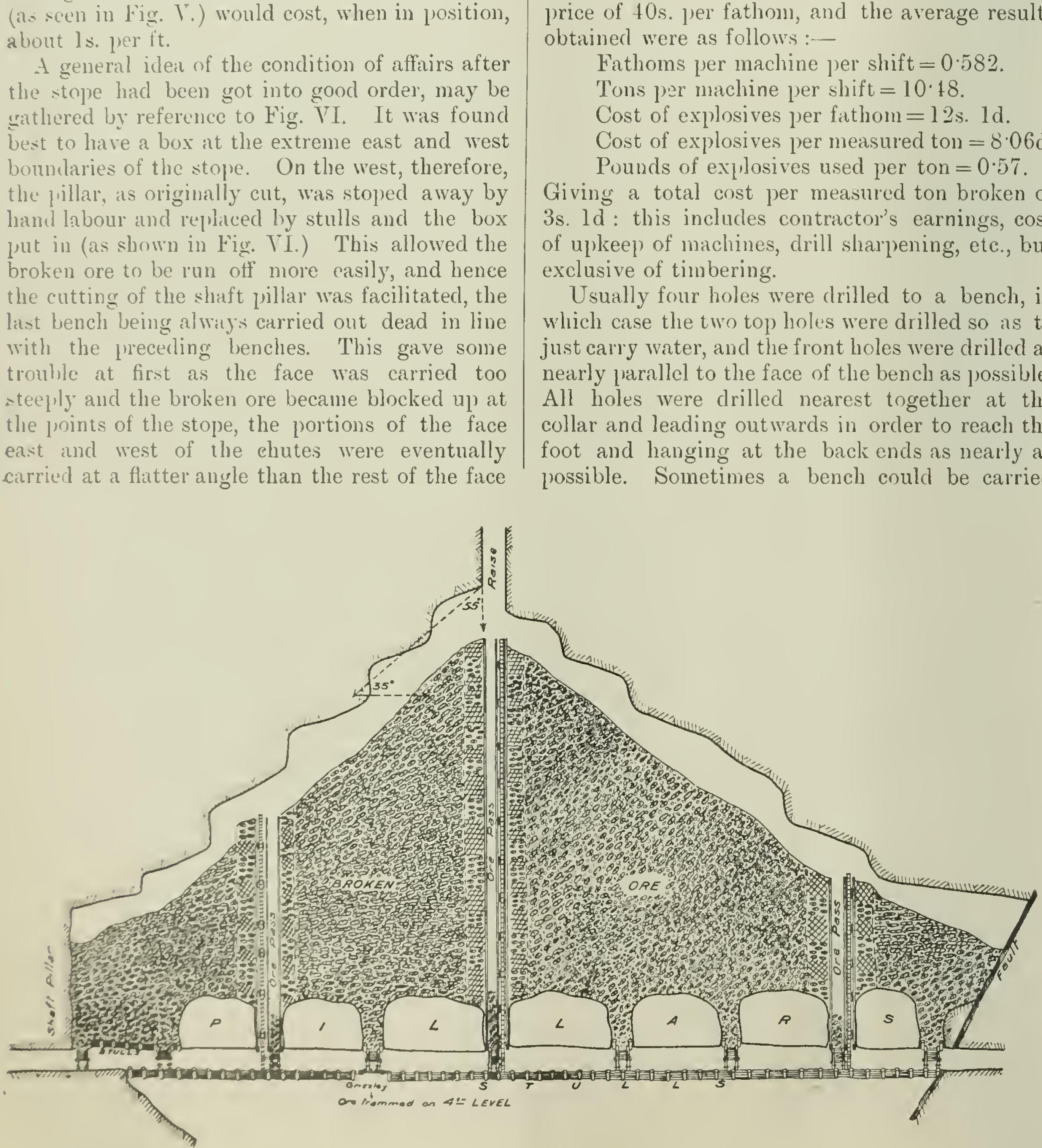

Fig. VI —(Section.)

with three holes, in which case the two back holes were drilled as usual and the third in front in the centre of the bench. There are one or two points in the handling of the machines which may be of interest. It often happens that the filling is not sufficiently near the face to allow of the bar being rigged vertically from it, in which case either (1) woodblocks were piled up from the filling a sufficient height to allow the bar to be rigged vertically; or (2) the bar is rigged across the stope from the hanging to the footwall, and in order to do this a temporary staging if necessary, may be fixed by either :—

(*a*) Placing two lagging pieces hitched in the hanging and driven down on the footwall, level with each other and a few feet apart, across which planks are placed, forming a level platform. As most of the weight on these is on the footwall side the tendency is for the lagging to be tightened in place ; or (*b*) the staging is formed by placing planks *aa*, Fig. VII., across two lagging pieces *bb*, or across one lagging and back on to the filling. These in turn are supported by means of carrying bolts *cc* placed as shown in Fig. VII. in holes drilled at convenient heights in the lagging pieces *dd* which are placed vertically on the filling.

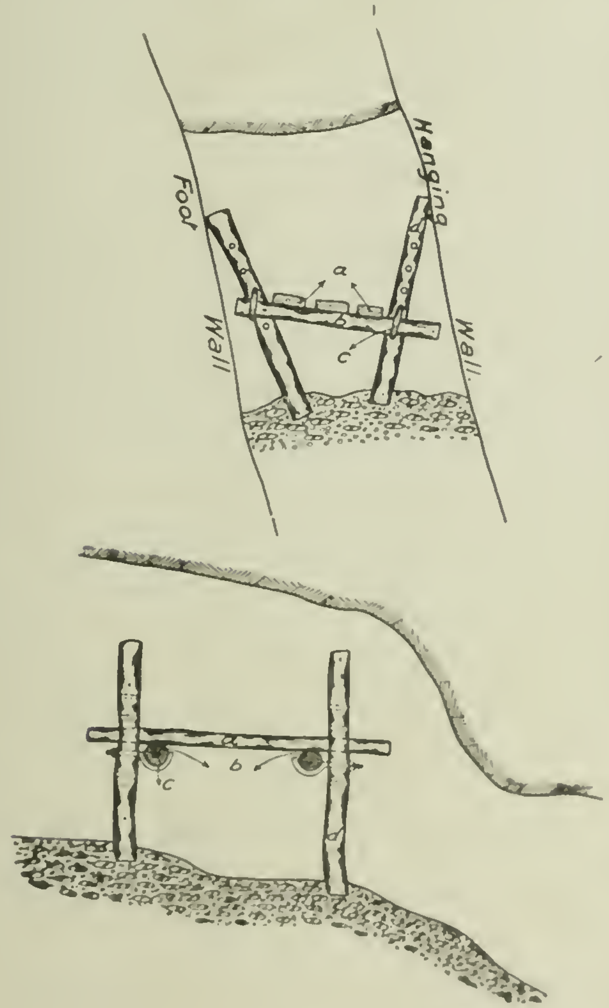

Fig. VII.

Advantages of the Shrinkage Met[illegible] The chief advantage of this method of sto[illegible] its simplicity and safety. No trouble is [illegible]red by bad hanging, as the only port[illegible] the hanging which can poss[illegible] fall away [illegible]ll area between the filling and the fa[illegible] can always be easily examined. The fa[illegible]ay be liable to fall away [illegible] it would [illegible] be temporarily support[illegible]g. Air pipes give no [illegible] as th[illegible]ed directly up the ladd[illegible]ss-ible for repairs, etc. [illegible] be carried as required [illegible] or narrowing down, an[illegible]tly experienced if holes a[illegible]ing thereby breaking it [illegible] A[illegible]ent of air is kept along th[illegible]s upcasts and owing to th[illegible]all area between the fi[illegible]nt ventilation is obtained [illegible]vel-ling is required and [illegible]un off where necessary [illegible]es. Prospecting crosscu[illegible]pnt into the hanging [illegible]nt required with great [illegible]ne trouble.

(4) *Disadvantages* [illegible] Only about 33 [illegible]ly available, the remain[illegible]til the whole block of g[illegible]s is not of so much co[illegible]he whole mine is work[illegible]he necessary tonnage t[illegible]ld be obtained from [illegible] but in cases where [illegible]es are chiefly worke[illegible]y means that so far as [illegible]is concerned, th[illegible] taken into accoun[illegible] ing up the requi[illegible] of interest on th[illegible] ground by this met[illegible] due to carelessnes[illegible] spared is run off [illegible] sinks too far away [illegible] and trouble is r[illegible] machines ; in fact [illegible] them up on the [illegible] the stope in prope[illegible] breaking the gro[illegible] affected adversely [illegible] of dry holes are of[illegible] of the rise or t[illegible] good bench. The [illegible] of a certain amou[illegible] of a spray is nece[illegible]

(5) *Conclusion*[illegible] *cability on the Ra*[illegible]

ally.—It is, after all, in a proper regard to detail that most care is needed to ensure success in any operation, and it is due to a proper regard to detail that success or otherwise is generally obtained. This is especially the case with regard to this overhand method of stoping as compared with the usual underhand method in vogue on this field. To obtain good efficiency underground it is first necessary, other things being equal, that the actual breaking of the ground should be done in as efficient a manner as possible. In order that this may be accomplished good benches must be carried and the holes given a full burden. Acting against this in the overhand method of stoping, we have, where the stope width allows of the bar being rigged across the stope, the difficulty of drilling a four hole bench in order to give each hole a proper share of the burden. This will perhaps be better understood from Fig. VIII

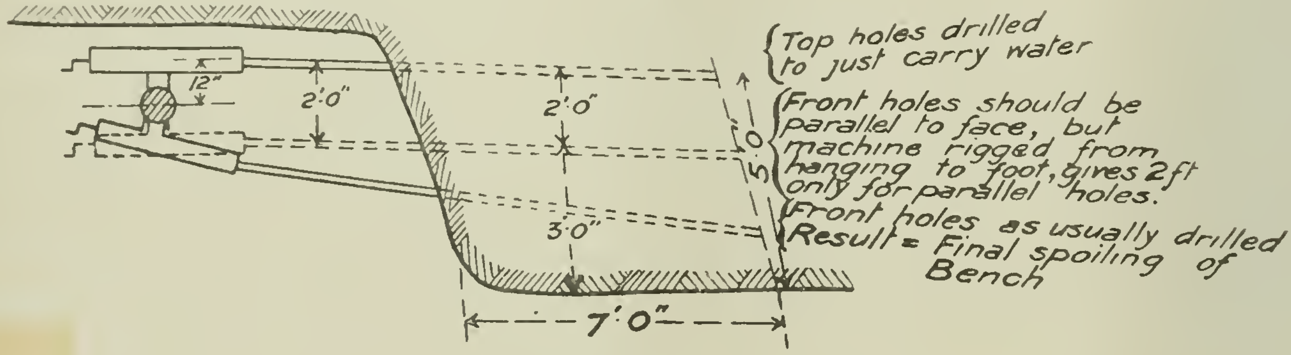
12"
2'0"
2'0"
3'0"
7'0"
Top holes drilled to just carry water
Front holes should be parallel to face, but machine rigged from hanging to foot, gives 2ft only for parallel holes.
Front holes as usually drilled
Result = Final spoiling of Bench

Fig. VIII.

The tendency is, as a rule, to run the bottom holes in such a way as to break off the triangular bottom corner of the bench rather than to run them parallel to the face of the bench. Thus the bench as started on may be a good one, yet after a few rounds the bench is spoilt, and in order to make a good bench again, dry top holes are necessitated and a short and bad round is the result. The chief point in which care should be exercised, is in the rigging of the cross bar in such a position to allow of the top holes being as high as possible and at the same time to just carry water.

As already mentioned, the best angle of dip for the stope face is about 35° (see Fig. VI.). If a steeper angle than this is carried, the broken ground from the bench higher up is liable, on blasting, to run down the stope, and the large rocks will become jammed between the next lower bench and the filling. It is an important point in this connection that a bench should not be carried forward far enough to run in above the next bench lower down the face and form one large bench, which cannot be carried in one lift. When this happens the broken ore will always get jammed against the point of the bench and thus cause delay and danger in shovelling and drilling.

With regard to the building up of the chutes when each has a ladderway, they should be so built up that the ore as it "rills" down will run directly into the ore-pass side and not into the ladderway side (see Fig. VI). This does not apply to the leading or highest chute—in this case the centre chute. The centre chute was carried at first directly up the line of the rise, but owing to the rise having gone off the original line from about the middle of the stope, the east side of the chute eventually came in line with the west side of the rise. This was found to be an advantage, as at first when benches were taken off each side of the rise at the same time, the chute became covered with a large pile of ore which took some time to remove, and eventually only the broken ore from one bench could cover the chute, which was thus more easily cleared in order that the chute might be built on to or the air pipes reached.

With regard to the air pipes, 1½ in. pipes were at first taken up each ladderway and kept close up to the top of the pass, but as already mentioned owing to the difficulty and waste of time in clearing debris from off the centre chute, 2 in. pipes were carried down the rise from the 100 ft. level, these being in 10 ft. lengths in the rise itself, to allow of a short length being taken off as stoping proceeded upwards.

Jumpers at first were carried up the ladderways from the 300 ft. level, but were finally let down by means of a hand winch and kibble from the 100 ft. level, and thrown down the passes to the 300 ft. level when finished with. This gave some trouble, as the jumpers falling down the ladderways and ore-passes smashed up the ladders and ore-pass doors, and it would be preferable to either have part of one of the ladderways bratticed off for jumpers only, or to carry old 4 in. pipes up a ladderway down which the jumpers might be dropped.

Some ore should always be left in the ore-pass sides of the chutes, or otherwise, as the stope proceeds upwards and the fall increases, the boxes will be continually broken.

It is also important that a kafir should be kept continually "pop holing" and breaking up large rocks in the stopes, or otherwise these rocks will eventually become jammed in the box holes as the ore is run off.

As may perhaps be noticed from an inspection of Fig. 6, the centre chute may seem unnecessary, and in the ordinary course it would be so, as the block of ground shown in Fig. 1 might with ease and less expense be stoped without it, but the idea of the management was not only to eventually fill up this stope itself with rock from the waste dump, which lies directly above it on the surface, but also all other stoped areas below the 300 ft. level, which had to be done as soon as possible, and for this reason it was necessary to have a pass way through for the waste from the 100 ft. level as soon as possible, and the centre chute answers this purpose.

As to the further applicability of the shrinkage or back filling systems on the Rand and in South Africa generally, it is possible that the back filling system would be especially suitable to many of the mines in Rhodesia and on the Rand where the dip is steep, and where sand, waste rock, etc., is at hand for filling. The shrinkage method might also be applied where the reef is narrow and highly inclined, in which case a modification of both systems might be employed, the waste being first blasted down for filling and the reef resued and shovelled into the ore-passes.

This might be suitable for mines in the Randfontein district. Either of the above methods would give far greater safety than the underhand method for the highly inclined portion of the reef.

(6) *Bibliography.*—For those who require to go more fully into this subject the following short list of papers on the same may be of interest:—

"Ore Mining Methods," by W. R. Crane (Wiley).

"Shrinkage Stoping in Western Australia." by F. R. Rolfe. *Transactions Institution Mining and Metallurgy*, Volume xviii., p. 29.

"Unsolved Problems in Metal Mining," by Professor H. Louis. *Transactions Institute of Civil Engineers.* Volume clxxiv., p. 240.

"The Treadwell Group of Mines, Alaska," by R. A. Kinzie. *Transactions Institute of American Mining Engineers*, Volume xxxiv., p. 334.

The author hopes that the many modifications of the shrinkage or back filling methods of overhand stoping, which are bound to suggest themselves to his hearers, will be fully discussed at future meetings, and his thanks are due to Mr. Palmer Carter, the manager of the Ferreira G. M. Co, for permission to publish this article.

The President: We are very glad to have a paper which will give us scope for discussion

Mr. C. B. Saner (*Vice-President*): In the early days when we were working from the outcrop downwards, the reef dipped at an angle of 70 or upwards. At the Rietfontein Estate the dip of the reef was 80 in places, at the Henry Nourse and at the New Heriot it was also very steep, and the same shrinkage system was adopted. Then there was no question of big batteries, shortage of labour, or "keep the mill going at whatever cost." So that I think Mr. Smith is rather severe on the poor unfortunate old workers on the Rand when he says in this third paragraph he is going to teach them what ought to be done. Of course the Central Rand, on the whole, has got away from the outcrop. We may have these steep dips at Randfontein, but we have not got them on the East or Central Rand now. This shrinkage method means that three quarters of the rock broken has to lie in the stope, an ideal position that few mines can enjoy. I have great pleasure in proposing a hearty vote of thanks to Mr. Smith for bringing this paper before us.

Mr. M. H. Coombe (*Member*): I take exception to a sentence in the second paragraph of the paper just read, wherein the writer describes the system of "shrinkage stoping" as being new in Rand practice. Shrinkage stoping is the oldest known form of extraction of ores from lodes. It is practised all over the world, is largely used in Rhodesia, and was in general practice on these fields in the early days. I was engaged on this same kind of stoping on the old Van Ryn, the Aurora and Spes Bona, and the majority of the old outcrop mines, if not all, had their ore extracted on this principle to supply the first mills erected on the Rand. So by no stretch of the imagination can shrinkage stoping on the Rand be described as *new*. The author deserves credit for his well written paper and his wealth of detail, but I cannot see that the square setts are necessary. Poles "hitched" in "foot" and "hanging" with pole lagging to form the "passes" are just as efficient and much stronger for the rough usage required. The author did not say how many of these setts were destroyed by blasting, and broken by rocks, necessitating replacing. I also seriously question his costs of box installation.

SAND-FILLING ON THE WITWATERSRAND.

(Read at June Meeting, 1910.)

By Edgar Pam, A.R.S.M. (Member).

REPLY TO DISCUSSION.

Mr. Edgar Pam (*Member*): In addition to replying to the criticism on my paper on sand filling, I will, with your permission, describe the various alterations which have proved valuable since June.

With regard to the arrangements above ground, the method of pumping and dewatering, as described by Mr. Powell, is most interesting, and has undoubtedly some advantages over the Silesian bin method. In my opinion, however, the advantages over the more recent plants built for the Rand Mines, Ltd., will not be such as to justify the considerable extra outlay in capital and working costs.

The bins recently built differ from that which I described in the following details: The bottom of the bin is inclined at 30% instead of 25%. The launders running to the shaft are inclined at 20% instead of 12½%. The apertures are 30 in. × 18 in. or larger, instead of 10 in. × 6 in. The nozzles are 2 in. × ⅛ in., instead of ½ in. diameter. With a bin of this description, it is possible to send down sand containing not more than 35% to 40% of water, and if the pipe line would allow it, I am sure that the proportion of water could be diminished still further. At present, however, the considerable horizontal distances along which the sand has to flow, prevents the use of a thicker pulp.

So far as the efficiency and economy of the cyanicide are concerned, the advantages claimed by Mr. Powell do not appear to be of great moment, as our chemists have demonstrated the possibility of using the cyanicide in the tanks, and I am certain that when the sand is being treated regularly, the proportion of chemicals wasted will be very small indeed.

On the question of soluble gold, I regret that I can express no opinion, but should there be any soluble gold in the current sands, it will surely be the duty of the metallurgists to have it extracted before sending the sand to the mine.

Whether the total elimination of slime will prove beneficial is, to my mind, problematical, as the admixture of slime will probably have a binding effect on the sand. I would also mention that on the mines with which I am connected the percentage of slime in the sand is greater than on the Simmer and Jack, and should the dewatering process be used, a plant for depositing the slime would be needed.

On mines where sand filling on a large scale is contemplated, I feel sure that a tunnel driven from the cyanide works to the shaft at about 20% grade will afford the most economical method of transport. The capital outlay on the tunnel might be very great in some cases, but the working and maintenance cost will be practically a negligible figure.

Pipes.—Several members have advocated the use of launders in place of pipes, and there is no doubt that they are preferable where the sand is not to be carried any considerable distance on the horizontal. It must be remembered, however, that in the deep levels the sand will be lowered through one shaft only and from the bottom of this it will be necessary to carry the sand in all directions with very little fall, and in these cases it will be found, I think, that piping the whole way will afford the most elastic and therefore the most economical service. Ordinary unlined black piping has proved very useful where the pulp is not acid, and on the Village Deep and Village Main Reef 75,000 tons were lowered through the same column before any considerable trouble due to wear and breakage was encountered. Californian red-wood lining was found to wear very fast and Jarrah wood, although better, is far from satisfactory. We have ordered some porcelain pipes from Germany and in addition are trying white iron, earthenware, flint, belt and wood pulp lined pipes.

Barricading.—The stope described by Mr. Hughes is, as he says, an ideal one for sand filling, and whenever possible similar dykes or solid faces are used in place of timber. Mr. Johnson has described the method in vogue on the Ferreira Deep of filling against the south side of the drive, and so minimising the amount of barricading required: this method is proving quite successful.

For ordinary stoping widths up to about 10 ft., I think that where barricading is necessary the method of timbering, as already described, will be found to be be the most satisfactory.

While discussing barricades, I should like to point out that although I questioned the advisability of eliminating all slime from the point of view of the final consistency of the sand pack, yet with regard to the barricading and to the drawing off of the water the cleaner the sand the less the trouble that may be expected. Cocoanut matting which, on the Simmer and Jack, allows water to percolate while preventing any sand coming through it, does not act so well on the other mines. The slime appears to choke the pores in the matting almost instantaneously, and neither water nor sand drain through. For this reason therefore the admirable contrivance for drawing off water described by Mr. Hughes has not been successful on other mines, and ordinary launders closed up with wooden slats, as the sand rises, are still in use.

On the Rose Deep Mr. Marquard is building a pack, similar to the one he described, and I think it will be successful in holding back the sand. His idea, which Mr. Laschinger also mentioned, of making the barrier porous will, I am afraid, never be realised with the slimy sand which is being handled. The use of a waste pack inside the timber would be undoubtedly valuable, but I do not think the extra expenditure will be necessary. Mr. Marquard and Mr. Johnson propose the use of ferruginous soil or coarse rock to give cohesion to the sand. Mr. Laschinger favours Portland cement for the same purpose. Any of these three might be good, but the cost of adding them would prohibit the adoption of sand filling on a large scale.

Any quite rigid barricade such as concrete would crack as soon as any weight came on to it from above.

I may mention that last week a portion of the side barricade of one of the stopes of the Village Main Reef was removed in order that the condition of the sand might be noted. The face of the sand remained quite vertical, and a considerable effort was necessary to scrape off any sand with the fingers. The stope in question had been filled three months ago. Both Mr. Raine and myself were quite satisfied that the pack would hold any weight which might come on it. On the Robinson mine the slimes dam method of building side barricades, mentioned by Mr. Marquard, is being used with success, but only a small amount of sand can be added each day and the pulp used must be very thick. In the same mine the sand is being sluiced off the dump down the shaft without any handling.

It will interest you to know that since June 300,000 tons of sand have been lowered into three mines, the Village Deep, Village Main Reef, and Ferreira Deep. Nearly all the mines of the Rand Mines-Eckstein Group are installing sand filling plants, and the daily tonnage lowered will steadily increase from about 1,000 tons per day at present to at least double this quantity.

I do not care to encroach on the chemical section of this subject, but would like to say that we are grateful to Mr. White for suggesting the the use of potassium permanganate, a suggestion which is being successfully acted on.

Finally, I tender my thanks to the members of the Society for the kindness of their criticism on a paper which was read perhaps at too early a stage of the process. There is, I am aware, still much room for improvement in our methods, and I trust that should any member have any further ideas on the subject he will give me an opportunity of testing them.

Mr. E. M. Weston (*Member*): I would like to draw attention to the *Sydney Bulletin* mining notes of October 29th. Its Kalgoorlie correspondent writing regarding the collapse of the upper levels of the Horseshoe Mine states this to be due to the stopes having been filled with coarse sand which will not pack properly, but runs under pressure. He states that with the finer grinding policy employed in later years the lower stopes are filled with what is practically slime which packs satisfactorily.

The President: I am sure we all thank Mr. Pam for his interesting and valuable reply. From our point of view, it is very reassuring.

THE EFFICIENCY OF LABOUR UNDERGROUND.

(*Read at August Meeting, 1910.*)

By Tom Johnson (Member of Council).

DISCUSSION.

Mr. E. M. Weston (*Member*): I must congratulate the author on the consistent manner at which he pegs away trying to hammer a few of the elementary principles of efficiency, by which our efforts should be guided, into our heads which appear to have their reputedly enormous brain development covered with a remarkably hard, tough, and impenetrable cranium. The author is apparently never discouraged, and returns year after year to his labours. When I was younger than I am to-day I had the idea that it was only necessary, in order to get a principle adopted and acted upon, to prove its theoretical truth and to show that in practice it had been borne out in actual results obtained. I fondly cherished the idea that every month the heads of the mining industry spent several nights with wet towels round their heads studying the pages of the *Journal*, and particularly Dr. Moir's chemical equations and nomenclatures. I imagined them taking copious notes and sending instructions out on every side to try this or that new idea. It was, I am afraid, in regard to mining at any rate, a vague delusion.

The author has never been deluded nor discouraged. Many years ago he tried to instil into our minds some elementary notions as to the value and need of artificial ventilation in our mines. By his efforts, and those of others, the facts began at last to be appreciated. In this paper he returns to the charge with regard to mine labour. The running of a Rand mine in the past always reminded me of the efforts put forth to keep some out of date and worn out locomotive going on the tracks. No matter if the tubes were leaky, the feed pump worn out, the bushings required taking up and the wheels were worn irregularly, no matter if she knocked and groaned and hammered and wasted half her power in friction, yet as long as she moved and hauled a truck or two she was kept in service. So with a mine—down below air might be leaking everywhere or wasted in worn out and inefficient machines working with bent and blunted steel. Shift boss might be working against boss, and mine captain at odds with surveyor and sampler. One man might really be undoing the work of another on the opposite shift, and the men working without energy, *esprit de corps*, or security of tenure. Stopes might be 6 ft. on a 2 ft. reef, yet as long as the machine went, though disabled and hampered by friction and lack of co-ordination of its parts, as long as so many trucks of something came every shift from so-called stopes, and as long as so many skips of so-called ore came to the surface to keep so many iron headed rods hammering at a pre-arranged and fixed speed seven days a week year in and year out, everything must be well. I wonder if anyone really believes that this policy of running the mill seven days a week has always paid in the past I am no strict Sabbatarian of the old school: but I was very much struck by a remark made by a mine captain, who died recently, having given his life to his work. "I wish there was no Sunday." What he meant was this,—the thought of and struggle all the week regarding the necessity of keeping those remorseless stamps pounding on something for an extra day at the end of each week poisoned all the work of the week, filled it with an over-worry, over-care, and tended to rush and inefficiency.

Regarding the prevention of dust, there is no doubt that for drives the water jet is at present the most useful appliance. The Leyner drill has been very greatly improved and simplified since it was last tried on the Rand, and now holds at least three speed records in driving adits in hard ground (granite) in America. I admit that the upkeep of steel is expensive and troublesome, yet I am persuaded that with a little care it could be used with economy for rising (especially on the flatter dips of the deep levels.) It and other hollow steel drills will, however, never be a success until pure neutral water is sent underground for their use, and trained labour available for running them. The Leyner air and water system down hollow steel is the only rational method of introducing water into holes while they are being drilled. A practical miner, of course, knows very well that in working with a piston drill the trouble in a flat upper hole is to get the broken rock out of the hole. In America the miner keeps a thin wire scraper always working alongside the drill in the hole, or uses steel with lugs between the cruciform sections in order to hasten the discharge of the drillings.

I am afraid that any scheme for forcing bagging around the mouth of the hole will prevent and retard the emission of broken chips and will be neither economical nor effective. With hammer drills working on steep upper holes the question is different, and Dr. Aymard has been long ago anticipated by devices such as sponges held against the mouth of the hole by springs.

Regarding Petersen's respirator, one can only remark that as there is always a danger of an unforeseen generation of large amounts of poisonous gas, for example, carbon monoxide in compressed air It seems, to say the least of it, just a little unwise to provide for the certain death of every machine operator in the mine in the event of a compressor explosion. Besides, are we going to allow the unfortunate native to continue to swallow dust as in the past? If we are not, I see a pretty picture of a machine at work in a rise with four or five masks with their attached pigtails getting hopelessly tangled up, and the supply hoses cut or broken by jumpers, hammers, etc., being thrown on them or by being trodden on.

Efficiency in mining can be only increased by educating the workers. Five years ago I urged the necessity of educating the rock drill workers, and pointed out the benefit that would occur from a practical training and a sound knowledge of some of the simpler laws regarding mechanics, explosives, and blasting. My recent experience goes to show that the average adult miner on the Rand has not the slightest belief that he could increase his earning power by any such means. Education of this sort he neither appreciates nor desires, though there are shining exceptions. The problem of the future is to make this education compulsory to the beginner and desirable and attractive to the older miner. Ignorance of important laws governing efficiency in the use of explosives is almost universal. Take the question of the relative efficiencies of machine and hand stoping. I have several times dealt with this question before the Society, and my excuse for wearying some of you is that I have come to believe, with the author, in the policy of hammering it in.

Mr. E. J. Wiseman in the September *Journal* seems worried about the question. He is quite right in all he says, but owing to ignorance of that law of blasting which states that the quantity of explosives necessary to break a hole varies as the square of the burden on it, his explanation is incomplete. A practical miner writing to the *South African Mining Journal* also appears to have somewhat confused ideas on the subject. He suggests that the conical shape of the machine hole tends to reduce the efficiency of the explosive causing the tamping to be blown out as one reason, and the fact that misfires or badly-placed holes in a bench must ruin holes behind them, as another. With regard to the first cause suggested it need only be said that provided the confinement of the charge is good, *i.e.*, the cartridges have been properly placed in the hole, and that a moderate amount of tamping has been employed, the high rate of detonation of blasting gelatine enables it to give quite as good results in a machine hole as in the more cylindrical hand hole.

The loss in waste of explosives due to badly placed holes in benches is merely the penalty of inefficiency. There should be no badly placed holes, and with a careful miner who understands what he is doing and who starts all his holes himself, there seldom or never are any. Hand stoping is bound to be more economical in explosives used, though not necessarily in work done, for the simple reason above stated—that if, say, in a stope of equal height it takes one plug of gelatine to break a hole with a burden of 1 ft., it will take not two but four plugs to break a hole with a burden of 2 ft., and nine to break a hole with a burden of 3 ft. This law is, of course, modified by the presence of heads and partings, but in practice I found it to hold true on the average. As I pointed out five years ago, the right understanding and appreciation of this principle at once shows us the lines along which our efforts in reducing the consumption of explosives in machine stoping in narrow stopes must run. We must remember first that with piston drills the actual drilling time bears a small proportion to actual working time, the rest being taken up by erecting and moving the machine. Mr. Wiseman suggests drilling one hole benches, but he will find that even if the miner's work does not suffer through the time he has to lose from the supervision of his machines in order to erect bars on other benches, it will take at least ten minutes, as I have proved myself, even in a fairly flat stope of 30° to perform every move, and four or five moves during the shifts means 40–60 minutes running time lost, which means at least one good hole lost, so his remedy is as bad as the disease. We must also remember that it is the starting and first 2 ft. of a hole that takes the time in boring, as the drilling speed is roughly proportional to the amount excavated. So that once we have our machine erected and a hole started it must pay to bore it as deep as possible—even if the diameter at the end is very small—if it be rightly placed. Mr. Wiseman thinks because he sees such a large number of 12 in. and 18 in. sockets standing that it is a mistake to drill these long holes, that gelatine often cannot be inserted in them, and that even if it can be, they will represent the diameter of a hand hole only. If this were the real trouble it could be got over by "chambering" the hole; for instance, it would be quite easy for the stoper to fire a plug of gelatine in the bottom of his back holes when he fires his front holes; but if you are going in for "bulling in" stopes with free faces limited in height, you may as well bore only short holes, as long bored holes will rarely break to the mouth but will "bull ring." This method is, however, quite useful sometimes.

In machine stoping we must try as far as possible to combine the advantages of hand stoping with the output of machine drilling, for which we are prepared to pay some few pence per ton extra in explosives. To do this, first we must get the best work out of our machines in the time allowed, and can only do this by drilling long holes, and we must drill long holes in both narrow and wide stopes. This again I contended for five years ago, and it has only recently been admitted to be good practice. We must avoid any possibility of the first hole in any pair fired being held up. To allow of this we must acquire judgment by practice, and we must work with some knowledge of the principles of blasting.

We must have some idea, and a very good idea also, of what work our first hole is going to do. Unless we rightly judge just how much rock this leading hole will break it is clear that either our second hole will be too heavily burdened or the charge in it will not have half enough work to do. It is just here that half the inefficiency of machine work comes in. I have tried to make three sketches to illustrate this system of stoping. I claim no originality of practice in drilling zigzag holes, though I have always tried to show the reason of the practice, and the absolute necessity of it in machine stoping in narrow stopes. In very wide stopes the top and bottom holes are further apart and do not influence one another quite so much.

I evolved for my own convenience the system of boring 6 ft. and 7 ft. holes with small drills, finishing with a chisel of ⅞ in. steel, in stopes of from 36 in. to 48 in. high, in the manner shown, and have tried to show how this method complies with the law of blasting already stated, in order to gain maximum economy of explosives and maximum drilling time and efficiency for the machine.

Fig. I. represents a bench one may see any day bored in the stopes of a Rand mine: the dimensions are, of course, relative and alter with varying circumstances such as hardness of rock and presence or absence of planes of weakness on foot

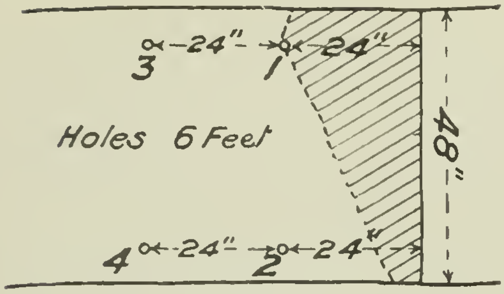

Fig. I.

and hanging wall. Here instead of showing Mr. Wiseman's 12 in. and 18 in. sockets I show 6 in. and 12 in. sockets. It will be seen that hole 1 has torn nearly all the burden off No. 2, which has wasted half its charge. Some miners have some dim realization that this occurs, and give No. 2 hole a slightly smaller charge, obviously confessing that they have bored the hole with too small a burden.

Fig. IA. shows a plan of the bench.

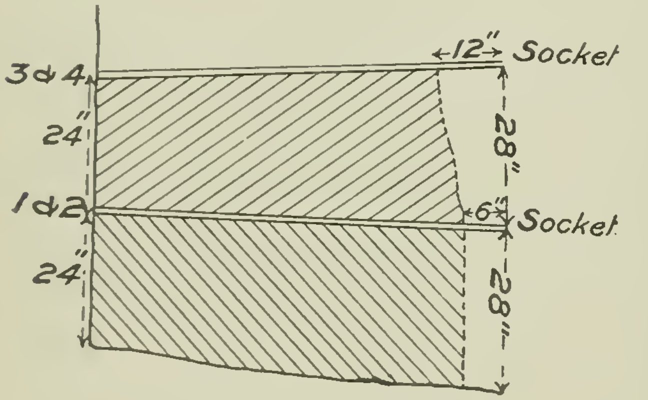

Fig. IA. (Tons Broken $=7\frac{1}{4}$).

In Fig. II. we have shown the correct method of placing the holes. If hole 1 has a burden of 24 in. it will "slab off" the rock as shown, and hole 2 can be given a burden of as much as 36 in. I will admit that seeing a bench bored thus appears madness; but I have often proved that it is only common sense. Why, it may be asked, if hole 3 in Fig. I. has only 24 in. burden can you give hole 3 in Fig. II. 28 in. or 30 in. I reply, because for one thing the face of the bench from 1 to 2 presents a larger free face for the hole to break to and the hole is also partially under cut by hole 2, and for the same reason hole 4 can be given 30 in. or more. On looking at the plan Fig. II., it will be seen that all these holes are bored looking out slightly, *i.e.* having several inches less burden on the bottom part of the hole where there is less explosive than on the centre of the hole where there is more explosive. Boring with large drills and finishing off the hole with a chisel of $1\frac{1}{8}$ in. steel, it is usually sufficient in a stope over 40 in. to merely keep the hole strictly parallel to the face, where the ground is tight. Where lines of weakness occur in more broken ground this rule may be modified. For the same reason the position of the holes and the extent to which zigzagging of holes can be carried depend on the toughness of the rock and the presence of lines of weakness on the foot and hanging wall. Were the rock solid granite breaking in the proper crater form, zigzagging holes would have little value; but the banket rock of our reefs more or less tends to "slab off" along planes at right angles to the walls, and every machine stoper should on first firing in a stope, fire the front leading hole only of each bench and go back and view the result. He would be surprised in every case. We will suppose that on trial the bench drilled on system (Fig. II.) with holes 6 ft. long gave satisfactory results, the holes breaking to the bottom when their diameter at the bottom was little over 1 in. We then ask ourselves: if it is as easy to blast a 6 ft. hole as a 4 ft., why not drill a 7 ft. or 8 ft. hole finishing up for $2\frac{3}{4}$ in. drills with a $\frac{7}{8}$ in. chisel, and a hole about $\frac{15}{16}$ diameter on the bottom? (Fig. III.). We may take it as a rough guide that in an extended charge occupying a borehole, each cartridge must be responsible for moving the burden in front of it. Then the first precaution to be taken is to see that the small diameter hole at its end has only just sufficient burden to allow it to break the rock in front of it, and this can be done by making the holes look out. It may be objected that by this one would always be tending to narrow up the face of the bench and to lose benches. I reply first that

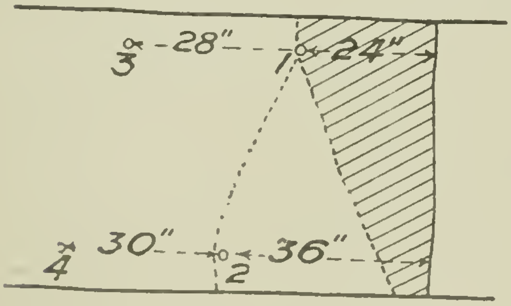

Fig. II.

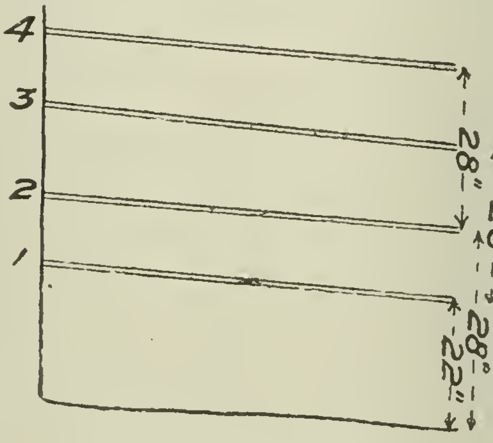

Fig. IIa. (Tons Broken $=9\frac{1}{4}$).

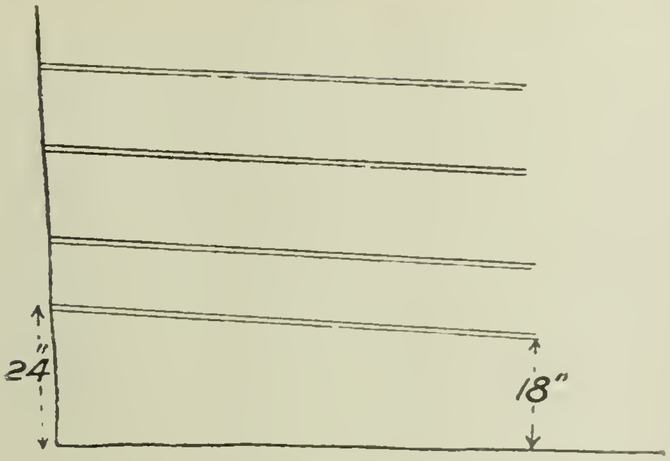

FIG. III.—Seven feet holes finished with ⅞ in. steel, loaded in end with ⅞ in. blasting gelatine. (Tons broken = 9¼ + 1¼ = 10½ tons.)

five holes can often be drilled and the fifth hole can be used for squaring up. I always used the short shift on Saturdays for putting in two hole or three hole benches, to square up benches and keep the stope in shape. Hand labour should also be used in machine stopes. With holes bored as shown the trouble of "bull ringing" arises unless the charge is spread over the hole. All I wish to say on this matter is that if the practice of dividing the charge into two portions each with its separate detonators and fuse is objected to as being dangerous, then if the principles on which this method are founded are correct, and lead to economy and to maximum efficiency (as I believe they do), electric or some other system of firing must be resorted to.

At the Robinson mine with an 8 ft. high stope it is found possible to drill holes 8 ft. to 9 ft. long finishing with 1¼ in. chisel, and to fire them without leaving any stumps at all. The efficiency tonnage per machine shift broken is something over 20 tons.

The stope drill contest started off with the regulation (the wisdom of which I queried at the time) that the length of holes bored must not exceed 48 in. It was soon found in practice that the only way to break ground was to increase the length from 5 ft. to 6 ft. During this trial the miners might just as easily have used 7 ft. holes, and the results gained would have been still more favourable as compared with hand drilling. This, again, I had advocated in a paper written in 1905. It apparently cost the mining industry five years and £20,000 to learn what they might have found out by studying the *Journals* of this Society. In these I have always contended first that any relative inefficiency with hand stoping in small drill work was due (1st) to short holes being bored, (2nd) to easily remediable alterations in design of the drills in the direction of making them stronger while using practically the same types then in use, (3rd) to low air pressures at drill due to bad design, want of supervision of air pipe lines and overworked compressors, (4th) that men must be taught their work under practical conditions in underground schools.

Mr. Johnson is always insisting on the necessity of shift bosses being themselves efficient in the work they have to supervise. He will be pleased to hear that in the mines controlled by the Consolidated Goldfields of S.A., a circular was very recently sent round laying down the rule that no one was to be promoted to shift boss who could not show good cost sheets for contract work in machine and hand stoping. The keeping of proper records aids efficiency.

To Mr. C. D. Leslie belongs the credit of evolving a system of records by which the actual efficiency of the work done in any mine can be very fairly judged, and that too without adopting any new fathomage system. By its means the actual cost of obtaining the valuable ore, and its relation to waste rock mined, and to costs and values of total rock broken and to milling ore are easily ascertained every month. This system or a variation of it is, I believe, being largely adopted by all the groups.

Mr. W. Cullen *(Past President)*: So far as I can recall I have never taken part in a discussion on a purely mining topic before, and the reasons for this are obvious, but Mr. Johnson's paper raises many points on which a mere tyro can speak. As far as I can judge, however, he has made his point regarding contract work with machines, but whether it pans out in practice only he and other practical men can tell. Judging from analogy his arguments are sound, and it does seem that in his hypothetical case both the best men and the company are done down. This is, however, only one phase of efficiency or inefficiency and since that is the universal doctrine, at the moment it behoves us to look around and see whether we really understand what we are talking about. In what I am going to say I think I may rely on Mr. Johnson's support at any rate. I should like to preface my remarks by saying that my talks with miners and others during the course of the recent election campaign have opened my eyes considerably to certain phases of a subject which may be generally termed "workmen's grievances." I purposely exclude "silicosis" and "ventilation," but to my mind it is far [illegible] to talk of getting efficiency in mines when *the working conditions are known to be bad.* I have been down many mines but I know that if it were my lot to do physical work in some of them my efficiency would not be 10%. And then when we discuss this question in its broadest aspect we think only of white men, but surely the same thing applies to natives. I heartily endorse Mr. Penlerick's statement that the ventilation installation *of the E. R. P. Mines, costly and all as it was* has already repaid the outlay. If one recalls the speeches of the chairmen of the different mining companies, one cannot fail to be struck with the constant references to the native, his number and his efficiency. Although in a sort of academic way we look upon the native as the one essential factor for economical mining, I am afraid that we frequently forget that what holds for the white man holds equally for the native. Neither a native nor a white man can work on an empty stomach, and good treatment is as essential for the one as the other.

I have some knowledge of what can be got out of the native under different conditions, and I have no hesitation in saying that on those mines where the management takes a personal interest in the native, where precautions are taken to see that he is treated as a human being and not as a dog, the efficiency is high and the numbers keep up. No! the native is our great economic factor: without him few of us would be here, and I venture to suggest that the subject forms one of the greatest unexplored chapters in the great subject of efficiency.

Mr. Johnson says rightly: "Keep men for long periods on the property, if possible, so that they get to look at the place as a home, they will then take an interest and pride in keeping the property to the front. Men thoroughly used to the mine will do more than newcomers."

Surely this is axiomatic, but I am afraid that on many mines the converse holds. As far as the native is concerned there is no immediate prospect of our securing continuity of employment, but Mr. Johnson's principle was our stock argument for getting the Chinese and keeping them as long as we could when we did get them. Here again the same principle must apply to white labour. We constantly talk about attracting a large settled white population, but on the mines are we really going about it in a proper manner! One of the grievances which I hear from my mining friends is the uncertainty of tenure. I don't believe the responsible heads really know how much unnecessary shifting there is among our working men and what an amount of dissatisfaction it causes.

As far as I have been able to ascertain a change of management frequently means wholesale changes in every department of the mine. Surely this is unnecessary. In the case of married men, settled on the property, if shifting must take place, let the men have a month's notice instead of the general one of 24 hours. Every praise must be given to certain mining companies for the excellent accommodation which they give their employees, but that is not all, as I have just indicated.

The question of the purchase of stores such as coal has been referred to frequently in our proceedings so I must say nothing on that subject, but quite apart from the different makes of explosives I am convinced that money, and a lot of it, could be saved by making a more intelligent selection of the explosive. To give an instance, practically only two gelatinous explosives are used on these fields—gelignite and blasting gelatine. An intermediate grade has been introduced from time to time under various fancy names, but at the moment practically none is used. I am convinced that there is a good wide field for an explosive of this nature if only the trouble were taken to try it. It would be a trouble, I admit, but nothing is achieved without it. It has been my constant experience to see an explosive of lesser strength than blasting gelatine doing better work than blasting gelatine itself. One cannot predict these things nor can one dogmatise about what is most suitable for a certain rock. One must simply try. Then on the subject of rock-breaking there surely must be right and wrong ways of doing it, but while thousands upon thousands are devoted to surface experiments, one hears very little about experiments on rock-breaking. Of course I know the answer to this, that it is mostly contract work, and a bad miner will draw very little at the end of the month, but that is only a very partial answer to a very important question. In this connection I do not know whether Mr. Johnson's scheme of tuition is feasible or not, but that something of the sort is necessary all are agreed. The best way, however, is to catch your man young, but so long as our mines have the reputation of reducing a miner's normal life to the present low figure this will be difficult of attainment.

Mr. S. Beaton (*Member*): The author has dealt with a great many points in his paper regarding small economies in our mines. He lays special stress on the introduction of young inexperienced shift bosses and foremen. These men must be allowed to survive and grow older by means of good ventilation and dust-allaying appliances wherever dust is being formed at stations in stopes and drives. They will then have a commercial value that will make it hard for younger men to oust them from their positions. At present there are not enough technically trained experienced men to go round, which fact allows a number of young and inexperienced men to attain positions for which they are not qualified. Health conditions govern the question of the efficiency of our underground men, white and black, by improved proficiency and ability to make a more sustained effort.

The author has started to economise at the wrong end, although a bit may be done there. The method of working pursued here is not conducive to economy: our method being an elaboration of the method evolved by what are termed the amateur miners of the early days, which method suited the conditions then prevailing, as labour was plentiful and mechanical appliances at a discount. With the vast improvements in mining machinery and not so much labour this method ought to be abandoned and a more efficient one substituted to take advantage of lower dips and the cheap power which is now available at the mines.

The following method of longwall working can be adapted to most of the mines of the Rand: From the main shaft drives ought to be driven under the reef on lines, all necessary bends having a large radius for haulage purposes. These drives should be sufficiently wide to allow of two tracks being put in. Drives should not be nearer to each other than 600 or 800 ft. on the dip of the reef, and even a greater distance may be employed. At 300 ft. on either side of the shaft crossents would be put to intersect all the reef, and at 600 ft. in either direction (east and west) other crossents will be driven. From these crossents intermediate shafts will be sunk on the various reefs, only clean reef being mined, all waste rock being blasted down and hauled separately, waste being used to pack in different parts of the mine. Those shafts would be sunk to the level below say 800 ft., when they could be used as self acting inclines. At 200, 400, 600 and 800 ft. on the intermediate shafts breast stopes will be started 30 ft. to 60 ft. wide depending on the condition of the hanging and other local considerations in the mine: at each breast a switch would be laid to each side to allow of the trucks being run directly into the stope, where a loop would be put in allowing a supply of empty trucks in each stope so that the shovelling gang may be employed continuously whilst hauling from the other breasts. An electric hoist can be employed for hoisting, the empty trucks hauling down the rope which is attached to the loaded trucks, and when hauled up the trucks are detached and put on to the main haulage and taken to the shaft. In working, the breast would be taken in a distance of 300 ft. when it would meet the one coming in the opposite direction. They would then be cleaned out and filled in with sand, the rails be taken up and all timber drawn. When the stope is full, another breast would be started, the track being laid on the sand and the switch shifted up to suit. The sand should be filled in on night shift so as not to interfere with the ordinary working of the mine; good packs could be built alongside the shaft to retain the sand in position. When only a thin pillar is left between one breast and the one above, this pillar may be stoped home towards the shaft leaving all the sand behind. The output of the mine would almost be obtained on one level with two reefs.

The following advantages may be claimed for this method of working:—

I.—Lower developing costs.

II.—Lower shovelling and tramming costs, as all broken ground would be blasted down to the track and no manual tramming to a greater distance than 300 ft.

III.—Better supervision, work being more concentrated.

IV.—Simpler ventilation and drainage.

V.—New mines would reach the producing stage in less time than at present.

VI.—Greater safety, no great stretches of worked out places being left to endanger the whole mine and surface works.

VII.—No pillars of reef need be left and no cost incurred in raising boxholes.

VIII.—Less timber and timbering.

IX.—Sand would be held safely in position and no mud rushes could take place in a wet stope.

X.—The output from any stope would not be limited by the number of shovelling boys it is possible to crowd into a stope.

VENTILATION AND HEALTH CONDITIONS ON THE MINES OF THE WITWATERSRAND, WITH SPECIAL REFERENCE TO THE VENTILATION SYSTEM OF THE EAST RAND PROPRIETARY MINES.

(Read at August Meeting, 1910.)

By S. Penlerick (*Member.*)

DISCUSSION.

Mr. Tom Johnson (*Member of Council*): Mr. Penlerick's paper shows plainly the great benefits to be obtained from increasing the ventilation of our mines. It is to be hoped that the experience of the East Rand Proprietary Mines will give others the encouragement necessary to try to better the ventilation of their own mines.

Although artificial ventilation is what should be aimed at, there is no reason why those who cannot spare the necessary capital to instal fans, etc., should sit down and say there is nothing they can do to better the ventilation.

As the author pointed out, adequate ventilation does not depend solely on the quantity of air going into the mine, but greatly on how the air is coursed and distributed through the mine: as a proof of the above, the more we course our air the less the quantity circulating. Much can be done quite cheaply by putting up brattice sheets in the entrances of the upper drives between shafts, and an odd door or two, as shown in the paper, so as to conduct the greater part of the air to the lowest point of the workings, instead of letting it get away to the upcast through the upper drives. As the author points out, and as I mentioned in 1903, a standard of purity of air is better than a standard of quantity per man, but more air per man is needed in wide stopes than in narrow stopes, irrespective of quality.

The author does right to lay stress on the water question. As Mr. Judge mentioned, it is not only at the holes being drilled but in all dry and dusty places, that the water is needed, for after all is said and done, what is the percentage of dry holes drilled in a producing mine? It should not be 10%. To send cool air through a warm mine without watering the mine would only aggravate the dust question, for the air as it got heated on its travels would pick up quite a quantity of water, making the mine drier and dustier.

The Rand is not the only place that is troubled with a dust question, many collieries have a more dangerous dust than ours. Here the dust danger is more or less in the handling of the man most concerned, but in many collieries one man may either ignorantly or carelessly endanger not only his own life but the lives of every one in the mine: still in the United Kingdom the death rate from fire damp and coal dust explosions has been reduced from 0·65 to 0·1 per thousand during the last 35 years. If all concerned were to co-operate we ought to pull down our death rate much faster than this. Phthisis is not a necessary adjunct to mining, so if each one concerned will only do his little bit the death rate would be considerably lowered.

There is going to be a great disadvantage in using water in the hotter mines of the future, and I believe we shall have to put cooling apparatus in the workings to cool and dewater the air somewhat, so that it will not be saturated and at a high temperature before leaving the mine; this has been talked of in colliery circles for many years, but I do not think any installation has been put in yet.

The E.R.P. Mines are fortunate in being able to set a shaft aside for purely ventilation purposes. It is very desirable that one shaft should be used solely as a ventilation shaft, but every mine will not be able to do this. It is not a heavy job for an outcrop mine to fix up, but it is an awkward one for some of the deep levels, more especially the mines with all compound shafts. If the deeps cannot come to an arrangement with the outcrop mines to compound the ventilation, then the mines with compound shafts have a little problem to solve. With our present methods of hoisting and handling rock, men and material, we cannot very well fix the the fans on the surface. They will have to go underground, and where are they to be put?

As to the bad effects of compounded ventilation of two mines, I do not see where there should be any trouble; if the whole of the area of the two mines was used as one mine there would be no talk of exhausting bad air into the outcrop portion. The exhaust bad air from the two mines with a fan will be better than the exhaust air from either of the mines with only natural ventilation. I should be glad if the author in his answer would give us the speed of the fans; how and where the water gauge was taken (that is how far from the fan, and in what direction was the pipe of the water gauge pointing), also the H.P. of the prime mover, size of opening in the fan, temperature, and the natural water gauge. I notice from the figures on the sketches that with air at 63° F. the Sirocco is giving up about 22 h.p. to velocity; this is too much. I think it could be brought down to 6 h.p. or 8 h.p. This 22 h.p. represents about 16% of the water gauge. The Barclay, on the other hand, is only giving 6% of the water guage to velocity, which is not because the Barclay is a better fan, but simply because it has a much larger *evasée* chimney. I would recommend the author to take down the Sirocco chimney and rebuild it, giving it an area of, say, about 200 sq. ft. and save 14 h.p. or 15 h.p. I make the present areas of the chimneys from the figures on the sketches to be say, 100 sq. ft. for the Sirocco, and 170 ft. for the Barclay. If these two fans were tested it would be seen that the Sirocco is heavily handicapped because of its chimney. Another thing in fan testing, is that it is quite wrong to take the all-over efficiency of engine and fan for a comparison, as is so often done; efficiency of the fans should be taken alone.

Last year the late Mr. A. H. Stokes,* Inspector of Mines, England, read a paper before the Midland Counties Institute of Engineers on water gauges and velocities. He found by experiment that by putting a bend on the end of the pipe leading to the experimental water gauge, that with a normal height of 2 in. on the gauge on the separation doors, by manipulating the bend he could get variations in height from 0·2 in. to 3·0 in. He found that with the same speed of fan and volume of air that if the area of drift varied so did the height of water gauge. It can be seen how important this is, for water gauge varies as velocity, and velocity varies nearly as area, and the h.p. in the air equals quantity multiplied by water gauge divided by 33,000, this last being used to get the efficiency of the fan. As fans become more general, and that will be soon, we shall have some of the same troubles in testing as they have gone through in other countries, for up to now I do not think a decent comparable manner of testing fans has been evolved; at least it looks like it from the published tables of fan tests. As an example of the curious things we hear of in this connection, a case was mentioned in the discussion on Mr. Stokes' paper, where a new fan was put in to replace a larger diameter fan in the same fan pit. The new fan was to give 62% efficiency, but on trying it, it only gave 50% efficiency; after the fan pit was altered to suit the smaller fan the efficiency was increased to 60%, the water gauge rose from 2 in. to 3½ in., but mark you, the air in the mine remained the same. It is things of this kind that we shall have to be careful of when buying and installing fans, or we shall be doing in this case what we are doing in other cases at present, that is, paying for paper efficiency. I notice that the decrease of temperature is only 2° or 3°. Is this an error, or is it because the difference of air temperature and rock temperature is small?

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* Trans. *Institute of Mining Engineers*, Part 1, Vol. 38.

I am afraid that we shall not be in a position to carry out the author's recommendation contained in the last paragraph of his paper, desirable as it is. In the future deep levels, as members know, I hold the opinion that we shall have to move our faces much faster than we do at present, both on the score of safety and cost, and before we get well set on single shift work we shall have to turn to double shift again in the deeps. This is where I see the greater need for artificial ventilation, for although we shall get larger quantities of air by natural ventilation in these deeper shafts owing to their depth, the quantities will not increase so fast as the need for them.

In conclusion, I must say that the E.R.P. Mines deserve, and will have, the thanks of all for piloting the way towards mechanical ventilation, and the author also for the manner in which he has brought the matter before us.

Mr. T. Donaldson (*Member*) on behalf of Mr. W. Cullen submitted the following contribution:

Mr. W. Cullen (*Past-President*): This paper — as has already been remarked —is perhaps one of the most important which has ever been read before our Society, and it is to be hoped that all the other mining companies will emulate the excellent example set by the E.R.P.M., by installing systems for improving the ventilation. It will come as a surprise to many that the total cost is so very low, but I can well believe the author's statement that it has already been recovered by increased efficiency alone. Quite apart from this aspect of the case, however, it is a moral obligation on the the part of those who control the industry to see that no further time is lost. Of course, it may be claimed that nothing could possibly be done so long as the "Mining Regulations Report" was under consideration, but now that this report is issued, we can see that hardly anything new has been brought to light. Indeed, the report is mainly a compilation (though a most useful one) of information which for the greater part was already public property, with certain deductions from this compilation, and finally certain recommendations made naturally with the idea of improving existing conditions. In case of any misunderstanding I wish to say that in the main I agree with the findings of the Commission, but it does seem a pity that such an important report was delayed for so long a time through circumstances which were apparently unavoidable. This is the only word of criticism which I propose to make regarding the report itself, but it is worth while in passing to note the attitude of the press and the public. The press to my mind did its duty nobly by endeavouring to arouse the public to a sense of the magnitude of the issues at stake, and let us all admit frankly and freely that the public responded equally nobly. Unfortunately, however, the report was issued during a political campaign—a most unwise procedure in my opinion—so that the subject-matter assumed a political aspect: but by this means it became more widely known. To-day nothing is heard of it. All those fine humanitarian instincts which were aroused at the time of the publication are "as they were:" at least I think so. The indifference of the Rand to the loss or the value of human life is to me astounding and incomprehensible. Perhaps, however, I am altogether wrong and it may be that plans are even now being prepared for the installation of artificial ventilation on every mine. If so I shall be delighted: but I "hae me doots" and for the following reasons. Years ago Drs. Macaulay and Irvine initiated what I shall briefly call an Ambulance campaign. They showed us clearly what was required of us: we all admitted that many things were left undone which ought to have been done, but since these days can any one of us, managers, officials and workmen, say that the campaign has made much progress? I say unhesitatingly, relying on my own experience alone, that the want of progress has been simply appalling. On some mines, which I have visited recently, prominent officials told me that they did not know whether there was any ambulance equipment on the mine, and others made the same reply about the oxygen cylinder, asking at the same time what its uses were. Now one can be excused if the matter involves a large expenditure or troublesome organisation but it involves neither. At the Dynamite Factory 70% of the employes have gone through a course of ambulance instruction—it is a condition of employment and has been ever since I was appointed manager—every department has a complete set of ambulance appliances, and it is the duty of the medical officer and the official in charge to see that they are always in order. Then again with regard to safety helmets or similar devices, mishaps within the past few months have demonstrated again, if that were necessary, that there should be at least two on each mine. But they would be of no use unless the men were practised in them and drilled in rescue work. Indeed, they can, in the wrong hands, become a veritable danger.

At the present moment there is sitting in England a Royal Commission on Mines, which has issued two reports from which I propose to quote freely, as they have an intimate bearing on our conditions here. The first one, issued in 1907, deals almost exclusively with safety helmets, their uses and abuses. The physiological tests, conducted by Dr. Boycott, are extremely interesting, and I commend them to our members. It is true that the mines referred to by the commissioners, when discussing the subject, are coal mines, but the accidental ignition of a case of explosives would furnish an analogous set of conditions to those which the commissioners had in their minds.

They say *inter alia*, page 7. "With the "exception of a few isolated instances of "the use of breathing appliances, little "attention was paid for some years to the "question in this country, and mining "engineers aware of the imperfections of the "existing appliances were apt to regard "discussion of their introduction into general "use as of theoretical rather than practical "value. Within the last decade interest in "the matter was revived by the construction "in 1900 of an experimental gallery for "testing life saving appliances at Altofts "Collieries in order to accustom men to "the use of the apparatus in conditions "such as are likely to be met with underground."

Page 8. "In some continental countries greater "attention has been given in recent years to "the use of breathing appliances In "Austria this provision has been made "compulsory, in Germany, though no "regulations have been made by the Government on the subject, a great deal has been "done voluntarily and many of the larger "collieries are provided with sets of "apparatus."

We have all of us fresh in our minds that terrible disaster at Courrières, and the excellent work performed by the German rescue brigade from Westphalia.

Then again, page 11. "We think that the "risks attaching to the use of the appliances "can be reduced to a minimum by a proper "system of training, and we desire to lay "particular emphasis on this point. In the "first place it is important for purposes of "training to select men who are most suited "for special work of this kind on the ground "of general intelligence, physique and "temperament. The men should then be "given a thorough knowledge of the construction and working of the appliances "and should be accustomed by degrees to "wearing them, at first in an experimental "gallery and afterwards, when they have "attained proficiency, in actual operations "underground."

The report then goes on to recommend the establishment of rescue stations which would serve either one mine or a group, but that is a detail. I think myself that each mine should have two at least, and the necessary men to man them together with reserves. The expense would be infinitesimal, and I do not believe for one moment that it would be grudged by any one in authority.

The second report issued last year deals with a very large variety of topics, but rescue and ambulance work—especially the latter—bulk very largely. Incidentally two of the commissioners record their impression of a visit of inspection to some of the German rescue stations. At the Shamrock, one of the largest collieries in Westphalia, it appears that the rescue corps consists of 31 men and 50 officials, and that during 1906 there were 163 practices with breathing appliances, which lasted on an average two hours each. Only on two occasions were the practices interrupted by small defects in the apparatus, which shows that they have now attained a high degree of perfection. Of the 67 officials on the mine 62 are fully trained in the work of rescue. The commissioners also record that since their last report the provision of breathing appliances has become compulsory in both Belgium and France.

I do not believe that rescue corps on anything like this scale are necessary here, but all must admit that something wants doing. It did occur to me that if the expense were such a bugbear as it appears to be, that the members of the fire brigade might be trained for this rescue work, seeing they are already accustomed to the use of safety helmets and ambulance, but there are obvious objections to this course not the least of which is the men's want of experience of mining conditions, for there is nothing more demoralising than feeling lost in the bowels of the earth. In this connection I may say that I have suggested to Professor Lawn and the Senate of the School of Mines that every mining student should be made to go through both an ambulance and a rescue course—the latter to include practice with breathing appliances. The Board of Trade have recently issued an order requiring that all candidates for the certificate of master or mate should possess ambulance certificates. How much more is it necessary with our mines? I shall avoid being discursive and therefore only quote further the recommendations of the 1909 report, under this heading, which recommendations will no doubt soon be embodied in a legislative act.

Page 209, par. 68. "*Use of Breathing* "*Appliances.* Schemes for the systematic "provision of breathing appliances and the "training of men in their use should be "pursued with greater energy.

67. "Every mine should be provided either "with a properly trained brigade of its own "or to have the right to call for a sufficient "number of equipped and trained men from "a rescue station."

69. "*Ambulance Work.* The number of men "qualified in ambulance work is compara- "tively small, and there is room for con- "siderable improvement both in the training "of the men and the provision of appliances. "There should be co-operation between "the colliery owners and the St. John "Ambulance Association. The officials of "a mine, including managers and under- "managers, should be required to possess "ambulance certificates."

70. "Ambulance appliances should be kept "underground as well as on the surface. "Proper arrangements for conveying injured "men to the hospital ought to be made at "all collieries.

71. "The Secretary of State should have "power to make regulations specifying the "nature and number of appliances to be "provided in proportion to the number of "men employed, the places where they are "to be kept, and the number of men "required to be trained in their use."

72. "*Organisation of Rescue Work.* A great "deal could be done in systematising the "arrangements necessary in the event of a "serious accident. We suggest the appoint- "ment of committees of the colliery owners "in each district to provide for the erection "of rescue stations, the training of men, "formation of brigades, etc., and we also "recommend the issue of a handbook con- "taining instructions in rescue work.

73. "Instructions to workmen in such matters "should form part of the ambulance train- "ing, and the preparation of a text book for "the use of workmen in such classes would "be an advantage."

Of course all these recommendations show that we are after all not really so very far behind the Old Country, but it must be borne in mind that the conditions are vastly different. Here we live in a new country, we have started afresh, we are concentrated in a large area, and above all we consider ourselves very up-to-date. In the Old Country traditions, prejudices and trades unions have all to be taken into consideration when any change in working conditions is proposed, consequently many necessary reforms are neglected. For instance, take the mere question of change houses—the provision of which by the way is compulsory in Germany as it is here. In Great Britain there is a distinct prejudice against them, although here and there one finds them. To anyone who has lived in colliery districts as I have, and known the revolting conditions under which the men live, the small uncomfortable dwellings, it seems incredible that they should prefer to return to their houses covered with dust and grime rather than leave it behind at their work. Such, however, is the case. Turning now to the Report of the Mining Regulations Commission we find that both ambulance and rescue work are given great prominence, and the recommendations are on all fours with those of the English Commission and in the Draft Consolidated Ordinance (Chapt. XV., p. 185) proposes to give them legislative sanction. As already said, whatever difference of opinion there may be about some of the recommendations there can be none about these, and I sincerely trust that they will pass as they stand.

Coming now to the question of ventilation, with which the author's paper is more particularly concerned, any one who has followed the local discussion for the past six years must admit that the E.R.P.M. Mines, Ltd., are to be congratulated on the excellent results which have been attained by the ventilation plant. I have no direct knowledge of the condition of the mine atmosphere 12 months ago, but all who now work under the altered conditions are agreed that they feel different men. One hears similar testimony from the Turf Mines and the Village Main Reef mine. I do not know whether an analysis has been made of the atmosphere of these latter two mines. If not, then I strongly recommend that this be done without delay—indeed, I go further to say that every manager ought to know the state of the atmosphere right throughout his mine. I am only saying now what I said over a year ago, but so far as I am aware little or nothing has been done, as we have had no results placed before the Society. It is, of course, not incumbent on those responsible to do this but results which indicate progress generally find their way into our proceedings. Might I therefore again suggest that the carrying out of this work is a moral obligation and should be done at once. It will have to be done in any case when improved ventilation becomes compulsory, as I am convinced it will be. In this connection I shall be glad to give all the advice and assistance which it is in my power to give, but at the same time I do say that a proper gas analysis survey will cost a good deal of money, because the processes are slow, special apparatus is required, and above all, very special skill in manipulation. There is another, but a minor aspect of the same question which I think has been forgotten. All will remember that the Report of the Miners' Phthisis Commission drew attention to the possibility of the compressors themselves being actual manufacturers of carbon-monoxide and since then we have been reminded here and elsewhere by the loss of a good many lives that this is no picture of the imagination. I wonder how many tests have been carried out in order to ascertain whether the compressed air is free of carbon monoxide or not, since the issue of that report. I venture to suggest few, if any; I personally know of none. I do not suggest for one moment that any is produced at all, but I do say that on a vital question like this we should have "clear consciences." Here again I must say that I cannot understand the general attitude to questions of this sort, because anything which affects the health and efficiency of employees should surely come first. Above ground we are trying to save pence per ton and spending thousands in experiments, while down below there is a possibility of saving shillings through health and efficiency alone—and all this, of course, apart from the moral aspect. I am convinced that those in authority are actuated by the very best motives and are sincerely anxious to do their best for all those under their charge, but at the same time I must confess that much more might have been done in the past.

Mr. H. Stuart Martin (*Member*): I had the honour to propose the vote of thanks for this paper, and at that time promised I would at a later date, when the paper was being discussed, add a few remarks on the subject laid before us.

The author's paper is illustrative of what has been done in mechanically ventilating those mines under his charge, and has certainly proved that there is little difficulty in adequately ventilating the gold mines on these fields on the same lines as at coal mines at home.

The cost of installation, a figure no less than £25,000 (already redeemed by improved efficiency), and the running cost of only 1d. per ton, are facts that certainly deserve our consideration. £25,000 appears to be at first sight a large sum of money to be spent on ventilation alone. However, in this case I have no doubt a considerable sum included in the estimate was spent underground in making the necessary connections and drives in an already large and extensive mining property which in its earlier days was not laid out for mechanical ventilation. Yet, on the other hand, the dykes running from N to S have formed excellent natural brattices, and by their presence have facilitated the splitting of the air.

I am glad to say, following close on the heels of East Rand Proprietary Mines, an interesting ventilation problem has also been successfully carried out at the Village Deep mine. The only interesting points of difference as compared with the East Rand Proprietary Mines are that at the Village Deep the fan is erected underground at a depth of about 2,000 ft from the surface, the quantity of air passing through the fan is about 280,000 cub. ft. per minute with a W.G. of 2·7 in., the cost of installation, including cost of cutting fan chamber, air doors, etc., did not exceed £6,000, and the running cost on the tonnage milled, covering the whole area ventilated by the fan, does not exceed 1d. per ton.

There are five other mines of the Eckstein-Rand Mines Groups, viz., City Deep, Durban Roodepoort Deep, Crown Mines, Nourse Mines and Robinson G. M. Co., for which fans have been actually ordered, and in a short time others will probably follow suit. The fans vary in capacity from 50,000 cub. ft. at 1 in. W.G. to 250,000 cub. ft. at 4 in. W.G., some to be erected on the surface and others underground. Frequently it will be found more economical to instal two, or even more, fans in a mine rather than one large one, as, if only one fan is installed, the W.G. must be sufficiently high to circulate the necessary quantity of air through the workings with most resistance, necessitating regulators which should be avoided. The placing of a fan underground is not so good as on the surface, when possible and economical to do so. The

Village Deep fan was placed underground in order to gain the necessary quantity of air at a low W.G. The small free air space, and the enormous frictional resistances in the main rectangular vertical shafts, prohibited the placing of the fan on the surface.

Speaking generally on the question of the ventilation of mines on these fields, undoubtedly the outcrop mines with their numerous shafts and outlets to the surface have provided ample ventilation in the past and have not called for mechanical means for better ventilation. However, following the outcrop mines, with the deeper levels (some of which already exceed 4,000 ft. of cover), and with the prospect of greater depths and fewer shafts—working much increased areas and larger tonnages—and increasing rock temperatures, the question of ventilation is going to be a serious one.

Apart from the installation of mechanical ventilation, suitable shafts with minimum frictional resistances and maximum free air space, as well as carefully thought out plans as regards underground lay-out and ventilating roads, will have to be considered, otherwise it will be impossible to work the deep deep levels economically and under favourable conditions. In consideration of this fact, at the deeper mines controlled by Messrs. H. Eckstein & Co., main levels and incline shafts are being driven in the footwall. These ways will act as intakes, and being driven in solid ground a minimum loss of air due to leakage will result. These main ventilating roads incidentally become excellent haulage roads for cheap mechanical underground transport, and therefore cannot altogether be charged against ventilation costs.

The author lays much stress on the quartz dust—the havoc it plays amongst our miners, and I am sure we all agree with him that with united action by the mine managers and their staffs much might be done to prevent the diseases due to this cause.

Improved ventilation, however, does not assist us in keeping the dust down, but rather helps to make matters worse, dries the mine, stirs up the dust and carries the fine particles throughout the mine workings wherever the air currents travel. Ventilation does its part by carrying away the foul gases and gives us fresh air to breathe, puts energy into our workers and gives us improved efficiency.

I cannot agree with the author when he makes the statement (p. 60 of the *Journal*) that "the fine dust in suspension must be constantly carried out of the mine, and that this demonstrates one of the necessities for adequate ventilation." With the proper application of water dust can be arrested. I am under the impression that the real mischief maker is this fine dust, and if we are going to do any good this dust must be arrested at the seat of origin, and to do this successfully the whole Rand must act together. Dr. Aymard's apparatus no doubt does arrest the dust where machines are used, whether in the face of a drive, raise or stope, but the difficulty is how to get the miners to use it. Unless the whole of our mines co-operate, Dr. Aymard's apparatus, or any other apparatus which may be equally as good, is going to make but little impression and will die a natural death.

The author seems to consider it advisable to place brattices or pipes in all dead ends, but a better practice would appear to be, where possible, to work only single shift and keep winzes well advanced. By this means the dead ends would seldom be more than a few hundred feet in advance of the main air current, and the air used by the machines is sufficient to ventilate the drives, except in special cases where local ventilation may be adopted. One of the great difficulties in working in high temperatures is when the atmosphere is also saturated. When working at great depths with high temperatures the air should be kept as cool as possible and dry, and therefore the application of water whilst arresting dust should be limited as much as possible.

In respect to health conditions of our miners, I entirely agree with the author that the proper housing of our workers is most necessary, and I can say that much has been done by Messrs H. Eckstein & Co., in this direction, and we are exceedingly proud of the new villages now being built for our workers, and in time look forward to holding all the best men on the Rand in our ranks.

There are many points in the author's paper I have not touched upon that call for discussion.

I again thank the author for bringing forward this important subject.

Mr. G. Hildick Smith (*Member*) referred to an article by Dr. Haldane,* and said:—

Although, unfortunately, unable to be present at the meeting at which the author read his interesting paper on "Ventilation and Health Conditions on the Mines of the Witwatersrand," I have read that paper with greater interest than perhaps most metal mining men, owing to the fact that I served my apprenticeship to mining in the large collieries of the Midlands of England, where the question of ventilation is of paramount importance and the first rule of the Coal Mines Regulation Act, states that "an adequate amount of ventilation shall constantly be produced to render harmless all noxious gases, so that all working places, travelling ways, etc., shall be in a fit state to work and travel therein." There is no reason at all why this should not also be the first rule in the local Mines' Regulations, and it is pleasing to note that this has at last been realized and systematic ventilation, as opposed to the old haphazard methods, is at last being adopted slowly but surely. In the installation of a mechanical ventilator, in the shape of any well known type of fan, we require for the fan itself, capable of producing 100,000 cub. ft. of air per minute, about £300. To this must added the cost of erection, ventilation doors, etc., the total cost of which, as the author remarks, is easily repaid by the increased efficiency obtained from the human factors in the mine, even if we assume that the only result of the ventilating current is the reduction of the wet bulb temperature alone. The production of a ventilating current presents no difficulties, but the proper circulation of this current underground in the mines of this field will be more difficult. When we come to the health point of view put forward by the author, we are dealing not only with an important question looked at from our point of view as mining men, but we are faced by a question of national importance and of vital consequence to the community at large. It is a well-known fact, I believe, that, on the whole, unhealthy parents bear on an average larger families than healthy parents. Unless, therefore, the phthisis and tuberculosis danger is tackled quickly and efficiently, we shall eventually have a community, in this new country, afflicted with consumption.

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* See this *Journal*, pp. 227-[illegible]

Going back now to the mining point of view, and looking upon the mines as the breeding grounds of the consumption, we may ask — What is going to be the result of the coming ventilation schemes? A general improvement in health conditions undoubtedly. But is it going to be the great improvement it appears on the face of it? Let us take the chief cause of phthisis—the dust—this opens the way for infective processes, etc., but is in itself apparently the primary cause. Prevent the dust getting into the lungs and cutting them, and we then have done a great deal towards the eradication of phthisis. We can never hope to completely eradicate it, but we shall have done all that can be done, so far as the present state of our knowledge goes. Are there therefore any known means by which we can absolutely prevent the dust from being inhaled by the miners? The answer I think is certainly, yes. An efficient respirator would do this. The author says in his paper, "I would here express my agreement with the opinion of the Mining Regulations Commission with reference to respirators. For continuous wear they are useless, although for occasional wear they are useful." How do the author, and the members of the Mining Regulations Commission know this? Have any of them in actual practice and for any length of time used respirators underground and noticed the difference in the state of their throats, nose or lungs as compared with the state of those organs after working over a long period underground without a good respirator. Have any experiments ever been conducted, with various materials such as sponge, gauze, etc., both when used wet or dry? This is the chief point I would like to emphasize. Experiments should be carried out by aspirating air containing dust, through various suitable materials, in order to ascertain which is the best material, of which an efficient respirator could be made. Personally I use a respirator of my own pattern, made from a sufficiently fine textured flat sponge, and I find that even in the work of shift bossing, which entails more heavy breathing than perhaps most other work underground, a respirator worn continuously causes no inconvenience in either breathing or talking. The coming ventilation schemes, as Mr. Stuart Martin remarks, will excite the dust, even with watering. Water pipes in practice will often be broken by blasting, etc., and as any breakage in the pipe lines will not directly affect the output, breakages of this description will always be the last job to be repaired, giving time for the ventilating current to dry up the stopes, etc., and pick up the finest and most dangerous dust as it goes along.

In conclusion, therefore, the respirator question should be gone into from a proper scientific standpoint, various types should be experimented with both in the laboratory and underground. An efficient one having been designed, it should be made a standard pattern, procurable by those of the underground workers at any rate who have sense enough to realize the danger to their health. This, quite apart from the question of fresh air and elimination of deleterious gases would tend to increase efficiency, as the class of miner who is careless of his health and seems to be anxious to suck in as much dust as possible and die quickly, will also probably be careless about his work. Given a sufficient time, therefore, the result would be that the sensible and careful man will still be mining after the careless men are "planted."

Mr. C. Toombs (*Member*): I would like to point out to the last speaker that the author described and gave an illustration of an apparatus used on the East Rand Proprietary Mines for accurately determining the dust floating about in mine air.

Mr. E. J. Laschinger (*Member of Council*): I should like to reply to Mr. Cullen that with regard to the amount of carbon monoxide which might be present in the air delivered by compressors, that our immediate Past-President, Mr. McArthur Johnston some years ago (after there had been a few explosions with air compressors on the Rand), made some experiments as to the amount of carbon monoxide in the air used by compressors. He made these experiments at the Simmer and Jack and the Simmer Deep, and said he could absolutely detect no carbon monoxide at any time in the air delivered by compressors. I am glad to make this announcement, for there has been a doubt on this point in the minds of mining engineers on the Rand for many years. After these tests I felt quite satisfied that no troubles due to carbon monoxide in the air delivered from the compressors need be feared if proper care were taken in using good oil and in efficient cooling of the air during and after compression.

Mr. C. Toombs: Were these carbon monoxide determinations made by the iodine pentoxide method? If not, I am sure that negative results obtained by volumetric absorption methods are not of much value.

Mr. E. J. Laschinger: They were not made volumetrically. They were done by the method mentioned.

The President: Perhaps Mr. McArthur Johnston might publish his figures.

Mr. A. McArthur Johnston (*Past-President*) (contributed): —Mr. Laschinger is correct in saying that in some tests we conducted two and a half years ago, no carbon monoxide was present in the compressor air. The samples were taken when the plants were under full working load, and were obtained from the Simmer Deep new compressor plant at the Rudd shaft, with an indicated pressure of 100 lb., from the Simmer and Jack Proprietary Mines plant — pressure 80 lb., and from the Simmer East compressor plant when the pressure was 70 lb. The samples were taken by blowing the compressed air into dry aspirator bottles, and this air was displaced by mercury when being passed over the iodine pentoxide. The tests showed in each case negative results, but the samples examined were insufficient in number to entitle us to conclude that carbon monoxide was always absent from these plants. Our main conclusion was that our crusade against the use of low flash point or adulterated oils, which we had previously waged, had borne good fruit.

The President: I am able myself to confirm what you say. I made some experiments, and my figures were published in the report of the Mining Regulations Commission this year, as also air analyses since installation of the Village Deep fan. Apparently, however, Mr. Cullen prefers to discuss these things without taking the trouble to examine the recent existing data.

Prof. J. A. Wilkinson (*Member of Council*): Before this discussion closes, there is one point of chemical interest to which I should like to refer. As is well known, this Society has for some years past taken a very lively interest in the composition of mine air. In the case in question very full details are given of a method of improving this by mechanical means, and this contrasts very sharply with the sparse details given, to show what improvements have been effected. These consist of a few average results only. Now I would ask the author to remedy this by stating in his reply the results obtained in the fullest possible manner, and thus to enhance considerably the value of his paper.

The President: I have great pleasure myself in supporting that request. It struck me as the only weak point in the paper. I must point out that the figures for average CO_2 which the author gives are practically impossible, or else they represent gases sampled in absurd places. The figures are much higher than any ever found on the Rand by any other experimenter. I think it would be as well too if he could explain why these two very large fans have only reduced the temperature by a few degrees, because apparently in Europe a fall of 10° or 15° might have been expected. Possibly the cubic volume of his mine may be the explanation as I notice that the ventilation has only reduced the CO_2 to ·15 in places where he has sampled it, whereas something like ·08 might have been expected.

The meeting then closed.

Contributions and Correspondence.

THE DESTRUCTION OF CYANIDE.

By their footnote on page 152 of the October *Journal*, Dr. Moir and Mr. Gray do not appear to understand the arithmetical reasons involved in the moisture figures given by Mr. O. P. Powell, in reference to sand filling.* Mr. Powell is now absent on a holiday, but the following tabular statement shows in detail the correctness of his calculations.

Moisture in Pulp.	Tons Sand in Pulp.	Tons Water in Pulp.	Tons of Pulp.	Tons water retained in Sand at 12% moisture	Tons of drainage water to be pumped
30	1,000	428	1,428	136	292
40	1,000	667	1,667	136	531
50	1,000	1,000	2,000	136	864

* See this *Journal*, [illegible]

November 23, 1910. G. O. SMART.

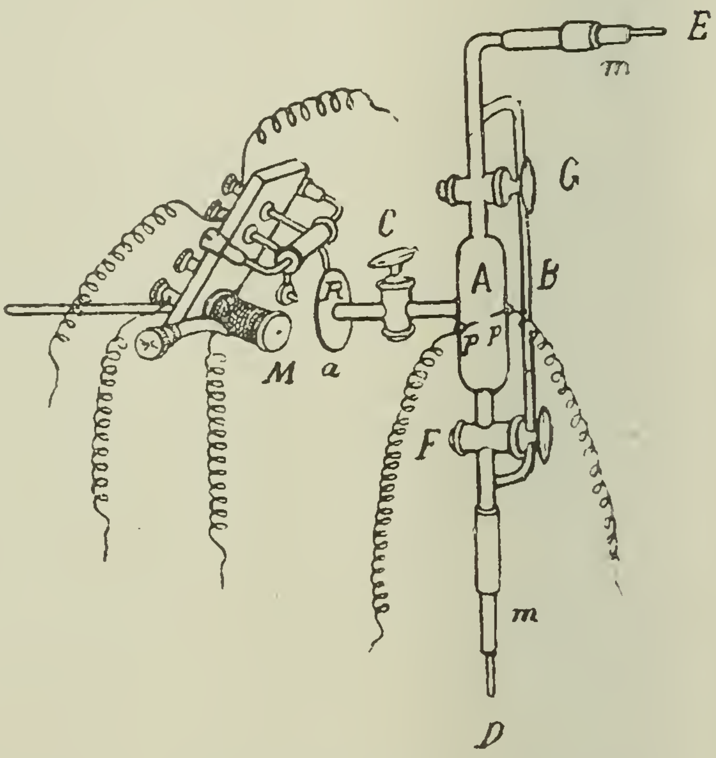
Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Determination of Sulphur in Pyrites.—"The most important sources of error in the determination of sulphur in soluble sulphates, under the conditions usually obtaining in practice, have been investigated by the authors and are found to be the solubility of the barium sulphate precipitate and the occlusion by it of sodium or other sulphate and 'free' sulphuric acid which is lost on ignition. In making a determination, it is recommended that nitrates, chlorides, and ammonium salts be, as far possible, avoided and that the following conditions be observed:—The solution, measuring 350 cc., is acidulated with 2 cc. of 2% hydrochloric acid and heated to boiling; precipitation is effected slowly (about 4 minutes for 2 grm. of precipitate), with constant stirring, and the precipitate is allowed to stand for 18 hours, when it is filtered off and washed until 25 cc. of the washings show a barely perceptible opalescence with silver nitrate; the filter is subsequently burnt very slowly, and heating of the precipitate is continued until a constant weight is obtained. To obtain results which are accurate to 0·1—0·2% of the total sulphur. corrections. which are preferably determined by experiment for the case in hand, are applied for the errors mentioned above. A very good uncorrected determination may be made by precipitating rapidly, but in this case there is a partial compensation of variable errors, and the method is not so reliable as that of slow precipitation with subsequent correction."—E. T. Allen and J. Johnston, *Journal of the American Chemical Society*, 1910, *32*, 588—617.—*Journal of the Society of Chemical Industry*, June 15, 1910, p. 692. (A. W.)

Rapid and Accurate Method for the Determination of Titanium in Ores.—"The method is outlined for the determination of titanium in rutile and iron ores is based on the the volatilization of the silica by hydrofluoric acid in the presence of sulphuric acid, evaporation to dryness and fusion with sodium carbonate and a little sodium nitrate to convert the iron and titanium to insoluble ferric oxide and sodium titanate, extraction with hot water to remove the soluble phosphates. sulphates and aluminates; solution of the ferric oxide and sodium titanate in hydrochloric acid, extraction of ferric chloride with ether; and then either reduction of slight traces of iron with sulphur dioxide, precipitation of the titanic acid by boiling in acetic acid solution, filtration and ignition to titanium oxide; or a colorimetric determination by means of hydrogen peroxide. The authors find that the volumetric method based on the reduction of titanium to the trivalent state and oxidation with permanganate gives low results. With the sulphur dioxide reduction when considerable iron is present, the titanium product is usually contaminated with some iron. The ether separation removes practically all iron from the titanium very quickly. Re-fusion of the ignited product with sodium carbonate for purification, which requires considerable time, is never necessary by use of this method. The final titanium oxide is pure, unless zirconium is present. in which case this element is removed by the usual phosphate precipitation. The method combines the colorimetric and gravimetric determinations and the colour comparison is made in solutions always totally free from iron. For convenience hydrochloric rather than sulphuric acid solutions of titanic acid are used. This method is accurate and not long."—O. L. Barnebey and R. M. Isham, *Journal of the American Society*, 1910, *32*, 957—962. *Journal of the Society of Chemical Industry*, Sept. 15, 1910, p. 1061. (J. A. W.)

Explosion Indicator.—"The instrument is intended to detect in a convenient place and at a distance from the locality tested, the existence of explosive gases in mines etc. The figure shows the construction of the instrument which is an explosion pipette, *A*, with safety packings of wire gauze at *m*, *m*, and provided with a by-pass, *B*. The end, *D*, is connected by a long 'composition' or tinned iron pipe with the place the air of which is to be tested, and the end, *E*, with a pump, so that a continuous current of the air is drawn through. The cocks, *F*, and *G*, are closed (the current still streaming through the by-pass) and the spark from an induction coil passed between the wires. If no explosion occurs, it is because the proportion of combustible gas is either above or below the explosion limit. Should it be above, then on opening for a while the cock, *C*, enough air will diffuse in to cause an explosion on sparking again. Should it be below, then if *F* and *G* are opened and sparks passed while the gas is streaming through, flame will be seen in a darkened room with a proportion of combustible gas considerably below that necessary for explosion. If the explosion be made to take place with the cocks *F* and *G* closed and *C* open, and the suspended iron plate, *R*, closing the side tube, the force of the explosion will drive R against the electromagnet, *M*, which will hold it, and electrical connections made by this contact may be made to ring a bell at the place tested, and thus give warning."—N. Teclu, *J. prakt. Chem.*, 1910, *82*, 237—240.—*Journal of the Society of Chemical Industry*, Sept. 15, 1910, p. 1084. (J. A. W.)



New Reaction for Copper.—"An intense blue coloration is obtained when a dilute solution of a copper salt is treated with an alkaline solution of 1·2-diaminoanthraquinone-3-sulphonic acid. The latter solution is prepared by dissolving 0·5 grm. of the 1·2-diaminoanthraquinone-3-sulphonic acid in 500 cc. of water and adding 40 cc. of a solution of sodium hydroxide of 40° B. The blue coloration is plainly visible with 0·0000019 grm. of copper in 1 cc. of solution, and even one-tenth of this quantity may be detected by means of the test. The reaction is characteristic of copper, other metals not yielding a blue coloration."—R. Uhlenhuth., *Chem.-Zeit.*, 1910, *34*, 887.—*Journal of the Society of Chemical Industry*, Sept. 15, 1910, p. 1085. (J. A. W.)

Potassium Cyanide and Silver Nitrate. "The interaction of potassium cyanide and silver nitrate is generally represented by the equations:—

(1) $2KCN + AgNO_3 = KAg(CN)_2 + KNO_3$;

(2) $KAg(CN)_2 + AgNO_3 = 2AgCN + KNO_3$.

Experiments were made with silver nitrate solution of such strength that 10 cc. were just sufficient to produce a white precipitate of silver cyanide with 10 cc. of the potassium cyanide solution. If equation (2) be correct, excess of silver nitrate should be present after 20 cc. of the silver solution have been added. The author found however that 25 cc. were needed before a reaction for silver nitrate in the solution could be obtained with potassium bichromate or sodium arsenate."—J. C. Bailar, *Western Chem. and Met.; Mining World*, Chicago, Aug. 13, 1910.—*Journal of the Society of Chemical Industry*, Sept., 1910, p. 1085. (J. A. W.)

The Synthesis of Ammonia.—Up to the present time the liquor obtained in the manufacture of coal-gas or from the coke ovens has been the sole source of ammonia, and its synthetic production has not been regarded as possible of attainment commercially. Attempts are being made to prepare ammonium salts from cyanamide, but even if these should meet with some measure of success the process would probably be too costly to prove remunerative. The Badische synthetic method is independent of electric power, and now that both hydrogen and nitrogen are obtainable with facility at a low cost it is by no means unlikely that the electro-chemical processes of manufacturing cyanamide and nitrates will find in it a strong competitor in the production of fertilizing material; but the available information hardly justifies a definite forecast, although there is apparently a fair prospect of success for the new process.

It is stated that the Badisch Anilin & Soda Fabrik has solved the problem of the direct production of ammonia from its elementary constituents, and will probably soon effect the preparation of synthetic ammonia on the commercial scale. Prof. Haber and M. R. le Rossignol, after much experimental work, found that at a high temperature and under a pressure approaching 200 atmospheres, the desired combination of hydrogen and nitrogen occurred, and the action was increased by certain catalytic agents, particularly osmium, but, in consequence of the high price of this metal, uranium is used in practical working. The difficulties involved were great, but the Badische staff appears to have been successful in grappling with them. The mixed gases are compressed and brought into contact with powdered uranium in a chamber where a pressure of 175 atmospheres is maintained at a high temperature, with the result that the union of a large part of the mixture of gases takes place. The ammonia and the residual uncombined gases pass into a freezing apparatus whence the liquified ammonia is withdrawn, while the unchanged gases pass through and with further quantities of hydrogen and nitrogen, are again treated in the pressure chamber. The heat evolved in the formation of the ammonia is utilized to assist in bringing the fresh mixture of gases to the requisite temperature. A satisfactory yield of ammonia is reported to have been obtained at the Badische works.—*Metallurgical and Chemical Engineering*, Vol. viii., No. 9, 539, Sept., 1910. (J. A. W.)

Commercial Cyanide.—"Dissolve 20 grm. in 150 cc. of water, filter into a 250 cc. flask, wash, ignite and weigh the insoluble residue. Add $Ca(NO_3)_2$ to the contents of the flask, shake well, dilute to the mark, mix, let stand an hour, filter off $CaCO_3$ and wash. The amount can be determined by strong ignition and weighing CaO.

The filtrate and washings are brought to 500 cc., of which 100 cc. is taken for hydroxide determination. Add $Mg(NO_3)_2$ solution, let settle for an hour, filter off, etc., ignite and weigh MgO. The weight ×0·85 gives that of hydroxide in 4 grm. of the sample.

For cyanide take 50 cc. of 500 cc. dilution, add 5 cc. strong ammonia and 5 cc. of the KI indicator. Then titrate with standard $AgNO_3$. Cl factor ×0·7324 = CN factor.

For Cyanate. To 50 cc. of the above add excess of strong $AgNO_3$ to precipitate all Cy, CNO and Cl. Shake well, filter and wash with ice-cold water. When washed, place a clean flask under the filter, and pour over the filter 10 cc. of standard HNO_3 made up by mixing 100 cc. HNO_3 with 900 cc. of water. Titrate acidity with standard NaOH. Each mol. of HNO_3 neutralized corresponds to 1 mol. CNO.

For Cl. Heat up 1 grm. of the sample in a porcelain crucible and add gradually a mixture of 5 parts Na_2CO_3 to 1 part KNO_3 until all CN is destroyed. Cool, dissolve, acidify with HNO_3, filter, and determine Cl in the solution by titration.

For K and Na. Decompose 0·25 gm. by heating with HCl, finally evaporate to dryness and weigh NaCl + KCl. Determine K by $PtCl_4$.

Commercial cyanide containing more or less NaCN is usually preferred, the lower combining weight of Na giving the greater proportion of CN."—*School of Mines Quarterly*, Columbia University, April, 1910, p. 280. (A. R.)

METALLURGY.

Laboratory Screens.—"In order to form an opinion as to the results of most metallurgical and engineering operations, we have accurate weights and measurements and methods of analysis: the qualitative result of the operation of grinding ore, on the other hand, is not so easily subjected to measurement, and the various systems proposed have so far received no general sanction. The suggestions here made are an attempt to reconcile the conflicting elements in former suggestions, and at the same time to present a system which has merits hitherto wanting; also to present a practical mechanical method of making screen analyses.

There are two phases to the problem:—

1. The measurement of the result of crushing different ores in the same machine.

2. The measurement of the result of crushing the same ore in different machines, or with different adjustments or combinations of the same machine.

Having decided to arrange a sample of crushed ore into portions, the individual particles of which have as near as may be the same size, and later perhaps to subject the different portions to chemical analysis, we are immediately confronted with the questions:

1. How many screens?

2. What sized holes in each?

Thesis. In laboratory screen analyses of crushed ores how many screens shall be used? What size shall the apertures in the screens be? and, as a corollary, what relationship shall the holes in one screen bear to the holes in the successive screen?

Imperfections to Date. Rittinger's series is too short. Richards' series is too long. De Kalb's series is too short and disappears too soon, and all three are based on the metric system. The I.M.M. standard is not a regular series.

Cube Root Series.—A geometric series of diameters of apertures for a set of screens for laboratory testing is now proposed;* its advantages are pointed out, a comparison of all the series available is made, and various curves are plotted to show their merits and demerits.

Based on $\sqrt[3]{2} = 1·259920635$.

Inches.	Diameter of Aperture. Millimetres.	Meshes per Inch. roughly.
1·0000	25·3995	0
0·7937	20·1585	1
0·6300	15·9999	1
0·5000	12·6998	1
0·3969	10·0793	2
0·3150	7·9999	2
0·2500	6·3499	2
0·1984	5·0396	3
0·1575	4·0000	3
0·1250	3·1749	4
0·0992	2·5198	5
0·0787	2·0000	6
0·0625	1·5875	8
0·0496	1·2599	10
0·0394	1·0000	12
0·0313	0·7937	15
0·0248	0·6300	20
0·0197	0·5000	25
0·0156	0·3969	30
0·0124	0·3150	40
0·0098	0·2500	50
0·0078	0·1984	65
0·0062	0·1575	80
0·0049	0·1250	110
0·0039	0·0992	130
0·0031	0·0787	160
0·0025	0·0625	200
0·0020	0·0496	250

To state the problem as simply as possible—We will start with an aperture of 1 in., and divide 1 in. by the $\sqrt[3]{2}$: this gives us our second aperture. Dividing the second aperture by the $\sqrt[3]{2}$ we get the third aperture, and so on down to the 28th term of the series, where we reach the limit of effective screening in a screen with an aperture of 0·0020 in. (roughly, 250 mesh). The decimals in this series were calculated accurately in five places, and when the fifteenth term of the series was reached it came to 0·03937 in., which decimal will be recognised as the legal British and American equivalent for 1 mm. Here, then, is a series which may start from either 1 in. or 1 mm. and, by varying the successive apertures so that they are in ratio of $\frac{1}{\sqrt[3]{2}}$, get a series of screens which obviates this and all other objections urged. Between the limits of 0·0197 in. screen (roughly 25 mesh) and 0·0020 in. screen (roughly 250 mesh) we have:—

Common series	4	screens
Rittinger's „	7	„
Richards' „	14	„
De Kalb's „	6	„
I.M.M. „	11	„
Cube Root „	11	„

The I.M.M. and the Cube Root series have the same number of screens, but the I.M.M. series is so unevenly spaced that much objection can be raised to it.

The Cube Root series proposed obviates the objections which have been raised from time to time to all the other series."—T. J. HOOVER.—*Bulletin of the Institution of Mining and Metallurgy*, May 9, 1910, p. 1. (H. A. W.)

GRADING ANALYSES AND THEIR APPLICATION.—"THE PRESIDENT thought it would be convenient to discuss these two papers together. He would therefore ask Mr. Theodore J. Hoover to introduce his paper,* after which Mr. Walter McDermott would introduce Mr. Stadler's paper.†

Mr. THEODORE J. HOOVER gave a brief summary of his paper, in the course of which referring to the series proposed by the Committee of the Institution which he had criticised, he said that all standardisation schemes had to have a start, and had not the Institution made a start, a considerable time might have elapsed before anyone would have taken the trouble to figure out any series at all. The action of the Committee certainly gave an impetus to standardisation in general.

The work which gave rise to the paper necessitated a mechanical appliance, which he had invented, in order not to have to rely on ordinary hand-shaking screens. Having had considerable experience of a more or less unsatisfactory character with patents, he had decided not to take out a patent for the machine, but to make a present of it to the engineering profession.

He wished to point out one advantage of the series, based on the $\sqrt[3]{2}$, which he recommended for adoption, namely, that the English and metric units of measurement were brought into a semblance of harmony. It was purely a coincidence: and as far as he knew, throughout the range of mathematics there was no other occasion in which the English and metric system were brought into so near a relationship.

Mr. WALTER MCDERMOTT, in introducing Mr. Stadler's paper, said that by comparing the sizes of the apertures, it would be found that a large portion of the author's grades agreed very closely with those of the I.M.M., and therefore, in practice, all his conclusions could be put in use and practically applied by the use of the Institution's standards only. At the same time, it was a scientific paper. It was an attempt to give a value in energy units to each grade, and, of course, it was worked out scientifically. It was worked out to five places of decimals of an inch, and while they knew that, in practice, such refined measurement would be impracticable and useless, still, for the purpose of a scientific paper establishing standards, it was perfectly justifiable.

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* [illegible] been suggested for use in [illegible] tests, but none of [illegible] were satisfactory for [illegible] purposes. In connection with work for the Minerals Separation, Ltd., 62 London Wall, London, E.C., on their [illegible]. After a [illegible] amount of research this new series was decided upon, but in looking over the literature of the subject [illegible] it was found that this series was mentioned by Robert H. Richards in a paper read before the American Institute of Mining Engineers, referred to [illegible] its use.

* See this *Journal*, p. 219.

† See this *Journal*, Vol. X., May, 1910, pp. 382-390.

Starting from a different requirement, and proceeding by a different process of reasoning, the author arrived at exactly the same conclusion as Mr. Hoover, their standard list being alike, except that Mr. Stadler worked to five places of decimals and Mr. Hoover to four.

The author had proved by a number of experiments that although in breaking various kinds of rocks the forms of the particles were very irregular, still, in effect, and on the average, they were subject to the same laws as though broken into perfect cubes, so that he was able to value the same various grades by what he called energy units, representing the proportional power required in crushing.

There were a great many figures and formulæ given for applying the author's conclusions; but in effect the argument was this, that the energy required in making a fine grade of material was more than the energy required in making the same quantity of a coarser grade. For example: in comparing the efficiency of the work of a stamp-mill with the work of a set of rolls, it was not sufficient to compare simply the quantity which each of them would pass through a 60-mesh screen; but the product must be graded below the 60-mesh into all the various sizes, even to below the 200-mesh, and the relative value allowed for in energy units for each grade, and in accordance with the proportional weights of the grades.

Many present had, of course, recognised generally the principle involved in this consideration of the efficiency of crushing machines, and it was referred to in the discussion of Mr. Caldecott's recent paper on coarse crushing by stamps; but Mr. Stadler had translated what was broadly recognised into actual figures, and had given a formula by which they were enabled to express duty done in actual energy units, and therefore to make comparisons, as the author did for instance, between a stamp mill and a tube mill.

Mr. Stadler's conclusion, which led to a great deal of discussion in Johannesburg, was that the tube mill was very inefficient mechanically: that the stamp mill was not credited properly with the proportion of fine grades it produced: and that the actual finishing work done by the tube mill was over-estimated as compared with the mechanical energy consumed in that work. Of course, this conclusion applied only to mechanical efficiency; and in itself, therefore, was only one element in calculations determining choice of machines.

The paper was one to which it was difficult to do justice in a summary; but the objects in view and the theoretical skill shown in working them out would command the admiration of all.

Since the paper had been written, Mr. Stadler had sent over the following note referring to a different method of estimating the efficiency of crushing machinery suggested by Mr. Caldecott, which, with the permission of the President, he would read as having some bearing on the paper.

Mr. STADLER'S communication was to the following effect:—

Since the above paper was written, Mr. Caldecott has published a method of his own for computing efficiency. He has introduced a new 'nominal crushing unit,' which he defines as a stamp with a running weight of 1,250 lb. and an assumed new weight of 1,350 lb., dropping 100 times per minute, with set drop of 8 in. and actual drop of 7½ in.

It is difficult to see the necessity or desirability of introducing a new unit for the energy of different stamps, when the two well understood and scientific units of foot-pounds and horse-power are available. The value in nominal crushing units of any other stamp of varying conditions has still to be determined by the ratio of foot-pounds of the particular stamp to that established as the nominal crushing unit; so that the usual units might be taken as the standard at once. It is further assumed that a tube mill 5½ × 22 ft. is equal to 30 of the nominal crushing units and entails an equal consumption of power, taken as about 100 h.p. It is admitted that this ratio of tube-mill to stamps 'varies somewhat according to the conditions of operating.' It would perhaps be more correct to say that it varies very largely with changes in conditions of working.

It is not correct to take the production of -90 mesh as a measure of the work done in crushing, as Mr. Caldecott does in saying that 'the more tons of this material that can be obtained per stamp, or per tube mill, or per horse-power, the better.' Such a basis of calculation gives undue credit to the 'finishing' work of the tube mill on the already finely crushed feed, and the high duty of the preceding crushing to the fine condition is ignored.

By similar reasoning, Mr. Caldecott might have said that the ultimate object sought is the reduction of the +60 mesh, and in this case he would have arrived at the opposite conclusion, namely, that practically all the work done in crushing is performed by the preliminary crushers, and little is left for the tube mill to do. The figures obtained by his method do not in effect show any advantage in favour of coarse crushing; indeed, according to his figures, the production of -90 grade is for the same stamp weights about the same for any coarseness of battery screens; whereas, by my system of calculation by energy units, the greater efficiency now being claimed for coarse crushing stamps is clearly shown, as given below for a stamp of 1,250 lb. running weight:—

Mesh and Aperture of Battery Screen.	Duty per 24 Hours.	Mechanical Value of Pulp.	Relative Mechanical Efficiency per h.p.
9 mesh, ·272 in.	13·0 tons	18·0 E U.	234·0
200 ,, ·053 ,,	7·5 ,,	27·5 ,,	161·3
600 ,, ·028 ,,	6·5 ,,	23·2 ,,	150·8
1,200 ,, .017 ,,	5·2 ,,	24·3 ,,	126·4

Mr. W. McDERMOTT, in opening the discussion on the two papers conjointly, said it was rather curious that they should have had two papers both of which were based on exactly the same proposed standard of laboratory screens; but the two papers were different in object, and could therefore be criticised in a different manner.

In the Transvaal and other places there had been criticisms of the I.M.M. screen based on what seemed to him to be a misapprehension of what was intended. The I.M.M. standard was practically a tool; it was the introduction of a uniform tool into practice which had been so varied in the past that there had hitherto been no means of comparing results between engineers. When engineers had written in the past about grading tests by various mesh screens, no one was able to judge exactly what the sizes of the apertures of the screens were; the meshes had been often specified entirely without regard to apertures.

Now in Mr. Hoover's case, and in the case of some other criticisms, the other extreme had been gone to, of describing entirely by apertures, and that to an extent which was not scientifically warranted by the conditions of laboratory tests. He thought no one could question that Mr. Hoover was absolutely correct in his theoretical conclusions. His standard table was a better one than the I.M.M. standard theoretically; and yet, if this column of figures and apertures were examined beside that of the I.M.M. it would be found that in only a few instances was there any departure which amounted in actual practice to anything worth mentioning. By omitting the 70 and 90-mesh screens from the I.M.M. table—which any engineer could do at his discretion—there was really nothing left of Mr. Hoover's improvement except the introduction of a 25-mesh size.

In his opinion, any discrepancies between the I.M.M. table and Mr. Hoover's ideal one were far more than outweighed by the inaccuracies which existed in the use of any screen as a tool, and particularly of some of the suggested screens. The screen was not a scientific instrument: it was a mere approximation, an empirical method of getting grades which experience had shown were useful divisions for the various purposes mentioned by Mr. Hoover. But he thought that in not a single one of the uses to which Mr. Hoover had referred was the accuracy at which he aimed either obtainable or necessary.

The inaccuracies due to the actual method of using screens; to the weaving; to variations in time; to the nature of the material; were vastly more than the theoretical advantages claimed by the author over the standards adopted (after consultation with engineers all the world) as useful average grading sizes for laboratory use.

Of course, the slight departures from regular progression between the I.M.M. apertures when translated into a curve might appear revolting to a pure mathematician; but what did such departures actually amount to in size of aperture? He could not help thinking that in those refinements regarding screens the main object of laboratory grading was lost to sight. After all, a grade of particles contained all variations between two different apertures, whatever the standards adopted; and, therefore, extreme scientific accuracy was unattainable, and screening could only be a rough classification.

What was wanted chiefly was a standard by which they could translate the work done by a man at one end of the world into the knowledge of a man at the other end, so that they should both be working on the same basis; or that at least, if a man liked to use another set of screens but translated his results into the nearest I.M.M. standard, people of more limited devotion to pure science but with greater interest in its practical application, would be perfectly satisfied.

All the conclusions which had been reached by Mr. Stadler and Mr. Hoover would be equally well understood and applied if the nearest I.M.M. screens were mentioned.

On that point he might be permitted to say something on the subject of the use of the term 'mesh' Of course, to the extreme scientific mind, the use of the word was still objectionable. It was said: Why should we use the word 'mesh' when it means nothing at all? Well, in the case of the I.M.M. standard it did mean something. It now meant the same all over the world, whereas formerly it was not definite. He thought all those who used screens would agree that it was very much easier to think in mesh. He found it impossible to either remember or realise four places of decimals of an inch; it conveyed nothing to his mind. But with the knowledge that the I.M.M. screen was one in which the aperture was equal to the wire thickness, and that a 40-mesh screen was double that of the 80-mesh screen, they had a simple workable standard with a consistent ratio between mesh and aperture. It was based on something which could be visualised in one's mind's eye.

Mr. H. L. **Sulman** said that it was a coincidence that they should have two papers on the same subject, involving questions of considerable complexity, upon which the authors arrived at markedly similar conclusions—these being in apparent opposition to the recommendations of the Institution of Mining and Metallurgy.

Messrs. Hoover & Stadler exhibited an uncanny agreement in the standard they independently adopted for seriating mesh apertures. That both authors without collaboration should select the cube as the typical geometric particle, and the cube root as the function to determine particle volume and aperture area in a new scheme for the grading of ore, at first sight appeared to imply a mathematical principle from which there was small chance of escape.

The question to be settled was: shall a simple, if arbitrary, scheme of apertures primarily dominate the size of the particles; or, shall some chosen one of several irrelevant functions, derivable from a geometric ideal, determine the size of apertures through which particles of a wide range of shape and irregularity must pass!

Now as to the cube as the ideal geometric particle, Messrs. Hoover and Stadler, in common with Richards, Rittinger, and other authorities who had suggested mathematical standards, appeared to impute some magic property to the cube, taking it to represent the solid geometric form necessitated by a square aperture. He could not accept that view. Cube particles were conspicuous by their absence in any crushed ore; even were the latter derived from cubic minerals such as galena, or pyrites, it produced nothing nearer than irregularly shaped tablets, etc., of wide variation in configuration.

Apart from punched and slotted screens, the simplest way to produce a sieve was to plait two parallel series of wires at right angles, thus obtaining square holes; that was the original of the cube ideal.

Had it been found easier to weave a sieve as a cane seat is woven, and to make hexagonal or octagonal in place of square holes, the mathematical ideal for aperture seriation would be founded upon a geometrical solid with a hexagon or octagon as the cross section. With punched screens the natural geometrical figure selected would be the sphere, with slotted screens some form of the prism, rhomb, or slab; thereafter in each case would follow the choice of the most *convenient* function from amongst the many appertaining to such geometrical shapes.

It was obvious that no seriation based on the functions of a cube could apply with the same mathematical cogency to circular holes or to slots; on the other hand, any rational series must comply equally with every shape of aperture.

The eye was unable to compute variations in average particle *volume* with the readiness it could appreciate variation in average particle *diameter*.

This had been recognised in formulating the I.M.M. series; their fundamental difference from the authors' recommendations was that particle size shall be quantified by average diameter, and not by volume or cross area. But, when all was said and done, every mathematical scheme had finally to express its apertures by fractions (more or less cumbrous) representing *widths*.

Not only was there an absence of any directly useful relationship between a selected function of a cube and the particle passing an aperture founded upon such, but a little consideration would show the cube itself to be inadmissible as a basis for determining the areas of even square holes.

Imagine a screen with square apertures of absolute regularity, a number of cubes placed upon it which would just pass through them (*i.e.*, cubes with sides of smaller area than the holes), and the sifting to begin. Not a single cube would pass through: it would require considerable dexterity to place one so that it would drop through its aperture. As soon as movement commenced the cubes would tilt cornerwise and become jammed: their *solid diagonals* instead of their sides would tend to become parallel with the plane of their meshes, the "screening" of the cubes ceasing before it commenced. The selection of the cube as the natural geometric solid to pass through a square hole of equal side area was an incorrect mathematical premiss. The only regular shape which would pass a square aperture of slightly greater width without jamming was the sphere.

This aspect received support from Mr. Stadler's experiments. No screen, taken singly, meant anything definite in relation to grading; it merely determined what particles should pass or be rejected by it; for the grading effect it must be considered in connection with the next below it in the series, the mean of the two apertures being taken. Thus, the grading effected between a 10 and 20-mesh screen would be the retention of particles of an average diameter corresponding to 15-mesh, given a uniform relation between the meshes of the screens.

In the case, however, of hypothetical cubes graded between any two sets of apertures, Mr. Stadler had shown that a large empirical deduction had to be made from the *mean* cube value before any approximation to the actual size of an ordinary (Rand) ore particle could be reached.

Mr. Stadler conducted nine experiments, and found that the mean cube content must be reduced to an amount varying between 58·3% and 66·9% of the theoretical value, and adopted the average figure of 61·4% for Rand ore.

But there was no mathematical relationship between a cube and 61·4% of a cube: in other words, the assumed cubic relationship was exploded by the quantifications which Mr. Stadler had made as to actual particle volumes.

Again, 61·4% (the mean between fairly wide figures) was only the average observed in regard to a particular quartzose ore: with differing rocks this empirical figure would likewise differ widely. Given on the one hand an ore of, say, igneous origin, and on the other, one of schistose or micaceous nature, one would find, respectively, long splintery particles or flakes, or, so to speak, minute lead-pencils or pancakes as extremes.

It was evident that the volume constant to be applied to the hypothetical mean cube as determined for Rand ore, would here suffer further distortion: *i.e.*, a fresh empirical constant was necessary for each and every ore.

He would ask Mr. Stadler for information as to the manner in which the particle volumes, ranging between 58·3 and 66·9%, were determined. It seemed that these could only be arrived at by some method of counting the grains retained between two given screens; details of the method adopted by Mr. Stadler would be of great interest.

It was curious to notice that 61% of a cube was nearer to the volume of a sphere (52·36%) than to that of the original cube, viz., 100%: the seriation proposed by Messrs. Stadler and Hoover was thus actually closer to the sphere in practical grading effect than to the cube.

Of course, he did not suggest any fresh seriation based upon the sphere: he merely used the illustration as a *reductio ad absurdum* in disposing of the cube as the mathematical basis.

Considering now the selection of the particular 'function' of the cube; what was the overpowering reason for choosing the *cube root*. Rittinger adopted the square root, and Richards took the fourth root of two: the authors have given good reasons for discarding these. But there were plenty of other fractional relations which might be selected in order to get any desirable or varying number of sieves between two given limits. It merely needed the application of a simple formula to determine this. The cube root was only a fraction, and a somewhat lengthy one, and stood in no necessarily simpler relation to a cube than did any other decimal or fraction. It happened, coincidentally, that the fraction, which in this case represented the square root, gave a series of the same number of screens as had been recommended by the Institution between the same limits: it was, therefore, merely a convenient *choice* after all, and no compulsory mathematical basis was involved. This, to him, seemed to dispose of the mathematical argument.

He would next deal with the practical limit of accuracy attainable by any screen series, and refer to Mr. Hoover's photo-enlargements of typical sieve apertures. The author had adopted the method, which he (Mr. Sulman) had used in connection with the I.M.M. investigation in 1907, in showing the wide range of error inseparable from the most carefully woven material. Mr. Hoover had also given the formula by which to compute the average diameter of the apertures, but refrained from drawing attention to their great variations.

In the photo-micrographs he made three years ago, he found a difference of over 22% between the largest and the smallest apertures in a given screen; he thought it probable that the differences shown in Mr. Hoover's photographs would be larger.

When one introduced this inevitable error into any series of precise aperture widths, the result was startling. Assuming a variation of 20% only and dividing this equally above and below the normal, it would be found that an aperture of 1 mm. in the cube root series would in some cases be increased to 1·1 in an upward direction, and decreased to 0·9 in a downward direction in others. The next screen of 0·7937 mm. would similarly vary between 0·864 and 0·71, and so on. In other words, owing to the unavoidable differences in weaving screens, whether of wire or silk, the mesh extremes almost overlapped. That did away with any theoretical accuracy, whatever the proposed basis.

Although both authors advocated the cube root seriation, there was a curious difference in their conclusions with regard to the Institution series. Mr. Hoover was very severe, and could say but little good of it. Mr. Stadler, who had gone further by introducing the important crushing-energy factor, gave a table, which, by application of the factor to both 'I.M.M.' and 'cube root' apertures, showed the two to be almost identical in practical result; certainly so within the limits of screen accuracy.

Mr. Stadler here virtually controverted Mr. Hoover. But the latter's statements that there is no relationship capable of simple expression in the I.M.M. series, and that 'the only simple law adhered to is that the series is arranged in the order of meshes per inch, and each succeeding screen has an aperture smaller than the preceding one,' should not be permitted to pass without criticism.

There *was* a simple and definite relationship between the I.M.M. apertures, and, with the exception of one or two meshes interpolated for practical convenience, the series could be plotted in a definite and regular curve.

Perhaps the most striking aspect of Mr. Stadler's paper was the author's extremely important introduction into grading analyses of the energy factor—and its qualification in the crushing of a given rock to a product of definite sieve analyses. The proposal, and the work done in connection with it, was new to him and called for careful consideration. It appeared to be a departure likely to prove of great practical value in the future.

Author's Reply to Discussion.

Mr. THEODORE J. HOOVER: Mr. McDermott and Mr. Sulman have defended their case valiantly and with adroitness. Most of their arguments are, however, effectually and sufficiently answered by the original paper and the accompanying mathematical curves; but the following points deserve attention.

Mr. McDermott's inference that too much scientific exactness can be introduced into laboratory methods is apt to be misunderstood. Certainly the I.M.M. stands for the very highest possible scientific accuracy, and any method or appliance which will in any way tend to increase scientific exactness is admissible. The attempt of the I.M.M. to standardise screens illustratates their true position, and it is not an effectual answer to say that the method is inaccurate at best and let it go at that. Science is never pessimistic; it is always looking for improved methods and standards, and is never content to say 'this is good enough.'

The strongest argument against the I.M.M standard screens is the fact that two men, working on problems of ore crushing and ore sizing, where the very highest attainable scientific accuracy was necessary, have found the I.M.M. screens, within two years after their adoption, practically useless.

The author cordially recognises the pioneer work of the I.M.M. Standardisation Committee. A pioneer in any field does good work when he lays the foundation for greater order; but the fact that he is a pioneer precludes the possibility of finality, and he need take no offence if those following him need something better than his best. A pioneer's log cabin is only a preliminary affair and is always followed by structures more adapted to later needs." Discussion on STADLER and HOOVER. *Bulletin the Institution of Mining and Metallurgy*, June 16, 1910, p. 5-33. (H. A. W.)

CALCULATIONS OF PERCENTAGE OF RECOVERY.—"In a recent article on the calculation of recoveries in concentration operations, the genesis of certain formulæ was given.

Where a = assay of ore
b ,, tailing
c ,, concentrate
x = tons of ore
y ,, tailing
z ,, concentrate

$$\frac{100c(a-b)}{a(c-b)} = \text{percentage of recovery.}$$

$$\frac{c-b}{a-b} = \frac{x}{z} = \text{ratio of concentration.}$$

The formulæ have several applications in practice other than those previously given, as for instance.

In a mine examination it is sometimes necessary to construct tentative figures as to the concentration of the ore. The assay of the ore will be known and a rough panning test can be made for ratio of concentration. A certain percentage of recovery can be assumed.

Thus:
a = assay of ore = 4% copper
b = ,, of prospective tailing = unknown
c = ,, of prospective concentrate = unknown
x = tons of ore = 100
y = ,, tailing = unknown
z = ,, concentrate = unknown
Assume 80% recovery
Then $a\ x = 4$ tons of metallic copper
$b\ y = 0.8$ tons ,, ,,
$c\ z = 3{\cdot}2$ tons ,, ,,

y and z can now have values from 0 to 100, b can have any values from 0 to 4 and c can have any value from 0 to say 30% (depending on what mineral carries the copper). However, if we assign a value to either b, c, y or z, the value of the others becomes fixed.

Suppose we assume that c, the assay of the concentrate, will be 20% copper, then

$$\frac{20}{100}z = 3{\cdot}2 \text{ tons}$$

$$z = \frac{3{\cdot}2}{0{\cdot}2}$$

$z = 16$ tons of concentrate
$y = x - z$
$y = 100 - 16$
$y = 84$ tons of tailing
$b\ y = 0{\cdot}8$
$b = \dfrac{0{\cdot}8}{84}$
$b = 0{\cdot}95\%$ copper

Our figures then are
Assay of ore = 4% copper
Tons ,, = 100
Assay of tailing = 0·95% copper
Tons ,, = 84
Assay of concentrate = 20% copper
Tons ,, = 16

We can now apply freight and treatment charges, and then proceed with the estimates of mine valuation.

In a case where it was important to learn something of the work in a mill where the management was very secretive, the following data were available:

Surreptitious samples of the tailings assayed 1·4% copper

Surreptitious samples of the concentrate assayed 15% copper

By counting, from the hotel veranda, the sacks of concentrate shipped per week, it was possible to obtain a fairly close estimate of the concentrate produced viz., 60 tons per day.

A question put to the mill-man elicited the information that 300 tons per day was being treated; this was also checked by counting the buckets on the tramway from the mine.

Assay of ore = unknown
,, ,, tailing = 1·4% copper
,, ,, concentrate = 15% copper
Tons of ore = 300
Tons of concentrate = 60

$$\frac{x}{z} = \frac{c-b}{a-b}$$

$$5 = \frac{0{\cdot}15 - 0{\cdot}014}{a - 0{\cdot}014}$$

$5a - 0{\cdot}07 = 0{\cdot}15 - 0{\cdot}014$
$5a = 0{\cdot}206$
$a = 4{\cdot}12\%$ copper
$ax = 12{\cdot}36$ tons copper per day in ore
$cz = 9$,, ,, ,, ,, concentrate
Percentage of recovery = 72

Later when access to the mine was secured the mill and mine records showed an actual recovery of 70·7 % for the month during which the week's observations were made from the hotel veranda. The ore for the month, however, had an average assay of 4·5 % copper. The books showed:

Tons of ore for month 7511 = x
,, concentrate 1602 = z
Assay of ore for month 4·5 % = a
,, concentrate 14·91 = c
,, tailing 1·37 = b

$$\text{Theoretical recovery} = \frac{100c(a - b)}{a(c - b)}$$
$$= \frac{(100)(14{\cdot}91)(4{\cdot}5 - 1{\cdot}37)}{4{\cdot}5\,(14{\cdot}91 - 1{\cdot}37)}$$
$$= 76{\cdot}6\%$$

Actual recovery:
7511 @ 4·5 = 337·995
1602 @ 14·91 = 238·8582
238·8582 ÷ 337·995 = 70·7 % recovery

The difference between 76·6 % and 70·7 % or 5·9 % represented the losses and inaccuracies, and is not worse than occurs in many cases.

A famous mine not a thousand miles from Melbourne publishes an interesting annual report. The following figures were on one occasion given, suppressing the information as to the tonnage and assay of the tailing:

Tons of ore = 237,265
Assay ,, = 11·5 oz. silver, 16·3 % lead, 18·7 % zinc
Tons of concentrate = 43,493
Assay ,, ,, = 29·7 oz. silver, 59·8 % lead, 10·3% zinc

The tons of tailing can be determined by subtraction and the assay thereof by calculating the contents and then dividing, but it is easier to apply the formula:

$$\frac{x}{z} = \frac{c - b}{a - b}$$
$$b = \frac{33{\cdot}09}{4{\cdot}46} = 7{\cdot}4 \text{ oz. silver}$$
$$b = \frac{29{\cdot}2}{4{\cdot}46} = 6{\cdot}54\% \text{ lead}$$
$$b = \frac{91{\cdot}8}{4{\cdot}46} = 20{\cdot}6\% \text{ zinc}$$

Multiplying according to the old system and dividing by the tons we find that the results are correct within the limits of the slide-rule.

In another mine not far away the following figures were given:

	Tons.	Lead per cent.	Silver oz.	Zinc per cent.
Ore	46,301	12	9·1	10·1
Concentrate ...	5,924	61	29·3	5·5
Jig-tailing ...	17,672	4·3	4·9	8·6
Table-tailing ...	17,085	3·5	5·7	12·0
Slime ...	5,618	10·6	11·0	13·8

For purposes of subsequent treatment of these residual products by flotation processes it is desirable to know what will be the probable assay of the combined jig tailing, table-tailing and slime. This can, of course, be ascertained by multiplying the tonnage of each by its assay to get the metal contents, adding together the metal contents and dividing it by the total tons. In order to show the usefulness of the formula this old standard method is given below and the same result obtained by the formula. I spent two hours on the first calculation, whereas the second was accomplished in ten minutes.

Ore	46301·1 tons ×	12 %	= 5556·132 tons Pb
	46301·1	9·1 oz.	421340·01 oz. Ag
	46301·1	10·1 %	4676·4111 tons Zn
Concentrate	5924·1	61 %	3613·701 tons Pb
	5924·1	29·3 oz.	173576·13 oz. Ag
	5924·1	5·5 %	325·8255 tons Zn
Jig-Tailing	17672·8	4·3 %	759·9304 tons Pb
	17672·8	4·9 oz.	86596·72 oz. Ag
	17672·8	8·6 %	1519·8608 tons Zn
Table-tailing	17085·6	3·5 %	597·996 tons Pb
	17085·6	5·7 oz.	97387·92 oz. Ag
	17085·6	12 %	2050·272 tons Zn
Slime	5618·6	10·6 %	595·5716 tons Pb
	5618·6	11 oz.	61804·6 oz. Ag
	5618·6	13·8 %	775·3668 tons Zn

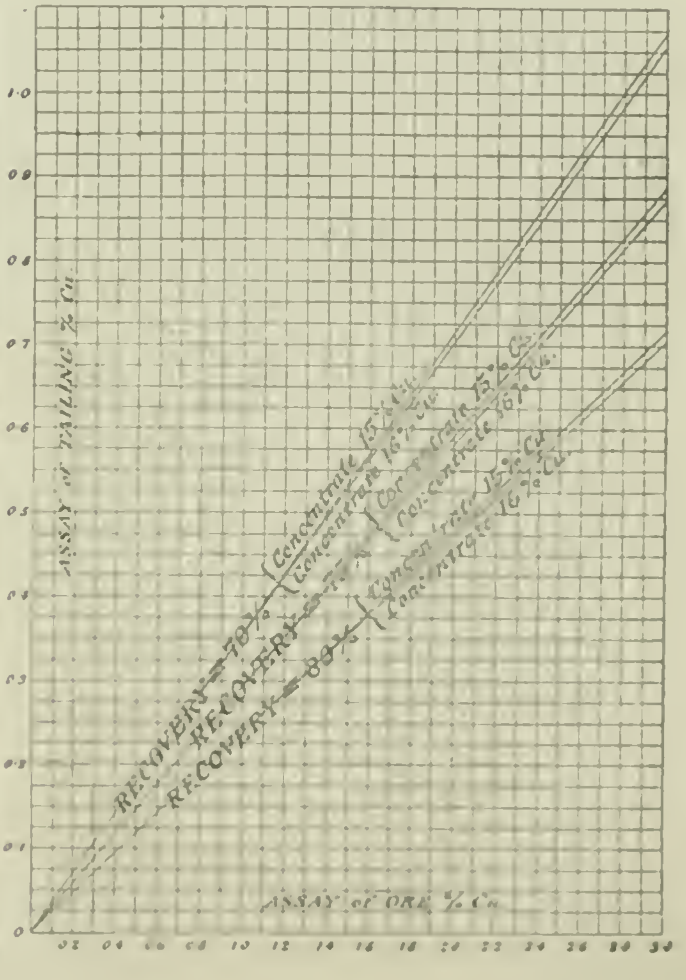
Then add, to get total tons of metal in total residual products:

	Lead	Silver.	Zinc.
Jig-tailing ...	759·9304	86596·72	1519·8608
Table-tailing ...	597·9960	97387·92	2050·2720
Slime... ..	595·5716	61804·60	775·3668
Total ...	1953·4980	245789·24	4345·4996

Then add the contents of the concentrate as a check on the figures:

Lead.	Silver.	Zinc.
1953·498	245789·24	4345·4996
3613·701	173576·13	325·8255
5567·199	419365·37	4671·3251

These last figures compared with the contents of the original ore show a commendable lock-up of a year's results.



Now the total of residues is :

Jig-tailing	...	...	17672·8
Table-tailing	...	...	17085·6
Slime ..	...	...	5618·6
Total	...	...	40377·0

Now divide the contents by the tons :

$$1953 \cdot 498 \div 40377 = 4 \cdot 8\% \text{ Pb}$$

$$245789 \cdot 24 \div 40377 = 6 \cdot 1 \text{ oz. Ag}$$

$$4345 \cdot 4996 \div 40377 = 10 \cdot 8\% \text{ Zn}$$

which we know from subsequent investigation fairly represents the average value of the combined residues.

By applying the formula

$$\frac{x}{z} = \frac{c-b}{a-b}$$

For the lead assay :

$$\frac{46301 \cdot 1}{5924 \cdot 1} = \frac{61-b}{12-b}$$

$$7 \cdot 816 = \frac{61-b}{12-b}$$

$$93 \cdot 792 - 7 \cdot 816b = 61 - b$$

$$6 \cdot 816b = 32 \cdot 792$$

$$b = 4 \cdot 81\% \text{ Pb}$$

For the silver assay :

$$7 \cdot 816 = \frac{29 \cdot 3 - b}{9 \cdot 1 - b}$$

$$b = 6 \cdot 13 \text{ oz. Ag per ton}$$

For the zinc assay :

$$7 \cdot 816 = \frac{5 \cdot 5 - b}{10 \cdot 1 - b}$$

$$b = 10 \cdot 77\% \text{ Zn}$$

All these calculations were worked twice, once by hand and checked with the slide-rule.

The above figures for tonnage call for some attention. The tons of ore and the tons of concentrate were probably obtained with fair accuracy by weight. But what shall we say of the other figures, which are given to tenths of a ton. It may be taken that the assays are fairly accurate, for if even the arithmetic average of the days' runs be taken the error will not be big enough to vitiate results. It is safe to say that the jig-tailing, table-tailing, and slime were not weighed. They may have been measured by survey and calculated from cubic contents. When the difficulties of moisture and losses are taken into account the accuracy of the results is noteworthy.

Based on the above formulae certain tables can be constructed for the use of the staff, so as to give graphically a clear idea of the result of a shift's work. As an example let us assume that samples are taken of each shift's work ; that the material is a simple concentrating ore assaying less that 3% copper ; the recoveries are never lower than 70% nor higher than 80% ; that the concentrate never assays higher than 16% copper nor lower than 15% ; these are all assumptions that could fit a case.

Taking the formula :

$$\text{Percentage of recovery} = \frac{100c(a-b)}{a(c-b)}$$

and writing it as a simple equation without the percentage we have

$$R = \frac{c(a-b)}{a(c-b)}$$

where R is a constant which is a decimal part unity. Then by transformation :

$$b = \frac{ac(1-R)}{c-Ra}$$

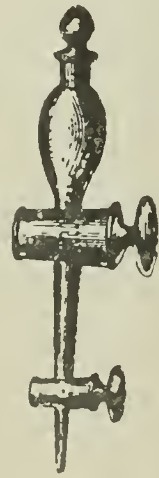
By giving a various values and taking values of and C as above indicated we can plot a series curves as in the diagram. These are of great u

Having done this let us suppose the shift samples assayed :

Ore	...	...	2·6% copper
Tailing ..	..	..	0·75 ,,

By reference to the co-ordinates we find that the point (2·6) (0·75) falls just above the curve for 75% recovery and 15% grade. By making a large wall-chart and plotting curves at intervals of ½% recovery between 70% and 80% the results of a shift's run can be read at a glance with a considerable degree of accuracy. Attention should be called to the fact that any variation in the assay of the concentrate between 15% and 16% has little influence on the curve ; in fact, for over half of its distance the y co-ordinate of the points is within unity whether the assay is taken as 15% or 16%. Near the end of the curve the difference becomes just sufficient to plot the two curves separately with comfort. On a mine where the average assay of the concentrate over a long period is known it would be sufficiently accurate to plot the curves with a fixed constant. This chart would also show at a glance any inconsistency in the assay results, as for instance, if the assayer reported ore 2·9%, tailing 0·55%, concentrate 15·2%, he may be suspected of inaccuracy unless the alternative assumption is made that there has been a marked improvement in recovery to much above 80%. The use of this chart constructed on assumptions of theoretical recovery has a salutary influence on the members of the staff when once they understand its import ; they are on the look out for losses on the one hand and inaccuracies in method on the other."—T. J. HOOVER.—*Mining Magazine*, Aug., 1910 (J. E. T.)

SEPARATING FUNNEL FOR AMALGAM TESTS.—"The conical shaped glass separating-funnel shown has proved to be an excellent elutriating apparatus in making bottle amalgamation tests in which the tailing and mercury were separated and assayed, much loss having occurred with other devices because a part of the slime was carried away by the wash-water. In using this funnel the lower cone is filled with clear water and both cocks closed. After amalgamation has been completed in the bottle, the content is emptied into the upper cone, water added to dilute the pulp, and the stopper placed in the top. After being shaken an instant the mercury settles and is drawn into the lower cone by opening the larger glass cock. Some of the sand will pass through with the mercury, but very little slime will accompany it if the lower cone has been filled previously with clear water. The glass stopper is then drawn and the tailing emptied on a filter, then the smaller cock



is opened and the mercury drawn into a dish. If the mercury be floured it can be quickened with a little sulphuric acid and the quickened mercury poured off from the small amount of sand which descended with it. The separation can thus be made with no loss of tailing."—W. H. COGHILL. *Mining and Scientific Press*, July 9, 1910, p. 53. (K. L. G.)

MINING.

THE VENTILATION PROBLEM IN DEEP MINES.—"As coal and other minerals near the surface become exhausted, mines are constantly tending to become deeper. New problems are in consequence presented to the mining engineer, and one of these relates to ventilation.

Mine ventilation fulfils several main purposes, any single one of which may be of predominant importance in any particular mine. It will be useful first to review these purposes.

1. *Removal of Blackdamp.*—In nearly all mines some constituent of the exposed mineral is liable to oxidation. As a consequence oxygen disappears from the air, and carbon dioxide (CO_2) is liberated. . . . . . .

2. *Removal of Fumes from Explosives.*—The fumes from explosives are usually not merely unpleasant, but also poisonous; and where little or no firedamp or blackdamp is produced, as in many metalliferous mines, the removal of fumes may become the main object of ventilation.

3. *Removal of Firedamp.*—In coal seams which are fairly near the surface very little firedamp is usually given off by the coal, as the seams can drain themselves of the gas through the superincumbent strata. With increasing depth this natural drainage becomes less [illegible] however, with the result that the coal may [illegible] heavily charged with firedamp. . . . .

4. *Cooling effects.*—When a mine is very warm, the working power of the miners is diminished, and with increasing heat work finally becomes impossible. Where, therefore, the cost of the miners' labour be[illegible] proportion to the value of the mineral, as [illegible] mining, it becomes a matter of vital importance [illegible] the workings cool. The deeper a m[illegible] the higher will be the temperature, other th[illegible] equal, and the greater the need for ve[illegible] with a view to cooling. *In very de[illegible] therefore, the main object of ventilation is [illegible] impurities, but to keep the men cool.*

[illegible] roughly distinguish a zone nearest th[illegible] the chief aim of mine ventilation is [illegible] blackdamp and fumes; an intermediate zo[illegible] mines) where removal of firedamp is the ch[illegible] ventilation; and a deeper zone in which th[illegible] of ventilation is to keep the men cool. It [illegible] ventilation in this deeper zone that th[illegible] is concerned.

[illegible] of keeping the men sufficiently cool to [illegible] without injury to [illegible] efficiency, is ev[illegible] physiological [illegible]; and the ph[illegible] the problem will be discussed fir[illegible] with other warm-blooded an[illegible] internal body temperature, w[illegible] maintained within narrow li[illegible] as measured in the re[illegible] lower during the ni[illegible] hours of m[illegible] We may roughly st[illegible] between 98 and 101° F.; an[illegible] of heat by the body must be so [illegible] body temperature within th[illegible]

How is this balance maintained? Heat is constantly being produced in the tissues of the body by the oxidation processes which are constantly going on. During rest this heat is produced at a rate sufficient to heat the whole body by about 2° every hour, and during work the production may be increased about three or four times. The corresponding loss of heat occurs partly by conduction to the air and radiation, and partly by conversion into latent heat in the evaporation of moisture. In ordinary cool air only a small part is lost by evaporation; and the loss by conduction and radiation is regulated by variations in the circulation through the skin. In a cold place the skin circulation is diminished so as to prevent the excessive loss of heat which would otherwise occur; whereas in a less cool place, or during muscular work, the skin circulation is increased so as to promote loss of heat. In a warm place, particularly during work, the increased skin circulation cannot get rid of the surplus heat by conduction and radiation; and in this case the loss by evaporation is increased by sweating. When the external temperature equals or exceeds the body temperature, as often happens in tropical or subtropical climates, the whole of the heat is got rid of by evaporation.

Provided the air-temperature is below the body-temperature, heat-loss by conduction to the air is evidently increased greatly if the air is in motion; and similarly evaporation is promoted if the air is in motion, provided that the dew-point is below the body-temperature. Movement of the air is thus of the utmost importance in keeping a man cool, although there is practically always some movement of the air close to the body, in consequence of the convection currents produced by the warming or cooling effects of the skin on the air surrounding it.

Evaporation from the body can only occur if the dew-point is below the body-temperature; but even if the dew-point were below the body-temperature it does not follow that any cooling action would be produced; for if the air were warmer than the body it might communicate more heat than was lost by evaporation. The body during sweating is, in fact, comparable to a wet-bulb thermometer, which indicates, not the dew-point, but a temperature intermediate between the dew-point and the actual temperature of the air, and dependent on both. If the wet bulb temperature is below the body-temperature heat is lost by the body; but otherwise there can be no loss of heat.

In order that the blood returning from the skin may be sufficiently cooled to prevent actual rise of internal body temperature it is clear that the skin must be kept cooler than the inside of the body. To effect this there must be a constant and free loss of heat by evaporation and conduction, or by evaporation alone, from the skin. Hence the wet bulb temperature must be well below both the body-temperature and skin temperature. But movement of the air greatly increases the loss (or gain) of heat by evaporation and conduction; hence with the air in motion the wet bulb temperature need not be so low as would otherwise be necessary in order to get rid of the necessary amount of heat.

In order to determine the maximum wet bulb temperature at which abnormal rise of body temperature can be prevented, the writer recently made a number of experiments, partly in Cornish mines, and partly in the laboratory or a Turkish bath. The result was that if the wet bulb temperature exceeded about 88° F. (31° C.) in still air, it was impossible to prevent rise of body temperature during rest and with all clothing removed except light flannel trousers. The result of remaining in a wet-bulb temperature exceeding 88° was that the body-temperature gradually rose, and symptoms of heat stroke began to develope. The more the wet-bulb temperature rose above 88°, the more rapidly did the body-temperature rise. In a good current of air, however, a wet-bulb temperature several degrees higher could be borne. It made no appreciable difference what the actual air temperature was. *Only the wet-bulb temperature and the movement of the air mattered.*

During muscular work, as could have been predicted with certainty, the body-temperature rose rapidly with a wet-bulb temperature at 88° in still air; and even with the wet bulb at 78° the body-temperature could hardly be prevented from rising during moderate work in still air. In a good air current, on the other hand, continuous moderate work was possible with the wet-bulb at 85°.

The effect of clothing is, of course, to lower considerably the wet-bulb temperature which can be tolerated. A wet-bulb temperature exceeding about 60 or 65 may, for instance, be very dangerous to soldiers marching in heavy uniform. Under such conditions a man may go on until he drops from heat-stroke: and unless he is promptly treated by applying cold water, ice, or whatever means are available for cooling him down, he may die. Among miners heat-stroke hardly ever occurs. The miner is free to divest himself of clothing and to stop work if he is too hot; and it is only in rare cases that he fails to do both. The only case of heat-stroke known to the writer was the tragic death of Welsby, the Yorkshire miner, who, while wearing a rescue apparatus, lost his life in a gallant effort to reach the men who were cut off by the fire at Hamstead Colliery in 1908. The wet-bulb temperature in the return air-way, which he and his companion Whittinghame were traversing, was probably about 80°, and they had ordinary clothing and a heavy apparatus to carry in a road which was low and not meant for travelling. They had pressed forward regardless of the heat, and in returning they both had all the symptoms of heat-stroke. When Welsby could go no further Whittinghame struggled on in the hope of bringing help, but he himself only reached the shaft with great difficulty. The symptoms which he described to the writer were clearly those of heat-stroke, and nothing was wrong with the Garforth rescue apparatus, with which both men were equipped. . . . .

When the body-temperature rises beyond 102° various symptoms of distress begin to appear. Any exertion becomes very difficult, and produces great panting and exhaustion. There is also marked disinclination for any work, either physical or mental, and considerable irritability. The pulse becomes very rapid, even during rest. With further rise of temperature a comatose condition developes, which may end in death. The following description of the symptoms is quoted from a letter to the writer by Dr. Boycott, who had somewhat incautiously moved about as usual while investigating Ankylostomiasis in Levant Mine, which runs out under the sea, and had at the time no ventilation which the writer could measure at the top of what were believed to be upcast or downcast shafts. The only moderately cool places in the deep workings were where compressed air was allowed to escape freely. Dr. Boycott was fortunately accompanied by Mr. (now Professor) Cadman, who was less seriously affected, and could thus render assistance:

'We climbed straight down one of the shafts (temperature 78° F. at the bottom) to the 278 fathoms level, and walked out under the sea by the pony road, soon discarding our coats, and going in vests and thin trousers only. We then climbed down the submarine shaft (86° wet and dry bulb at the top and 87° at the bottom) to the 302 fathoms level, and walked out westwards, about half a mile I should think, to an 'end,' where the temperature was 93° wet and dry bulb. The going was bad, and I had to stoop most of the way, as the road was largely timbered. When we reached the 'end' we had been underground I dare say three hours, as we had been collecting samples of fæces, etc. I felt very hot, and was glad to sit down. My mouth temperature was 103·5° by a clinical thermometer which Cadman read for me. There was a man and a boy in the 'end,' supposed to be hand-drilling, but they did not seem to be doing anything except sweating. As you know, the men are said to wet the drill-holes by pouring the sweat out of their boots! Coming back, I did pretty well till we had to come up the ladders from the 302 to the 278. The ladders were, I suppose, about three fathoms each, and I had to lie down at the top of each one and gasp. When we got up to the pony road and had a drink of water, I soon felt pretty well all right, and came up the man-engine (my first experience) all right, except that I lost my light. By the time I had washed and changed I felt quite well, and regretted that we had wasted our opportunities of collecting samples. After being in the end I felt no interest in the matter.'

The foregoing discussion makes it possible to formulate roughly the requirements which must be fulfilled by ventilation designed to keep men cool enough for continuous work. It is clear that what is needed is to maintain such an atmosphere *that the body-temperature does not rise beyond normal limits during work, assuming that the men are working stripped to the waist, or entirely naked, as is the custom at present in some of the deepest mines.* This end may apparently be attained by keeping the wet-bulb temperature under about 78° if there is no air current, or under about 85° if there is a good current. More accurate and extended data on this subject are still required, however.

If the limits of wet-bulb temperature are overstepped, the inevitable result in a mine is that the working power of the miners must fall off very seriously. In other words, they have to be paid much more highly for any definite piece of work; and even so they will rightly object to working in places where they are uncomfortable from the heat, and feel that they are mostly wasting their time. The increased cost of production of coal, or any mineral of similar low value per ton, must thus make the mining of it impossible under present conditions, so that the ventilation problem becomes one of life or death for the success of the mine.

We may now consider the conditions which lead to excessive wet-bulb temperatures in mines. The most important of these is doubtless the increase of rock temperature with increasing depth. It is often assumed that, except in volcanic districts, the rate of increase in rock temperature with increasing depth is fairly uniform. There seems to be now no doubt that there are in reality wide variations in this respect. To take two or three instances, there is no doubt that at Dolcoath Mine, in Cornwall, the real rate of increase in virgin rock is about 1° F. in 90 ft, whereas at Hamstead and other deep mines near Birmingham, the rate of increase is only about 1° in

110 ft., and in the Transvaal the increase is a great deal less—about 1° in 220 ft.

The actual temperature in a mine depends on the ventilation as well as on the depth and rate of increase of rock temperature. . . . . .

The return air represents, however, oxidation processes occurring not only in the roads and working places, but in large extents of goaf or closed workings. What is of importance is whether the air is cooling or warming the parts where men are working or passing. We can judge of this by comparing the actual air-temperature with the original temperature of the virgin coal or rock, if this is known, as it ought to be; but a far clearer insight can be obtained by analysis of the air at different points in the ventilation current. To be of any use these analyses must be accurate. So-called 'technical' methods, which allow of errors amounting to more than 0·1 % in the oxygen, are of little use.

From the existing knowledge of mine air, to which an exceedingly valuable addition has been made in the recent report made to the Royal Commission on Mines by Professor Cadman and Mr. Whalley, it seems clear that as a general rule under present conditions the ventilation of coal mines has a very considerable heating effect, except on the intake roads. On the working places themselves, the effect may be either cooling or warming, according to the amount of ventilation, as compared with the rate of oxidation. In some collieries the rate of oxidation on the intake roads themselves is so great, or the ventilation so small, that before the air reaches the working face it is heated much above the natural rock temperature.

When air is passing month after month, or year after year, down a shaft or along a roadway, the surrounding rock is either cooled or warmed to near the average temperature of the air. The rock round an upcast shaft is, for instance, gradually warmed, while that round the downcast is cooled. Any effect on the temperature of an alteration in the ventilation is thus very gradual, and may take months or years to develop fully. The gradual effects of ventilation are, however, often very striking. Thus in Dolcoath tin mine, in Cornwall, where the natural ventilation is much more than sufficient to carry off all the heat produced by oxidation, the temperature of the open stopes at the deep levels is much below the natural rock-temperature, so that in spite of the great depth and a rock temperature of nearly 100° at the bottom, work can be carried on fairly freely. A large new vertical shaft, which is now approaching completion, will supply abundant cool air direct to the deepest workings, and obviate any further difficulty from the heat for a long time to come. In Levant tin and copper mine, on the other hand, where the ventilation does not carry off more than an insignificant fraction of the heat produced by oxidation, the temperature of the deeper workings, which are not nearly so deep as at Dolcoath, was found by the writer to be much above the calculated rock temperature, and so high that very little work could be done by the men.

A rapid increase of rock-temperature with increasing depth may be in some respects not so much of a disadvantage as at first sight appears; for the greater the rate of increase of rock temperature the more assistance will be given to natural ventilation. *A fan would, for instance, be wholly superfluous in the case of a deep Cornish mine. In the Transvaal, on the other hand, natural ventilation is so feeble that much difficulty is experienced in the removal of fumes from explosives, etc., and it seems as if fan ventilation may require to be introduced as the mines get deeper.* As the air descends or ascends a shaft it is heated or cooled by the compression or expansion which occurs, the rate of heating during the descent being about 1° F. in 180 ft. But in the Rand mines the rate of increase in rock-temperature is only about 1° in 220 ft. It appears, therefore, as if there might often be no natural ventilation at all in these mines. Cooling by evaporation occurs, however, in the downcast shaft, while cooling of the upcast is hindered by the condensation of moisture and consequent liberation of latent heat, which tends to occur in all upcast shafts, and so helps to keep them warm. Once the ventilation has started, as it must do in cold weather, it will thus tend to continue, since the upcast shaft will become permanently warmer than the downcast.

The previous physiological discussion showed that the problem of ventilation for coolness may probably be summed up in that of keeping the air moving and the wet-bulb temperature below about 80° F. We can now follow the process by which this critical point is reached in a deep mine. As the air descends the shaft or incline, it is in the first place heated by compression. This effect is, of course, not directly measurable from day to day, as the shaft walls may themselves either warm or (in hot weather) cool the air. Heating by compression warms the air about 5½° F. for every 1,000 ft. of descent. Hence, in a mine 4,000 ft. deep compression accounts for a rise in air temperature of about 22°. If we assume an increase of 1° in 70 ft. the rock temperature at the bottom of the mine will be about 50° + 57° = 107°. The mean air-temperature at the shaft bottom will be about 50° + 22° = 72°, apart from the influence of the shaft wall in warming or cooling the air by conduction or evaporation. The more rapid the air-current, and the wider and drier the shaft, the more closely will the mean temperature at the shaft bottom approach the latter value. As the air passes to the working face it will pick up heat from the rock surrounding the intake roads; but the more rapid the current the less will its increase in temperature be. It will also pick up moisture; and if coal dust or small coal is lying on the roads or exposed on their sides this will continuously oxidise and thus help to warm the air. In some mines this oxidation may assume very formidable dimensions. At Hamstead Colliery, for instance, the oxygen percentage of the air in an intake 3,000 yards from the downcast shaft was found to be reduced by 0·88 %, which corresponds to oxidation capable of continuously heating the whole air current of the road by about 25° F. At that time the main intake roads were in the thick coal. The actual temperature on the road was only about 15° above the natural rock temperature, so that most of the heat was escaping into the surrounding strata.

If the intake air takes up much moisture in the downcast shaft or main intake roads it will, of course, be correspondingly cooled or prevented from heating. Every grain of moisture added by evaporation to a cubic foot of air corresponds to a cooling effect to the extent of about 8° F. Average fresh air in this country contains about 3·2 grains of moisture per cub. ft., while air saturated at 80° contains 11 grains. In saturating an air current to 80° the heat required would thus be the same as in warming it (11 − 3·2) × 8 = 62 degs. Hence, air at a mean temperature of 50° takes up about twice as much heat in being saturated at 80° as in being warmed to 80° in a mine. There is no immediate physiological disadvantage in cooling the intake air by evaporation of moisture, as in spite of the increased saturation of the air the wet bulb temperture is slightly lowered. A little consideration will show, however, the secondary effects of such a course must be disadvantageous in a deep or warm mine; for the cooler the intake air is kept the more rapidly will it tend to take up heat from the surrounding strata, and if it is already highly charged with moisture, *the range within which its temperature can be allowed to rise without interference with the working capacity of the men is much restricted.* For instance, if air has already been saturated with moisture at 70° its temperature cannot subsequently rise beyond about 96°, even if no further moisture is added, without affecting working capacity; whereas, if no moisture had been added far higher temperatures could be borne without inconvenience. Watering the main roads is certainly desirable as a safeguard against coal dust explosions. But if the roads will not stand it, or if it causes such a serious rise in wet-bulb temperature at the working face that working costs become excessive, other means must be taken for guarding against explosions.

It is clear that, so far as difficulties from heat are concerned, air may *be kept in a fit state for working in by either keeping it from taking up moisture or keeping it cool, or by both methods combined.* To keep the air cool in a deep mine, the air-current to the working face must be sufficiently increased to cool the rock round the downcast shaft and intake roads to the desired extent; the oxidation processes along the roads must at the same time be so reduced that their effects in warming the air remain within manageable limits. To keep the air dry all unnecessary moisture must be avoided, and the air-current must be so large that the percentage of moisture which it picks up along the roads is as small as possible. With the ventilation properly laid out to secure these objects there should not be much difficulty in working coal at depths up to at least 5,000 ft. Success will certainly depend, however, on realising the nature of the difficulties and taking the proper means to meet them."—DR. J. S. HALDANE. *University of Birmingham Engineering and Mining Journal*, July, 1909, p. 49-57. (G. H.-S.)

DEPTH IN MINING.—"In *Bulletin* No. 424, issued by the United States Geological Survey, the question of maximum depths of mining is discussed by Mr. George H. Ashley. Reviewing the advance of deep mining in Europe, he remarks that there appears to be no engineering or mechanical difficulties which cannot be successfully overcome within the limit of 4,000 ft. In the United States to-day coal lying more than 3,000 ft. below the surface is being disregarded in connection with the disposition of coal lands on the public domain, whereas coals below 3,000 ft. are being successfully and profitably mined in Belgium, England, Wales, France and Australia —a group of countries which supply about half the world's production. Of course it should be recognised that as coal-mining in different parts of the United States is extended to great depths, there may be localities in which the difficulties arising from deep mining may combine in such a way as to make further operation impracticable. It is believed, however, that such localities will be highly exceptional and that in general the obstacles encountered in the United States will be no greater than those which have been successfully overcome in other countries. At present the deepest mining which occurs in the anthracite region reaches about 2,200 ft. The absence of deep mining is due to the fact that, in the Appalachian region, where coal-mining began and the greatest development has taken place, the coal beds, with few exceptions, are near the surface, and the structure of the rocks is such that the coal beds do not extend to great depth. Likewise in the northern and eastern regions of the Interior province, where mining has gone on for a long time, the structural basins are relatively shallow, and the coal beds are near the surface. In the Rocky Mountain province, where the coal generally lies in deep structural basins, mining is not far advanced, and for the most part is confined to the margins of the basins. The deepest workings in the bituminous fields of western Pennsylvania probably do not exceed 1,000 ft.; in Ohio and in West Virginia, which includes the New River and Pocahontas fields, the present depth or cover of coal-mining is still less. The same maximum is found in Eastern Kentucky, Tennessee, and Alabama, except possibly at a few places. In fact, throughout the Appalachian region, with the exception of the south end, or that part in north-eastern Alabama where there are some deep basins, the maximum cover of the unworked coal beds, probably does not exceed 2,000 ft. In the Rocky Mountains province the deepest mine about which information has been procured in mine No. 1 of the Union Pacific Coal Company, at Rock Springs, Wyo., where a depth of 2,000 ft. has been reached. Of the eight regions included within the Rocky Mountain province five are large structural basins in which the coal beds outcrop more or less persistently around the margins, and if these beds are continuous underground they must reach great depths in the centres of the basins. The coals are late cretaceous to tertiary in age, and are contained for the most part in rocks which are prevailingly soft. It is obvious that as mining is extended to greater depths in the soft coal bearing rocks of the west, more or less difficulty may be expected."—*The Science and Art of Mining*, Sept. 3, 1910, p. 31. (A. R.)

STATE CONTROL OF MINING.—"Mine managers are hedged about by too many laws and regulations. They are frequently trying to master their duties, as laid down in voluminous acts of Parliament, when they might be better employed studying the natural peculiarities of their own particular mines. The gentlemen who frame these laws seem to imagine that all mines can be worked upon one uniform principle. Sufficient allowance is not made for the varying natural conditions. Too many laws of an inflexible nature imposed upon all mines and all managers regardless of local and individual circumstances, must have the effect of checking initiative and inspiring the idea that it is more the business of the State than the manager to safeguard the workers. The State, having gone so far in the matter of law-making, surely ought to follow this up by maintaining an inspector at every mine to see that the law is observed alike by worker and employer. If the State is so eager to protect labour that it leaves mine owners and managers with but a mere shadow of freedom then let the State carry its policy to a logical issue and relieve owners and managers of the duty of looking after the workers' as well as their own interests. Let there be an ample staff of inspectors to attend to the safety of the workers and leave the managers free to attend the business of the employers, subject to the requirements of the inspectors. Either let us have less State interference or full State control in matters of safety. The policy of the State being what it now is, it is not sufficient that the inspectors employed by the State should only, as a general rule, visit the mines in case of accident. Let the inspectors be on the spot constantly to prevent accidents. If we had fewer laws and more inspectors it would be better."—T. Good, *Cassier's Magazine.—The Science and Art of Mining*, Sept. 3, 1910, p. 25.

Coal Winding Record.—"The Bolsover Colliery Company's Mansfield colliery now claims to its credit the world's record in bringing coal to the surface. The amount of coal raised to the surface for a week, which consisted of $5\frac{1}{2}$ working days, ending July 19th, was 24,582 tons, which averages about 4,469 tons per day of 7 hours 40 minutes winding. This works out at 9·7 tons of coal raised per minute throughout the week's winding."—*Science and Art of Mining*, Aug. 6, 1910, p. 601. (A. R.)

Prospecting for Coal.—"1. (*a*) To prospect for coal in an unexplored country, the exposed strata must be carefully examined for contained fossils in order to determine to which formation the strata belong. To do this requires some knowledge of geology, and having decided that the strata belong to the carboniferous, it remains only to find the coal either by exposing an outcrop by digging, or putting down bore-holes.

(*b*) In a well mapped and well developed country, there will usually be no doubt as to the kind of strata in any particular district, and ordnance maps will be of great assistance as indicators of outcrops and also of rate and direction of dip The search is then largely an intelligent use of the map, intelligent observation in the field, and perseverance with the pick and shovel. Bore-holes may be required in this case also.

2. For boring to this depth a derrick or shearlegs will be necessary, together with a windlass, so as to easily raise and lower the rods when required. The rods will be of a uniform length of 6 ft. and joined together by the usual male and female joint. From 4 to 6 cutting bits will be required so as to allow time for sharpening, etc.

A bracehead will be required, and for the first 30 yds. the percussive motion of the tool will be provided by from two to four men at the bracehead. Beyond this point it will be necessary to relieve the men by the application of either a rocking lever or a spring pole. With the latter appliance boring will be easily accomplished to the required depth. Other appliances necessary during the work will be sludgers for cleaning the hole, keys for the support of the rods while unscrewing or screwing up, and spanners for the same operation. As important appliances as any will be those for dealing with accidents, should any arise. They are usually the crow's foot, the screw bell and the spiral worm.

3. The principal methods employed for deep boring are:—

(*a*) Hand boring by rigid rods.

(*b*) Mather and Platt system of boring by flat rope.

(*c*) Diamond system of boring.

(*d*) The American system of boring by round rope.

The points to be considered in deciding which method to adopt are:—

(1) The object for which the boring is made.

(2) Probable nature of the strata to be passed through.

(3) Speed of boring by the different systems.

(4) Cost of boring by the different systems.

(5) Whether a supply of water is available, this being necessary in some of the methods employed.

4. The advantages of sinking a shallow shaft at the mouth of a bore are:—

(*a*) It saves lining the hole through the same distance because this will usually be loose ground.

(*b*) It increases the effective height of the shearlegs

(*c*) In exposed situations it provides good shelter for the men.

(*d*) It is often cheaper to sink a shallow shaft than to bore that same depth at 50 or more yards.

By the latter statement is meant that two or three yards of shaft, say five or six feet diameter through loose soil, can be sunk at a less cost and with infinitely less trouble than boring the last two or three yards of a boring which is 50 or more yards. We will take an example: Suppose a bore hole to go to a depth of 100 yds. is contracted for at the rate of 3s. 6d. per yard for the first five yards and increasing by 3s. 6d. per yard for every additional five yards. The first yard costs 3s. 6d. and the hundredth yard cost $\frac{100}{5} = 20$ times as much, *i.e.*, £3 10s. The last three yards of such a bore-hole will be at the rate of £3 10s. per yard. If, therefore, three yards of shaft be sunk at the commencement the hole will only need to be 97 yds. deep and a sum equal to $3 \times$ £3 10s. = £10 10s. will be saved.

5. The most frequent are the following: (1) Breakage of the boring apparatus: (2) Tools, etc., falling down the hole. There is also in deep boring an excessive vibration set up in a long length of rods, loosening the sides of the hole, causing the rods to buckle and often to fracture. To overcome the latter difficulty various forms of sliding joints and free fall cutting tools have been devised, among the best known being Oeynhausen's sliding joint, and the free fall apparatus of Kind and of Dru.

The remedy for accidents (1) and (2) will be in the direction of the recovery of the broken pieces: and failing that, if the obstruction is large, to pound it to pieces small enough to be brought up by the sludger, or to pound them into the strata at the side of the hole and out of the way of the boring. Another process is to bore alongside a broken tool which has become firmly embedded and afterwards put into this hole a heavy charge of dynamite in the hope of loosening the broken tool. Before adopting these extreme measures, the tools usually employed are those for the recovery of broken pieces, such as the crow's foot, the screw bell, and the spiral worm.

6. (*a*) Since it is a fact that bore holes are very rarely vertical, it is sometimes desirable to have the course of bore surveyed: and if such is done a crooked bore gives just as valuable information as a straight one.

(*b*) Beds of conglomerate composed of very hard pebbles in a comparatively soft matrix will readily cause deflection: a mixture of flint and chalk, the flint being on one side of a bore and the chalk on the other will do likewise: and highly inclined beds of varying degrees of hardness also give considerable trouble in this direction.

(*c*) To straighten a crooked bore hole, the crooked portion may be filled with cement or concrete, and after it has set well, re-boring may be carefully carried out.

(*d*) To survey a bore-hole, probably the very best apparatus is the Clinostat of MacGeorge. Several of these should be encased in a suitable brass tube, and, after liquifying the gelatine, lowered to the desired point by means of a strong hempen cord or copper wire. Generally it will be advisable to take observations at regular intervals, say every 50 ft. throughout the length of the bore, and if the deflection at each of these points is noted, a line drawn through these points will represent the course or path of the bore hole."—*The Science and Art of Mining*, Sept. 3, 1910, p. 35. (A. R.)

EXPLANATION OF EFFICIENCY OF SAND AS TAMPING MATERIAL.—By way of illustrating the advantage of sand over clay for tamping, the author fills a glass tube like the globe of a lamp with clay and shows how easily the contents can be pushed out by means of a wooden rod of a diameter a little less than the bore of the glass tube. He next fills the same tube with sand tightly packed in, and shows that it is practically impossible to force it out again.

He holds that clay tamping approaches the state of things which would be produced if a wooden or iron rod were used as tamping.

The pressure which would be exerted at the first instant of the explosion, on the inner end of the rod would immediately be communicated through its whole length and it would begin to move bodily from the hole. The rod in fact would act as a projectile and would probably be quite clear of the hole before the full energy of the charge had been exerted.

On account of the non-cohesive nature of sand tamping, and the fact that even when pressed together the grains are separated by air spaces, the pressure exerted on the inner cross section of the tamping by the gases from the explosion is not felt through the whole mass of the tamping and some of the energy is expended in forcing the sand against the sides of the borehole. The gases from the explosion therefore get time to exert something like their full force against the surrounding rock. The author recommends sand cartridges, and instances cases in which after the explosion of heavy charges these cartridges were found still in the boreholes. As further support for sand tamping, he refers to the fact that snow in the muzzle of a gun barrel can cause bursting of the barrel with a normal explosive charge in use.—W. SCHULL.—*Zeitschrift fur Gesamte Scheiss und Sprengstoffwesen*, No. 13, July 1, 1910. (T. D.)

DEEP MINING ON THE RAND.—"At the Transvaal University College Mr. Lionel Phillips, one of the Rand mining magnates, in lecturing on the "Rand Gold Deposits" pointed out that mining operations on the Rand could, with profit, be carried on to a depth of at least 7,000 ft. from the surface, and that, therefore, the Witwatersrand goldfield would not be exhausted during the present century.

It is generally considered here that, owing to the extremely favourable conditions for deep mining prevailing on the Rand, that mining operations will be possible at a much greater depth than 7,000 ft. from the surface, providing an adequate method of hoisting is available, and the coming extensive use of electrical power for all mining operations on these fields seems the best likely to afford a solution to the deep winding difficulty. It must not be forgotten that on the Rand, as the mining operations deepen, the rate of increase in underground temperature steadily decreases. At first it was estimated at 1° F. for every 82 ft., but when actually tested down to a depth of 1,000 ft., it was found to be 1° F. for every 200 ft. Recent observations down to a depth of 3,500 ft., however, showed 1° for every 208 ft., and more recent tests have shown that over 4,000 ft. the rate of increase was only 1° F. for every 255 ft. There seems every likelihood of a constant decrease in the temperature gradient as greater depths are attained on these fields. Mines are working here at a depth of 4,500 ft. without the slightest attempt at fan ventilation and, as a matter of fact, are dependent on natural ventilation and the compressed air used for power purposes. Eventually, it seems possible that the temperature gradient will fall as low as 1° F. for every 400 ft. or 500 ft. in depth, so that as regards temperature that does not at present seem likely to form an insurmountable obstacle to deep mining on these fields. Then it must not be forgotten that the native labourer here is capable of working efficiently at much higher temperatures than the white labourer and this, again, is a valuable local asset to the prospects of ultra deep mining on these fields. The atmosphere here is also much drier than in other parts of the world, whilst the hanging and foot walls leave nothing to be desired. The banket beds are equally hard and capable of standing a pressure of 7,000 lb. per square inch and, even if crushed, are not like coal seams, but will still retain their value. No matter what is said to the contrary the Witwatersrand is destined to become the scene of the deepest mining operations in the world, and is destined for many years if not for centuries, unless some new field is discovered, to continue to be the premier gold producer of the world."—*Canadian Mining Journal*, Sept. 15, 1910, p. 571. (A. R.)

MISCELLANEOUS.

34TH ANNUAL REPORT OF H.M. INSPECTORS OF EXPLOSIVES, 1909.—"*Factories and Magazines.*—There were under continuing certificate 33 factories and 90 magazines, and under licence 107 factories and 350 magazines. 261 visits were paid to factories and 494 to magazines.

Accidents.—The number of accidents by fire and explosion of which the department had cognisance was 453, causing 60 deaths and injuring 445 persons. The number of accidents in factories for explosives was 51, causing 6 deaths, and injuring 12 persons. The number of deaths was below the average for the decade (7·3).

Importation.—The amount of foreign blasting explosives containing nitroglycerine, imported in 1909, was 1,311,488 lb., against 1,086,205 lb. in 1908. Of the former amount 829,480 lb. were transhipped to other countries. The non-glycerine explosives imported amounted to 11,635 lb. against 6,100 lb. in 1908. The number of detonators imported was 18,146,815. The importations of fireworks amounted to 783,502 lb.

Chemical Advisers' Report.—Messrs. Dupré report that of 496 samples examined by them, 62 were rejected for the following reasons: Low heat test (21), exudation (20), and incorrect composition (21). 41 new explosives were examined, of which 30 were passed, 8 passed preliminary tests, and 3 were rejected.

Testing Station at Woolwich.—Captain Desborough reports that 66 explosives are on the 'permitted list.' 15 explosives were tested during the year, of which 5 successfully passed the test.

Amount of Explosives Used in Mines and Quarries in Great Britain.—The following table gives the details:—

Explosives.	Lb.	Percentage of Total.
Permitted explosives ...	8,502,232	28·3
Gunpowder	17,595,475	58·3
Gelignite	3,085,529	10·2
Blasting gelatine and gelatine-dynamite...	616,436	2.0
Cheddite	123,531	0·4
Dynamite	117,260	0·4
Various	51,424	0·2
Total	30,091,887	100·0

The following list gives, in tons of 2,000 lb., the quantity of the various permitted explosives, with a consumption of more than 50 tons annually. Bobbinite (559 tons), arkite (375), monobel (369), ammonite (277), samsonite (253), saxonite (252), rippite (242), roburite (239), westfalite (222), bellite (221), carbonite (215), stow-ite (130), Faversham powder (113), ammonal* (107), albionite (101), excellite (88), rexite (83), abbeite (60), and permonite (60 tons)." *Journal of the Society of Chemical Industry*, Sept. 15. 1910, p. 1082. (J. A. W.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

PRINCIPLES OF CHEMICAL GEOLOGY. By JAMES VINCENT ELSDEN, D.Sc., F.G.S., is made up into 222 pages, 10 mo., with index. There are 10 chapters and 44 diagrammatic illustrations. Published by The MacMillan Co., New York, City, and Whittaker & Co., London, England. Price $1.60 net.

"The main object of the book is to present in a concise form the application of the principle of equilibrium to certain geological phenomena. To the student of geology nothing is at first more bewildering than the multiplicity of changes which rocks and minerals undergo in nature. To elucidate and stimulate interest in this branch of geology, Mr. Elsden has divided the subject into ten chapters, of which the following are synopses: I. Equilibrium between the crystalline and amorphous states, based on Le Chatelier's principle that 'any external change in the factors of equilibrium of a system is followed by a reverse change within the system.' II. Equilibrium influenced by viscosity plays an important part in retarding the transition of minerals. III. Diffusion as a factor of equilibrium in geological transformations may be due to gases, liquids, or solids. IV. Surface tension as a factor of equilibrium is considered as one of the most important features in supersaturation. V. Vapour pressure as a factor in equilibrium is based on experiments that show that many solids possess an appreciable vapor pressure at moderate temperatures. VI. Equilibrium conditions of polymorphous forms, that is where transformation of a mineral from one crystalline form to another with differences in specific gravity, melting point, and other physical properties. This subject possesses geological interest owing to the effects produced by dynamical metamorphism. VII. Equilibrium in solution. From the point of view of the phase rule, unsaturated solutions are trivarient systems that possess three degrees of freedom, and can only be defined by the three variable factors, pressure, temperature, and concentration. VIII. The eutectic theory is elucidated from ice to its projection on a plane triangle, making it necessary to examine how far the eutectic theory may apply. IX. The theory of solid solutions applied to geological problems, covers 23 pages, and forms one of the most interesting chapters. X. is on the conditions of chemical equilibrium in geology. The book is interesting throughout and will appeal to the advanced mineralogist, physicist, chemist, and geologist." *Mines and Minerals*, Sept., 1910. (A. R.)

Accidents in their Medico-Legal Aspect. Edited by D. Knocker. Roy. 8vo. *Baillière.* Net 30s.

Allen, J. R., and Bursley, J. A. Heat Engines. 8vo *Hill Pub. Co.* Net 12s. 6d.

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Becker, O. M. High Speed Steel. 8vo. *Hill Pub. Co.* Net 17s

Becker, G. F. The Age of the Earth (Smithsonian Miscellaneous Collections). Roy. 8vo., sd., pp. 28. *W. Wesley.* Net 1s. 6d.

Beringer, C. and J. J. A Text Book of Assaying. For the use of those connected with mines. 12 ed. Cr. 8vo., pp. 468. *C. Griffin.* 10s. 6d.

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Campion (A) Iron and Steel Analysis, Vol. I. Ordinary Constituents. 18mo., pp. 80. *Fraser, Asher* (Glasgow).

Cape of Good Hope. Report of the Senior Analyst for the year 1909. Folio, sd., pp. 32. *W. Wesley.* Net 1s. 6d.

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Philip, Lady, and Rastall, R. H. A Text-Book of Geology. 8vo., pp. 510. *E. Arnold.* Net 16s.

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Abstracts of Patent Applications.

(C.) 509/09. Francis Joseph Watts (1), Fraser Joseph Alexander (2). Improvements in electrically actuated apparatus for cleaning and scouring amalgamating plates or surfaces. 10.11.09.

This patent application covers an electric motor with gearing attached for rotating a suitable brush for cleaning amalgam plates.

(C.) 525/09. Alfred George Newkey Burden. Improvements in guides for the stems of the stamps of stamp mills. 17.11.10.

This application refers to metal guides for stamp mill stems constructed as a base piece permanently bolted to the guide girt and the guide sleeve proper constructed to seat into a dovetailed groove in the base piece and secured thereto by a single bolt.

(C.) 122/10. William Arthur Caldecott. Improvements in means for transporting and de-watering pulps. 23.3.10.

This application relates to dewatering sand pulp in connection with "sand-filling." Sand from treatment vats is sluiced or trucked to a sump and thence pumped to dewatering cones placed above a borehole delivering to the mine. The overflow is gravitated or pumped back for re-use with a further quantity of sand. Claims 1 and 2 relate to the combination of means of transport, with dewatering classifiers having in view the saving of pumping back surplus liquid and the confining of the carrying liquid to a limited circuit.

(C.) 384/10. Hans Peter Hansen. Improvements in and relating to the manufacture of pipes, tubes and the like from wood and in apparatus therefor. 5.8.10.

This invention relates to a process for the manufacture of wooden tubes and pipes by means of wooden shavings spun round a collapsible mandril which is inflated with air or other fluid during the spinning operation. This mandril can be collapsed to permit of the removal or shifting of the spun pipe, etc.

(C.) 413/10. Elizabeth Branston Parnell. Improvements in the treatment of ores. 19.8.10.

This application relates to an improved process of treating complex ores for the recovery of the contained metals. The essential features of the process consist in roasting the ore to remove volatile constituents, converting the copper to sulphate, and iron to oxidised condition, then removing iron as dry powder and boiling the residue in water and sulphuric acid by means of the "hot blast," then removing the liquor and washing thoroughly when the precious metals are recovered by solution in cyanide solution or other known means.

(C.) 426/10. Emil Deister. Improvements in ore concentrators. 26.8.10.

This application relates to an apparatus for concentration of finely divided ore by means of a shaking table. Different portions of the surface of this table have different inclinations, and their arrangement is specially adapted to the concentration of slime.

(C.) 447/10. John Collins Clancy. Improvements in the treatment of precious metalliferous ores. 9.9.10.

This application relates to improvements in the cyanide process which consist in methods of regenerating cyanide solutions or converting cyanogen bearing compounds into solvents for gold by the addition of amide compound in the presence of an electric current.

For example, carbamide added to potassium cyanate and electrolysed, will form a solvent for gold in alkaline solution.

Again, the electrolysis is performed while the mixture is in contact with the ore to be treated, of which several examples are given with modifications, such as using halogen compounds to promote oxidation, etc.

(C.) 455/10. Henry Hooke. Improvements in centrifugal separator or filter. 14.9.10.

This application relates to a centrifugal separator or filter in which the "basket" is provided with a filtering medium, and is revolved by a hollow shaft, through which is a solid shaft carrying a conveyor revolving at a higher or lower speed, thereby causing continuous discharge of the separated solids. The separator differs from the ordinary centrifugal machine in being placed horizontal instead of vertical, and in having means for continuous discharge.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 478/10. Frank William Girdler-Brown. Improvements in guiding skips, cages, and the like, in mine shafts and otherwise. 29.9.10.

(C.) 479/10. Robert J. McNitt. Improvements in the electrolytic manufacture of alkali metals and alloys of alkali metals. 30.9.10.

(P.) 482/10. James A. Lamberton. An improvement in safety catches for use on any form of conveyance used in mine shafts. 4.10.10.

(P.) 483/10. George Francis Jones. A process for producing genuine white lead, basic hydrated carbonate of lead (commonly known as white lead) by the quick or precipitating process. 5.10.10.

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Nisbeth, G. F., *l/o* Colleen Bawn; Coquet and Mines (per Private Bag), c/o S. Cohn & Co., Antenior Halt, Rhodesia.

North, W. J., *l/o* Roodepoort; May Consolidated G. M. Co., Ltd., P. O. Box 50, Germiston.

Olds, H. F., *l/o* Gadzema; Kimberley Tyne G. M. Syndicate, Ltd., Hartley, Rhodesia.

Parsons, C. E., *l/o* England; P. O. Box 213, Bulawayo.

Pope, W. T., *l/o* Johannesburg; Machavie G. M. Co., Ltd., Klerksdorp.

Reynolds, J. P., *to* Glasgow Mine, Gatooma, Rhodesia.

Roberts, S., *l/o* Krugersdorp; P. O. Box 361, Salisburg, Rhodesia.

Ross, A. W., *l/o* Day Dawn; Vivien G. M. Co., via Lawlers, W. Australia.

Sartorius, K., *l/o* Geduld; May Consolidated G. M. Co., Ltd., P. O. Box 50, Germiston.

Simpson, F. L., *l/o* Lonely Reef Mine; (Private Bag), Inyati, Rhodesia.

Summers, Edward, *l/o* Klerksdorp; Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights.

Sydenham, C. T., *l/o* Germiston; Luipaardsvlei Estate and G. M. Co., Ltd., P. O. Box 53, Krugersdorp.

Taylor, W., *l/o* Barberton; P. O. Box 43, Ventersdorp.

Thomas, Arthur, *l/o* Roodepoort; Penrice, Park Road, Redruth, Cornwall.

Thurston, G. H., *l/o* Johannesburg; Consolidated Goldfields of S.A., Ltd., 8, Old Jewry, London, E.C.

Wade, R. A., *l/o* Bulawayo; Dawn Mine, Turkois P. O., Gatooma, Rhodesia.

Wilmoth, L. J., *l/o* Germiston; Sheba G. M. Co., Ltd., Eureka.

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OF THE
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OF SOUTH AFRICA.

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VOL. XI. DECEMBER, 1910. No. 6.

Proceedings
AT
Ordinary General Meeting, December 17, 1910.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the South African School of Mines, on Saturday evening, December 17th, Dr. James Moir (President) in the chair. There were also present :—

39 Members : Messrs. W. R. Dowling, F. F. Alexander, K. L. Graham, Tom Johnson, A. Richardson, G. O. Smart, A. Whitby, H. A. White, W. A. Caldecott, E. H. Johnson, A. McA. Johnston, H. A. Adams, A. Avent, W. Beaver, E. Blewett, J. H. Dinwoodie R. Gascoyne, R. Fiander, J. H. Johnson, G. A. Lawson, W. Lea, Geo. Melvill, J. S. Murphy, C. F. Parry, F. D. Phillips, J. F. Pyles, G. A. Robertson, A. Salkinson, C. O. Schmidt, A. Schwarz, G. H. Smith, Ralph Stokes, A. M. Stuart, A. Thomas, Chris. Toombs, C. F. Webb, E. M. Weston and A. Wilkinson.

13 Associates and Students : Messrs. H. Abas, N. S. Arnot, W. E. Burrows, J. Cronin, H. L. V. Durel J. Gibson, B. W. Holman, G. W. Leach, L. T. Leyson, G. F. Mathews, P. Scatterty, H. Stadler, and F. Wartenweiler.

10 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as printed in the November *Journal*, were confirmed.

NEW MEMBERS.

Messrs. A. Salkinson and W. Beaver were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows :—

GRIFFIN, NOEL HENRY, The Charterland and General Exploration and Finance Co., Ltd., P. O. Box 592, Bulawayo, Rhodesia. Engineer and Manager.

HINKS, CYPRUS, Wolhuter Gold Mines, Ltd., P. O Box 1160, Johannesburg. Cyanide Foreman.

RIDER, JOHN HALL, M.Inst.C.E., M.I.E.E., M.I.Mech.E., Messrs. H. Eckstein & Co., P. O. Box 149, Johannesburg. Consulting Electrical Engineer.

The Secretary : Since the last meeting of the Society the following have been admitted by the Council :—

As Associates.—

ANDERSON, ALEXANDER, Chin Mine, Mount Darwin, Mazoe, Rhodesia. Engineer.

AYMARD, JOHN LAW, M.R.C.S. (Eng.), L.R.C.P. (Lond.), Castle Mansions, Eloff St., Johannesburg. Physician.

BANKS, JOHN HOWES GRIPPER, Waihi G. M. Co., Ltd., Waihi, New Zealand. Metallurgist.

MILLS THOMAS LANDALL, 128, Gower St., Bedford Square, London, W.C. Mine Surveyor, Falmouth Consolidated Mines, Ltd. (*Transfer from Students' Roll.*)

SIMS, CHARLES FREDERICK, 14, Kenny St., Waihi, New Zealand. Metallurgist.

As Student.—

COCHRANE, EDWIN ARTHUR, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Learner.

GENERAL BUSINESS.

The President : I have to announce that Mr. E. H. Croghan has resigned from the Council, and that at our last meeting Mr. Stuart Martin was elected in his stead.

A PLEA FOR UNIFORM TIME ON THE RAND.

The Secretary read the following :—

Mr. W. M. Butler (*Member*) : This is a question that affects our business population as well as all workers on the mines, but my excuse for bringing the matter before this Society, if excuse be needed, is that the mines are the principal offenders. I have often sought a reason for the outlandish time kept on various properties but have not yet found a satisfactory one. The idea put forward in some quarters that if one blows the six o'clock hooter at 5.30 a.m. it is easier to get the Kafirs turned out on shift, is surely too childish to entertain, and savours more of my country of origin than of the Transvaal. It recalls the story of the parish priest of the west of Ireland who announced that "next Wednesday being Good Friday, there will be no first mass, and the eleven o'clock mass will be held at seven." Is there any reason why we should not have a uniform time? If not, it is surely none too soon that this haphazard way of regulating our hours was looked into seriously and amended, for I submit it belongs to an age long past and is unworthy of our present status as the hub of the gold mining industry of the world. It is rather trying to a visitor who has an appointment to keep, say at one of the mines on the East Rand, to find on arrival at Park station that the two railway clocks in sight differ by seven minutes, while the post office chimes ring quite a different hour; but he is hardly prepared for yet another shock, when on reaching his destination he discovers that the mine time is half an hour ahead of railway time and that he has missed his appointment. If the various mine times were constant so much confusion would not arise, but there seems to be no system regulating the present happy-go-lucky method, and I have known the difference between post office time and that kept on a mine to vary from five to forty-five minutes in the space of a month. One can never tell the time by a mine hooter, as these musical instruments seem to break forth at all hours of the day and night just when the humour takes them. I know of one household where three clocks are kept, labelled respectively "Railway time," "A—— G. M Co.'s time," and "B—— G. M. Co.'s time." This is the only means of regulating the movements of the various members of the family who work on different properties.

We have an Observatory in Johannesburg where the correct time is kept, but it does not seem of much practical use to the community at large. Kimberley has its daily gun, with the result that all its hooters send forth their dulcet notes together—whereas our measurements of time vary from "the length of candle burned" method of the latest recruits of our town police to the watch (home repaired) of the engine driver who manipulates the sirens on our mines. A good gun fired from the Observatory ought to be easily heard in Germiston and Florida. By this means and with the co-operation of the various mine managers we should soon remedy the present state of chaos and remove the stigma from our otherwise up-to.date reputation.

The Secretary: I took the opportunity of sending an advance copy of this to Mr. R. T. A. Innes, of the Government Observatory, who has kindly sent us the following contribution:—

Mr. R. T. A. Innes (*Director, Transvaal Observatory*) (contributed): Mr. Butler correctly states that exact time is kept at the Transvaal Observatory. I should like to add to this, that the clock in the Central Telegraph Office, at the Johannesburg Post Office, is controlled by an hourly signal, and that several subscribers are regularly furnished with the correct time from the Observatory; one of these has a private wire so that at any time the correct time can be ascertained.

At the beginning of 1905 the Johannesburg Chamber of Trade endeavoured to arrange the firing of a time-gun at 1 p.m. daily. This led to an offer from the Post Office to consider favourably any proposals or suggestions on the subject from the Mayor or Town Council. The Post Office would be prepared to provide for the running of the necessary electric wires and for firing the gun, but the Town Council would have to make all other arrangements, that is to supply the gun and ammunition, maintain the former and to arrange for charging it daily. This offer was duly considered by the General Purposes Committee, and after consideration was declined on account of the expenditure (of I believe about £20) which would be involved. I might add that some of the railway clocks on the Capetown suburban lines are controlled either directly or through the Post Office there by signals from the Cape Observatory.

When the people of Johannesburg, or the mine authorities demand the correct time, I do not think that they will be able to say that the Government is unwilling to afford them every facility in its power. At present any telephone-subscriber can be supplied with the correct time for a merely nominal yearly subscription. I should, however, like to see the Johannesburg Municipality take some interest in the establishment and maintenance of a daily time-gun, and to arrange that in its projected Town House all its clocks shall be suitably controlled.

The President: I think the Society fully agrees with Mr. Innes. The saving of £20 is a parsimonious idea in comparison with the advantages we would gain. I think the Town Council should endeavour to finance some scheme for the purpose of keeping proper time on the Rand.

DANGER OF EXPLOSIONS OF AMALGAM BARRELS.

Mr. W. R. Dowling (*Vice-President*): I wish to draw our members' attention to some danger attending the use of barrels used in mill clean-up rooms for grinding black sand and other rich products.

At times gas is generated in the barrel during the running and escapes with considerable violence when the barrels are opened. An unfortunate accident due to this violent escape of gas whereby a man lost the sight of an eye is my excuse for bringing this matter forward. There is no vent through which gas if generated may escape, and it is difficult to fix a device on a revolving machine which will carry off the gas and not be choked by the sticky material being treated. The accident, although the barrel had been open long enough to allow gas to escape, seems to have been due to the plug being shorter than the thickness of the side of the barrel. The recess thus formed filled with sticky black sand during grinding, which was strong enough to withstand the pressure of the gas after the plug had been removed. The attendant wishing to add lime to the charge broke the skin of black sand in doing so, and the violent escape of gas threw the lime back into his face, with the disastrous result mentioned above.

Coming now to the cause of the trouble, it appears that gas to a dangerous pressure is generated only when acid material containing partially oxidized pyrite is treated. The finely divided iron produced from the cast iron barrel and the steel balls, together with that possibly in the material itself, being ground in the presence of iron pyrite results in the formation of ferrous sulphide.* Ferrous sulphide and finely divided iron acted on by the acid of an acid charge would result in the formation of sulphuretted hydrogen and hydrogen :—

$$FeS + H_2SO_4 = FeSO_4 + H_2S$$
$$Fe + H_2SO_4 = FeSO_4 + H_2$$

That the gas is explosive has been proved before now, when a naked light was used in the inspection of a barrel after opening.

The remedy to prevent excessive pressure seems to be the use of sufficient lime with charges to neutralize the acid. Care should be exercised in opening to allow gas to escape gradually, and the barrels should not be more than half full. Attempts to collect gas from two barrels of the same material, one without lime and the other with an excess, were not successful probably owing to the charges not being sufficiently acid originally. The excess lime barrel, however, had a partial vacuum. The use of sufficient lime seems to have a beneficial effect in collecting amalgam and mercury in the final separation of the amalgam from the sand.

To elucidate the matter further a parcel of die sand was watered and turned periodically for a forthnight until the sand was acid. The lot was then divided into two approximately equal parts, transferred to two separate barrels and equal quantities of water added to each. One charge was rendered alkaline with lime. The barrels were then closed and run for three days. Samples of the gas in the barrels and of the liquid were taken and analysed by Mr. McArthur Johnston, and showed the following percentages by volume:

Original acidity of charge per ton of sand $= 0{\cdot}0107\%$ H_2SO_4.

	Without lime.	With lime.
Carbon dioxide (CO_2) ...	0·03 %	Nil
Oxygen (O_2)	Nil	4·4 %
Hydrogen (H)... ...	42·80 %	8·6 %
Sulphuretted hydrogen (H_2S)	Trace	Nil
Nitrogen (by difference)..	57·17 %	87·0 %
Solution	0·0073 % H_2SO_4	0·0026 % NaOH

There was a considerable pressure of gas in the acid barrel but very little in the alkaline barrel; in fact another experiment showed a vacuum in the alkaline barrel possibly due to absorption of oxygen by pyrite and nascent hydrogen. From the foregoing it appears that the gaseous pressure in amalgam barrels is mainly due to hydrogen liberated by the action of acid black sand upon iron and increased by heat, whilst the absorption of oxygen and addition of lime produce the reverse effect.

Mr. W. Beaver (*Member*): I would also like, with Mr. Dowling, to add a word of warning to those whose duty it is to empty amalgam barrels. It is well known that a considerable quantity of gas is generated during the process of grinding black sand, etc., particularly when little water is used, therefore extreme care is essential when opening. I have often found the pressure sufficient to lift a door (weighing approximately 50 lb.) about 1 in. from its seat. I dislike to see the small plug used, which could, on being unscrewed, blow out with great force, which makes it extremely dangerous to the operator. I am conscious of having neglected a duty due to my fellow mill-workers in not having made known sooner an experience I had with barrel gas. Some old battery chips had undergone the usual treatment for something like 36 hours; when the barrel was opened, it was seen that a large amount of gas had been generated, sufficient to lift the door right out, while it was being raised with a crowbar. After an interval of about five minutes I looked into this barrel; being unable to see, I lit a match immediately over the opening, the result being a violent explosion, with a blue flame, which severely burnt me about the hand and face, laying me up for a week. I considered myself most fortunate it did not blind me. The violence of the explosion can be gathered from the fact that it blew my cap 10 ft. away. Needless to say, a light and a barrel will never again figure in my experiments. No matter how long a barrel has been open, I would advise great caution when a light is needed, and both doors should be open to allow a current of air to pass through, and so expel any gas that may be there.

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* See this *Journal*, Vol. II., July, 1897, p. 99.

THE EDITORIAL COMMITTEE.

Mr. G. Hildick Smith (*Member*): There is one matter which may come under the head of "General Business," namely, that we should have a mining man on the Editorial Committee. We now have special mining meetings, the idea of which is, I presume, to have more mining papers than before. The idea of having practical mining papers is not that they should be written from a journalistic point of view as that they can be read by the general public. I take it that they are written for mining men, and should contain mining terms or miner's slang, so that they can be read by practical miners and students. At present our mining papers are edited in such a way that mining terms are cut out. I should like to propose that a mining man should be put on the Editorial Committee.

The President: The Council recognizes that the Publications Committee and Editorial Committee are both rather weak in that department as compared with the other two, but on the other hand I am afraid that in the Council you will not find much sympathy with your view that the Society should accept imperfect publications. It is so exceptional in a scientific Society to have slang used—I cannot use any other term—where perfectly good English words can be found to express the meaning.

Mr. H. A. White (*Member of Council*): There seems to be some slight confusion between mining terms and miners' terms!

"CHEESA STICKS."

The Secretary read the following:—

Mr. J. Moyle Phillips (*Member*) (contributed): Mr. Weston is to be thanked for bringing before the Society, and therefore making public a method of lighting fuses other than with a "cheesa stick."

As Mr. Johnson infers, it is certainly not a *new* method, as I have personally used the same method in this country for the last 20 years. The only difference between my method and that of Mr. Weston's is that I do not use tobacco bags to wrap around the candle, but the paper in which the packets of candles are packed. It is not necessary also that the tobacco-bag or paper should be *tied* around the candle as it can be securely fastened thereto by running the hot grease from the candle along the edges of the tobacco-bag or paper, and thus forming a secure fastening. The other difference between Mr. Weston's method and mine is that he cuts two nicks in the fuse whereas I only cut one, but certainly the second nick is a safeguard against misfires. I quite agree with Mr. Weston that the Committee's suggestion of splitting the end of the fuse down to about 1 in. is certainly not a *good* suggestion if only for the reasons suggested by Mr. Weston.

One has only to be the last of a line of blasters coming along a drive at blasting time each one carrying a lighted "cheesa stick" to be convinced that the said stick should be prohibited except in very wet places, or where the water drops from the roof of the working place.

NOTES ON THE MATTE ASSAY.

By L. J. Wilmoth, A.I.M.M., M.Am.I.M.E.

Since reading a paper on the assay of battery chips and screens before this society some two and a half years ago* I have at odd times done a little work on the application of the method mentioned there to the assay of copper bearing material. In spite of the adverse criticisms of several members I have not had any occasion to alter the opinions expressed in that paper: in fact time and more experience have deepened my conviction that the use of a matte is under certain conditions of the greatest value in assaying. I have carefully gone through all the literature available in connection with the assay of gold bearing material and as yet have found no mention of matte except as an unintentional product in the assay of such material as pyrite In all cases the production of a matte under these circumstances is condemned, and rightly so, as it is always accompanied by unreliable results. The use of a matte, however, for the removal of certain base metals, when proper care is exercised in its production, seems to offer the assayer a convenient and accurate method of handling base ores and products. In my previous paper I referred to and described theoretically and practically the use of the matte in the assay of such troublesome material as battery chips, which consist of iron and steel, the idea having been borrowed from South Wales, where a similar method is used for the assay of copper drillings. I have searched in vain for a description of the method in use there, and so have developed a method, making use of this principle. The usual method of assaying copper drillings, etc., is by the scorification and combination methods as described in several text books. I have found little difference in the results obtained by either of these methods for the determination of the gold values. The matte assay compares very favourably indeed with both, even when as much as 1A.T. of the copper drillings is taken for each estimation. The principle of the matte assay is exactly the same as that of the assay of battery chips, the same charges yielding excellent results with the copper drillings. The following charge was used with all the experiments on the

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*See this *Journal*, February, 1908.

metallic copper, which in all cases was in the form of drillings:—Drillings 10·0 gm. litharge 1 A.T., sodium carbonate 1·5 A.T., borax 1·0 A.T., silica 0·5 A.T., charcoal 1·25 gm., sulphur 8·0 gm., and a stout iron nail. The average result from three assays was 10·417% fine gold. The mattes on scorification gave 2·8 mg. of fine gold, and the slags ·17 mg. This equals a total of 10·446% fine gold in the sample. This might be considered a severe test for any method. Another batch of assays was made on the same sample by scorification, taking 0·5 gm. of the sample for assay, and the average for these assays was 10·31% of fine gold. The treatment of the slags, however, yielded another ·31 gm., thus bringing the corrected value to 10·37% fine gold. Still another batch, using 0·5 gm., was taken and run by the combination method, yielding better results than either of the other two methods, as the average of the results was 10·49%.

Comparing these figures we have the following:

Uncorrected Assay.

Matte.	Scorification.	Combination.
10·41%	10·31%	10·49

Corrected Assay.

Matte.	Scorification.	Combination.
10·44%	10·37%	—

To still further test the method on the copper drillings I made three more assays with the same flux, but of a different sample, with the following results:—

Average from lead button 112·64 oz. per 2,000 lb.

Average from matte 14·80 dwts. per 2,000 lbs., on original sample.

Average from slag 0·24 dwts. per 2,000 lbs., on original sample.

It would therefore seem that the use of a matte with the proper precautions would be admissible with this type of sample.

I next turned my attention to the assay of rich copper bearing material in the form of copper matte carrying between 25 and 30% copper and some lead, the material being fluxed in the following way:—Matte 0·5 A.T., sodium carbonate 1·0 A.T., litharge 1·0 A.T., borax 1·0 A.T., silica 1·0 A.T., sulphur 0·1 A.T., charcoal 1·5 gm., and one 6 in. nail. The charge was fused at a fairly high heat and became perfectly fluid. On cooling, the button with the matte adhering was detached and the slag placed on one side for re-assay. The lead button and the matte were separated, the separation being perfectly clean and presenting no difficulty. The matte was put on one side for assay by scorification. The lead button gave on cupellation: Gold 3·55 mg., silver 71·02 mg. The matte on scorification with test lead and a little borax gave:—Gold *nil*, silver 5·65 mg. The slag was fused with a little sodium carbonate, litharge, borax and the necessary charcoal, and was given a wash with litharge and charcoal before pouring. The results were as follows:—Gold *nil*, silver 0·95 mg. These results are shown in tabular form thus:—

	By scorification of lead button	By scorification of matte	By fusion of slag.	Total	Assay equivalent per ton
	mg.	mg.	mg.	mg.	oz
Gold …	3·55	*Nil*	*Nil*	3·55	7·10
Silver..	71·02	5·65	0·95	77·62	155·24

A few experiments were next made on a sample of litharge dross from the pan furnace, carrying from 20·30% copper, the rest being mainly oxide of lead. This was fluxed as follows:—Dross 0·5 A.T., sodium carbonate 1·0 A.T., litharge 0·75 A.T., borax 1·0 A.T., silica 0·5 A.T., sulphur 0·25 A.T., charcoal 1·5 gm., and an iron nail. The amount of sulphur in this case was kept high, much higher than theoretically required, but with no injurious results, as the iron nail acts as a check, preventing the passing of lead into the matte. This charge fused without trouble, pouring with great fluidity. The slag and the matte were put on one side for assay. The cupel from the actual assay barely showed a stain of copper, that from the assay of the matte was heavily stained, and that from the slag was perfectly clean. The matte and the slag were treated the same as in the assay of the matte. The following were the results:—From assay: gold 95·00 mg., silver 90·00 mg.; from matte: Gold 0·12 mg., silver 16·72; from slag: gold 0·42 mg., silver 1·48 mg. These gave a total of gold 95·84 mg., silver 108·21 mg. on an assay value of gold, 191·68 oz.; silver, 216·42 oz. per ton.

Taking the two preceding samples and tabulating the results we have:—

		Uncorrected Assay	Corrected Assay	Yield in Uncorrected Assay
		oz.	oz.	
Matte …	Gold	7·10	7·10	100 0
	Silver	142·04	155·24	91·4
Dross …	Gold	190·00	191 68	99·1
	Silver	180 00	216·42	84·9

The application of the matte assay in assaying copper gold bearing ores and products presents no difficulty and is of great advantage when the percentage of copper is high and the gold value low. In this case the method is superior to the ordinary scorification method, as larger charges can be taken and the difficulty, arising in the latter process from the smallness of the gold button, is overcome. This advantage is well illustrated by the assay of a slag carrying some 20% copper. A scorification of such a slag gave 48·5 dwts. and a matte assay 53·0 dwts. per ton.

I trust the results of these few experiments will be of some interest to the assaying members of this society.

The President: I have great pleasure in moving a hearty vote of thanks to Mr. Wilmoth for his interesting new results.

GRADING ANALYSES AND THEIR APPLICATION.

(Read at May Meeting, 1910.)

By H. Stadler (Associate).

REPLY TO DISCUSSION.

Mr H. Stadler (*Associate*): The fascination of the subject hardly accounts for so many detailed contributions of value as have been made: it is rather professional pride which keeps interest alive in this important but trying and tiresome question. Since the contributions to the discussion have not been simultaneously printed here and in London, I am obliged to write my reply in such a fashion as to form a more or less independent whole, whereby I can not avoid going over some parts of the ground again.

I.—GRADING ANALYSES.

Standard Grades.—To get a reliable basis for comparison and measurement of crushed products, we must first agree on a logical system of classification, based on a definite mathematical foundation, even at the risk of worrying critics with decimals. It must be completely independent of any particular actual method of measurement, which arise from the use of existing screens or hydraulic methods, and must not be itself dependent on the errors of the instruments used.

Since the extraction of gold by amalgamation (or chemically) is mainly dependent on the surface exposed by crushing, it has been suggested that this standard should be based on the exposure of surface; but amongst the many reasons which made me decide for classification by volume (or weight) are the following:—

1. All assay values are referred to the unit of weight.

2. The energy absorbed in crushing being proportional to the volume, the ordinal numbers of the grades represent at the same time the relative energy required for reducing unit volume (or weight) to the respective grade.

3. Closer accordance with classification by elutriation is obtained, which method of classification is principally determined by volume and weight. The elutriation method is the only method possible beyond 200 mesh.

4. It is interesting to note that in sorting mixtures of pieces into groups by hand, all persons automatically roughly classify them in steps of equal weight or volume, showing that this is a natural classification. I do not claim any great scientific value for this fact.

The concentration of gold values in certain grades, the physical properties of the impalpable sands and many other facts also prevent the extracting processes from acting proportionally to the amount of surface exposed to the attacking agents, and this proportionality is therefore more apparent than really existing in practice. The particles of fine gold are of definite size which varies with the particular ore under consideration, and encased in a gangue which only needs breaking up to liberate them for metallurgical treatment, and a further reduction of the matrix does not improve extraction. The surface area increases so rapidly that practically the whole exposed surface, in a pulp, consists of surfaces exposed in the finest grades which, being produced quite uselessly, can hardly be considered as a criterion for the metallurgical efficiency.

Just as co-ordinates in analytical geometry are a mere matter of convenience, so the determination of the sizes and steps in grading is simply a question of convention. The calculations give the same final results in all cases. Once, however, we have adopted a particular standard of measurement, we must stick to it. We cannot calculate crushing efficiency on the basis both of surface exposed and reduction in volume. Volumes and surfaces vary in different ratios with the size of the particle, and the rate of increase of one does not, in itself, imply a corresponding rate of increase or decrease in the other, any more than a given rate of increase in temperature implies a corresponding rate of increase of steam pressure.

Relation of Screen Products to Standard Grades.—To classify the ground products in accordance with the system adopted, any practical method may be employed but, for obvious reasons, screening is the only one that will be considered here.

The screen products are actually of irregular form but, for the purpose of mathematical study, I have assumed that the average particle of each group is a regular polyhedron of equal weight or volume. As Mr. McDermott remarked in discussing the matter, I adopted the cube for the sake of mathematical simplicity. The sphere would have been equally suitable from the mathematical standpoint, but the reduction of a sphere to smaller spheres is not easily pictured by the mind, while the reduction of a cube to smaller cubes is easily imagined.

If the diameter of the geometrical representatives of the irregular particles is taken as clear mesh aperture, it follows that the mathematical relations of the particles are determined by the respective mesh aperture. This is really a truism, as the products of screens with a constant ratio of wire and mesh as those of the I.M.M. are similar in each grade to those in every other, differing only in scale.

It was interesting, however, to note how far this last is actually true in an extreme case, such as that of ore coming from a jaw breaker and containing a large proportion of "so to speak small lead pencils or pancakes." With this in mind, I carried out some actual experiments in a a rough way only, as I attached no great importance to them, but as I have been asked for them I give the results of these experiments for what they are worth :—

Grading				No. of counted pieces	True Weight p. piece	Weight of theoret. cube sp. Gr. 2·7	Ratio
Grade	Mesh p. lin. in.	Mesh-aperture					
EU.	No.	in.		No.	grms.	grms	R
0	–	1 in.	Commercial Screens.				
1				75	14·099	22·1211	0·63
2	–	3/4"					
3				320	3·693	5 5304	0 67
4	–	3/8"					
5				608	·8695	1 38256	0·63
6	–	1/4"					
7				865	·2312	·34565	0 67
8	–	1/8"					
9				1870	·0535	·08640	0·61
10	5	0·10	I M M. Standards				
11				1760	·0114	·0216	0 53(?)
12	8	·062					
13				630	·0032	·0054	0·59
14	12	·0416					
15				634	·00079	·00135	0 58
16	20	·025					
17				950	·00021	·00034	0·63
18	30	·0166					

From the figures in the table, it appears that the actual and theoretical results do not vary even as much as might have been expected for such an extreme case. It is probable that just as many long and heavy particles get through the mesh endways, as there are similarly shaped lighter particles which are stopped by striking the screen broadside on; and that reduces error to a minimum, even in widely differing qualities of rock. However, imperfections of this kind are unavoidable inconveniences connected with screening in general. Since we are dealing only with relative values, the absolute sizes of the particles which differ with different ores, may be ignored. All we need to know is that the mesh aperture is representative or proportionally representative of the relative average values of the irregularly shaped particles of each grade.

Seriations of Standard Screens.—If we base our system of grading on a regular reduction of the particle to half its volume at each step, it follows that the mesh apertures must be decreased at each step in the ratio of $\frac{1}{\sqrt[3]{2}}$ or, if we omit the alternate grades of the series so obtained, in the ratio of $\frac{1}{\sqrt[3]{4}}$. The cube root series is therefore not a matter of convenient choice but the necessary mathematical consequence of the reduction scale assumed.

Although I advocate strongly the use of the cube root series for the system of standard *grades*, it is not essential that the same series should be used for standard *screens*: any empirical, even irregular series of mesh apertures may be used, provided the accurate sizes of these mesh apertures are known. The only advantage in having the screen series in exact accordance with the theoretical grade series is in simplifying calculations. The advantages are similar to the advantages obtained by using the decimal system instead of the British standard units of measurement. Weights, measures of length, etc., show no regularity in steps according to a mathematical ideal, but with both systems equally correct results can be obtained. Since the full set of screens will, in practice, seldom be required, whilst on the other hand a closer grading over a limited range is often wanted, there was really no necessity to scare engineers by magnifying the small existing deviations from mathematically correct seriation and by plotting them in form of diagrams consisting of curious looking spirals and alarming earthquake records.

I.M.M. Standard Screens.—Mr. H. L. Sulman and others, responsible for the I.M.M. standards, doubt the usefulness of the proposed cube root series as being based "upon ideal factors which had no necessary mathematical connection with screen apertures" and yet, compelled by the same logic which is the basis of this series, they arrived empirically at a practically identical reduction scale. With the exception of a few odd screens the reduction ratio of the I.M.M. series between the limits of the 20 and 200 mesh screens is, as shown by Mr. Howard Higgins, practically constant and, with small exceptions, equal to the cube root of 2. The mesh numbers 5, 8, 12, 20, 30, 50, 80, 120 and 200, determine a reduction scale at the ratio of 1 to 4, which is practically identical with the alternate series of the theoretical series. The mechanical value, for instance, of the 120 mesh, which shows the largest deviation if calculated from the aperture (S) by the formula: $EU. = -10 \log S$, is (for $S = 0·0042$) = 23·767 EU. against 24, the ordinal or EU. number of the standard grade. In the same way we obtain for the 80 mesh (0·0062 in. aperture) 22·076 against 22, and the arithmetical mean representing the mechanical value of the grade determined by these two screens is therefore 22·921 EU. instead of 23·0 EU.

The mechanical values of the battery pulp-grading given in paragraph 31, calculated in both ways, compare as follows:—

Grading Analyses.		Ordin. No. or Mech. Value.			
		Correct Grade.		Nearest I.M.M. Screen.	
I.M.M. mesh.	Wghts.	per Grade	Total	per Grade	Total.
p lin in.	%	EU.	EU.	EU.	EU.
	19·9	13	2·59	13·0	2·59
12		14		13·81	
	12·1	15	1·82	14·92	1·81
20		16		16·02	
	9·7	17	1·65	16·91	1·64
30		18		17·80	
	11·5	19	2·19	18·90	2·17
50		20		20·0	
	10·9	21	2·29	21·04	2·29
80		22		22·08	
	2·6	23	0·60	22·93	0·59
120		24		23·77	
	4·2	25	1·05	24·90	1·05
200		26		26·02	
	29·1	27	8·15	28·0	8·15
	100·0		20·34		20·29

The experimentalist has the choice of taking the tabulated values of the standard grades as identical with those of the nearest I.M.M. screen or of taking the mathematically correct values of the I.M.M. standards. If he selects the former, the error introduced is unimportant and certainly far beyond the limits of accuracy of all the practical applications of grading analyses mentioned in Mr. Hoover's paper.

The I.M.M. series does not pretend to mathematical perfection with regard to the stepping of the screens. It leaves in this matter a free hand to the metallurgist to make his own selection in meeting the requirements of each individual case. The chief purpose of the series is to introduce a uniform instrument of measurement of the greatest accuracy obtainable in practical manufacture, so as to get accurate comparisons of grading analyses made anywhere in the world. The institution has carried out this purpose, involving much painstaking, hard and tedious work and considerable expense, and has put on the market a series of laboratory screens which are scientifically faultless.

All mining engineers should now unite to make a success of grading unification by using these screens exclusively for all scientific and experimental work. The full utility of this screen series can only be obtained by its universal adoption.

Mr. McDermott, Mr. Sulman and other speakers, have so ably explained the indubitable merits of the I.M.M. standards that little remains to be said. An example may, perhaps, show most strikingly the great need of correct and uniform instruments of measurement. The fineness of final pulp is usually judged by the percentage left on the 50 mesh and a percentage or two more or less are earnestly discussed, although the errors due to the inaccuracy of the screens employed, may easily amount to 10%.

2—APPLICATION OF GRADING ANALYSES.

Laws of Crushing.—In his Inaugural Address Dr. Moir drew attention to two dangers which science has to face in the present and immediate future: (1) Over-specialisation, *i.e.*, too wide divergence of the sciences. (2) Dogmatism and blind worship of hypothesis. Certain geological gentlemen caught sinning against better knowledge of chemistry, formed the scapegoat who furnished the case in point for Dr. Moir's charge of over-specialisation, but in speaking to mining men, it would be more fitting to accuse the science of mining of a complete disregard of the great advances in knowledge, made by technological science with which various branches of mining ought to keep in close touch. Fortunately for us mining engineers, the technologists also have little time in which to read mining books and papers. If it were otherwise, they would smile on discovering that so many mining engineers still have blind faith in an old blunder which, under the name of Rittinger's law, has been thoughtlessly reprinted and carried on in edition after edition of mining books since the days of the "Dark Ages" (1850). This "law" is a myth which has never been recognised by science, and it is unfair to Rittinger's memory to connect his name with what is clearly nonsense, and the currency of which in the mining world is due either to misunderstanding or the error of a translator. All that Rittinger himself really said on the laws of crushing, in his famous book on ore-dressing, fills barely two out of about 600 pages, and in these two pages he says clearly: "*the increase of the surfaces exposed is directly proportional to the force required for reducing.*" (These words are printed in italics in the original text.) As Rittinger goes on to say: "and therefore also to the work absorbed in effecting the separations," it may certainly be taken for granted that he supposed the force acting in the fracture planes themselves. Now, this may be conceivable for chemical, electrical, thermal actions and reactions, but not for external forces as applied in mechanical crushing. The "force" is measured by the product of mass into acceleration, whereas "work" is the act of changing the configuration of a system in opposition to forces which oppose this change. Had Rittinger stated that the increase of surface is proportional both to "force" and to "work done," he would have been guilty of mistaking "force" and "work done" as identical. Rittinger's law, therefore, simply states the proportionality of the increase of surface to the *force* required in crushing and not also to the *energy* absorbed, which, in conformity with Kick's law, varies as the volumes. This volume factor which, treated as the mass factor, plays so important a part in connection with work in physics and dynamics, comes into play also in the laws of crushing. For more than a quarter of a century this law has been universally recognised in science. No authority has ever contradicted it, and its truth has been proved over and over again, experimentally and in practice. When Galileo, the great experimental philosopher, discovered Jupiter's satellites with his telescope, he thought that the actual seeing of this small scale demonstration of planetary motion would convince his opponents that the Ptolemaic system of astronomy was wrong, but these bigoted Aristotelians would not look through it and insisted on the truth of their time-honoured doctrines more acrimoniously than ever. This was exactly 300 years ago. But have we changed so very much? The modern scientific testing machines used by technologists record, not only the force required for crushing, but also the energy used in producing the stresses causing rupture, and yet, in the kindred science of mining there are still many engineers who will not admit what they could easily see without spectacles.

One of the contributors in speaking of Kick's law is "not at all satisfied that this commonly accepted hypothesis is correct," another admits that it is "of mathematical interest," and Mr. W. H. Jane finds in it "a very striking example of the possibility of theory, unaccompanied by practical application, leading to false conclusions." The correct application of theory to practical conditions is exactly the true test of technical training, but Mr. Jane finds an easier way of avoiding false conclusions: he does away with theory altogether by excluding from this country laws of nature as "undesirables." By a chain of argument Rumford, Robert Meyer, Joule, Hirn and other physicists, long ago proved that the quantity of heat produced by an expenditure of work is in strict and universal proportion to the work spent and *vice versa*; and for the measurement of heat a unit has been agreed upon called a "Calorie." Now, however, Mr. Jane turns up and says: These laws and theories are not applicable to this country where the local conditions, with reference to quality of coal, barometric pressure, etc., are quite different, and he objects to Kick's law with the words: "This law is sufficiently clear to enable us to discard it at once in connection with the reduction of Rand banket."

Admissibility of Kick's law in practice.—In the hope that by-laws may soon be introduced to make the more important laws of nature workable on the Rand also, let us see whether the average conditions of ores in general, and those of the Rand in particular, agree sufficiently with the premises of Kick's law to justify its application in practice.

Perfectly homogeneous materials do not exist. The definition "of equal technological state" is broader, admitting irregularities of the material as long as they are similarly distributed, so that the average physical peculiarities of particles in different stages are identical. Such irregularities may be fissures, veins of softer materials, cracks caused by natural means or produced by blasting and so forth. If these irregularities are, on the whole, similarly distributed, our calculations are not affected. The same reasoning applies also to conglomerates of hard and soft components. In this latter case the softer material gives way first and is reduced in size more rapidly than the harder stuff, but on this very account the softer particles filter into the spaces between the coarser particles, so escaping further crushing until the coarser particles are further reduced. At any given stage the softer material is therefore a little ahead, possibly only one grade. No segregation, however, takes place, and the crushed product containing slightly finer sizes of the softer material is quite uniform. This equalising correction is specially noticeable when the undersize is returned to the crushing machine, in which case an automatic regulation takes place. The capacity of the machine to take up fresh feed is reduced if the returned amount of harder grit is in excess, and *vice versa*. Similar results are also experienced with wind classifiers in dry-crushing plants, where also a perfectly uniform product is obtained, with the distinction that in this case, instead of the softer particles, the specifically lighter particles are a little ahead in the reduction process.

Before segregation is noticeable in the process of crushing, the difference in the resistance which the various components oppose to comminution

tion must be very great. In fact, no machines have yet been devised in which such a segregation is well performed. In the metallurgy of base minerals such a machine would be of enormous value, but all machines designed for this purpose have failed in practice except in the extremely favourable case of freeing agricultural grains from their husks.

The description "equal technological state" may therefore, be interpreted very widely without affecting the conclusions drawn from Kick's law. Mr. A. W. Warwick, in a rather lengthy article published by the *Mining Herald*, of Chicago, gives us the benefit of his many years' experience in crushing ore, and tells us of certain ores, which having been dry crushed in the first stage, slaked just like quicklime, when afterwards wetted; or which, after classifying, were no more of the same technological state as the original feed. He comes to the correct conclusion that in these and many other cases the application of Kick's law would be absurd and untrustworthy. I should not have mentioned this statement had it not been made in a manner unfair to the members of the Mines Trials Committee, in two ways — namely — first, by drawing the members of the committee into a discussion at all when I had clearly stated in my paper that "I am personally responsible for the mathematical reasoning and considerations derived therefrom together with the opinions expressed," and secondly, by implying that they were capable of completely ignoring most essential facts in the application of physical laws.

It is a factor of the greatest importance that (owing to the averaging that automatically takes place to a great extent in the processes of breaking, underground transport, hoisting, sorting and feeding the battery bins), the battery feed is already practically constant in composition. Considering that in each 5-stamp mortar box about 700,000 drops take place every 24 hours, there is little doubt that, if 50% of softer limestone were added to the hard quartz underground, the battery and final pulps would contain the same percentage of quartz and limestone particles and the relative sizes of the particles would be the same: in other words, the battery and final pulps would be products of equal technological state. The fact that there are over four million of particles of the size of the 80 mesh I.M.M. in a cubic inch of material gives some idea of the averaging which must take place during crushing. The averages of the physical properties of material in such a state of comminution are practically constant and their values are as good as exactly defined figures.

It is a remarkable fact that the behaviour of heterogeneous materials when crushed differ very little from that of any homogeneous material. This seems to bear out the hypothesis adopted by many technologists that the separation of the molecules is entirely effected by shearing and that all the other forces are inoperative until they produce, as a resultant, the shear, which finally does the work. In any case, neither compression nor its variety impact, have any existence as reducing agents, it having been clearly proved that the properties of materials popularly described as hard, brittle, elastic, liquid, and so forth, are merely relative, depending on the pressures to which the materials are exposed.

Rand ore in particular complies fairly well with the term "equal technological state" in the following ways:—The bulk of the ore is mainly quartz in form of pebbles of all sizes. The matrix in which these quartz pebbles are embodied and which, with them, form practically the whole of the ore, gives the same resistance to grinding as the pebbles. One would have to examine a large amount of banket before finding a piece which would give any indication *by the character of the fracture* that the resistance of the quartz pebbles differed appreciably from that of the matrix. The pieces of banket used in tube-mills wear out regularly and smoothly without forming bosses or undulations as they would do if the components differed in hardness. External scratches similarly show equal hardness of the pebbles and matrix.

Impurities such as gold, pyrite, and other minerals of quite different physical structure form such a small percentage of the whole material that their effect on crushing may be neglected. Mr. A. J. R. Atkin was unfortunate in stating that the calcite pebbles of the Meyer & Charlton consume less power in crushing than the harder pebbles from other mines, since these calcite pebbles are almost a geological curiosity, and constitute only about ·001% of the total ore, but I am however much obliged to Mr. Atkin for the publication of a couple of lantern slides which clearly show that irregularities in texture, noticeable in larger pieces of Rand ore are similarly existing and visible under the microscope in the very smallest.

Some of the contributors have remarked that whilst all that the stamps really do is the relatively easy work of freeing existing soft and earthy materials from the harder material enclosing them, they have been unduly credited with the production of the − 80 grade. I am afraid that these critics are under the impression that a portion of the slime chiefly consists of minerals other than quartz. By elutriating the battery slime it can however be proved that the grades obtained even with so low a velocity as 0·63 mm. per second are quite leachable sands which settle quickly in water and which under the microscope are seen to be exactly similar to the sands of coarser grades obtained by screening. It is therefore unlikely that the average technological state of the tube-mill feed has been by the process of classifying appreciably altered through an increase in the percentage content of the harder portion of the ore, as compared with the battery pulps. The specifically heavier pyritic particles naturally predominate in the return, and, being as Mr. Atkin says, "a brittle mineral easily broken," are actually an advantage to the tube mills.

Several critics lay stress on the different working conditions in various crushing appliances and the impossibility of applying one and the same law to all of them. They forget that the object of such laws is to establish a standard by which to measure the relative values of the work done. Of course, different working conditions affect efficiency, and it is just the extent of these effects that we measure by means of grading analyses taken at the inlet and discharge of the machine.

It may be noted in passing that the working conditions of stamps and tube-mills are not so widely different as is generally believed. The following simple example will show this. Assume that a stamp (1,250 lbs. falling weight, 8in. set height and 100 drops per minute) crushes 5·2 tons per 24 hours through a 0·017 in. mesh, producing a pulp of 24·3 E.U. The duty per minute is thus 7·22 lbs. The time required to crush 1 lb. is 0·138 minutes, and during this time 13·8 drops take place. It would be interesting to know how many E.U. per blow can be produced by direct impact under the most favourable conditions. In any case, it is clear that the work done by the above 13·8 blows is very small, and it follows that a large portion of the work must have been done in a very inefficient way, probably by attrition and abrasion of the particles as they are shifted and moved about in the mortar box. As a matter of fact, the particles of a battery pulp are all more or less rounded, with the corners broken off, and in this respect do not differ materially from the particles of the pulp leaving the tube mills.

Methods of Computation of Crushing Efficiency.—If the so-called Rittinger's theory of proportionality of the work done in crushing to the surfaces exposed were correct, the work done would also be proportional to all those functions of the particles which vary in the same ratio as the surfaces, for instance, the areas of the fractures, each fracture being equivalent to two surfaces. As far as I am aware all methods of computation of crushing efficiency hitherto published replace, for the purpose of determining the relative value of the work done, the factor of the area of surfaces or fractures by those of other functions varying in the same ratio and measurable by grading analyses. Messrs. S. H. Pearce and W. A. Caldecott "rather to start an investigation of the subject than to lay down any rules in regard to it," suggested the representation of the efficiency factor by the reciprocals of the diameters of the particles.* Messrs. Klug and Taylor assumed the squares of the diameters† and Mr. R. W. Chapman, of the Adelaide University, proposed as the efficiency factor the number of meshes per lin. in. of any set of screens with a constant ratio between diameter of wire and mesh aperture, such as the I.M.M. standard screens.‡

All these methods being based on an erroneous theory, are bound to give false results, which, though clearly apparent over the complete "curve" are not so visible over small sections of it. In the case, for example, of the small range of grades covering the intake and discharge of tube mill pulp, the error is not so obvious and serious as it is in that of the feed and discharge of a stamp battery, or in those of efficiency comparisons between coarse and fine crushing appliances. In such a case as this latter, if the calculations are based on the surfaces, the results are obviously absurd.

The energy absorbed in crushing (according to Kick's law) being proportional to the reduction in volume, the volume or weight of the particles is the only true basis for establishing the relative values of the work done. However, as the determination of the average weights of the particles of each grade is impossible in practice, we may substitute for the volume any other function which varies as the volume and is measurable by grading such as, for instance, the cubes of the diameters, or better still, the ordinal numbers of the suggested standard grades of the reduction scale having a ratio of ½. It will be shown later on that this theoretically unobjectionable method holds good in practical work without limitation in the range of grades.

Relative Mechanical Merits of Stamps and Tube Mills.—The intention of my paper being primarily to submit the method proposed for the calculation of efficiency to discussion, the question of the relative merits of crushing appliances must stand over, and it is only dealt with here in so far as it enables us to ascertain the correctness and to demonstrate the practicability of the method.

Most of my Johannesburg critics took to heart the incidental remark that tube mills, "under the condition of the example given," have only ¼ the mechanical efficiency of stamps. Mr. H. A.

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* See this *Journal*, Sept. [illegible], and March, [illegible]

† *Journal of the Chamber of Mines*, Jan. [illegible]

‡ *Proceedings Australasian Institute of Mining Engineers*, Oct., [illegible]

White is at a loss to undertand why I did not go a little further, and show by adopting stamps for secondary grinding how three-quarters of the power, now wasted in the tube, could be saved.

The question is reasonable, seeing that he also finds the other statement contained in the same paper, that the efficiency of stamps falls off rapidly with the fineness of the screen used, scarcely in accord with practical experience.

Mr. E. H. Johnson communicates the results of two typical crushing trials conducted at the Cason and Angelo mills, which, he claims, hardly support the contentions of my paper and which, in his opinion, express the position more accurately than my theoretical development of Kick's law. His calculations are based on what I consider a purely imaginary law and on the empiric formula of the – 90 tons. Since I have already, without contradiction, stigmatised these methods of calculation as horribly unscientific, barbarous and ghastly, he will appreciate my courtesy when in criticising his figures I simply say that I am very glad that they are not in accord with mine. The following considerations demonstrate how the enormous increase of surfaces in the finer grades may easily lead to erroneous interpretation. By the statement made by Mr. E. H. Johnson in his contribution that the stamp and tube mill combination produces 9·5% more surfaces than the stamps alone, the impression is—perhaps unintentionally—produced that the *useful work* had been increased in this proportion. The assay value of the ore being given by weight is, however, in both cases the same, and the only gain to be derived from the additional energy spent for the further reduction of the pulp, is to be looked for in the direction of higher extraction, which in the present case, under fair assumptions, amounts to about 1%.

Making use of the data given by Mr. Johnson and completing it by an additional example of single stamp crushing (New Kleinfontein mine), the relative efficiencies of stamps and tube mills, calculated by my method, work out as follows :—

			Double Stage Crushing 9m (·27") (Angelo & Cason-Mills) 10 Stamps to one Tubemill.							Single Stage Crushing			
										1600m. (·015")		1200m (·017")	
Grades.			Battery-pulp.		Entering Tubemill.		Final-Pulp.			34 Stamps 1680 lbs		220 Stamps 1400 lbs	
Mesh & opengs.	Mech. Value of Grade		Wghts.	Mech. Value	Wghts	Mech. Value	Wghts.	Mech-Value		Wghts.	Mech-Value	Wghts	Mech.-Value
p.lin in. in.	EU.	mean	%	EU.	%	EU.	%	EU.		%	EU.	%	EU.
		8·0	18·4	1·47	24·0	1·92	–	–					
6 (·1212")	9·0												
		10·0	8·0	0·80	9·6	0·10	–	–					
10 (·08")	11·0												
		13·3	16·0	2·13	22·0	2·93	–	–					
20 (·034")	15·5												
		17·6	14·2	2·50	18·4	3·24	–	–					
40 (·013")	19·5												
		19·8	6·4	1·27	8·4	1·66	2·3	0·46		8·7	1·65	17·4	3·31
60 (·01")	20·0												
		21·0	8·0	1·68	8·6	1·81	9·8	2·06		16·5	3·47	21·9	4·60
90 (·006")	22·0												
		23·5	7·8	1·83	7·4	1·74	23·2	5·45		17·1	4·02	15·7	3·68
200 (·003")	25·0												
		28·0	21·2	5·94	1·6	0·45	64·7	18·12		57·7	16·16	45·0	12·60
			100·0	17·62	100·0	13·85	100·0	26·09		100·0	25·30	100·0	24·19
							less Intake	13·85					
							Work done by Tubemill	12·24					
Duty p. 24 h.	Tons.		p.Stamp. 20 03	Total 200	160		200			p.Stamp. 5·88	Total 200	p.Stamp. 6·0	Total 1320
Energy cons.	HP.		4·5	45	95		140			4·5	153	3·7	814
Relat. Mech Efficiency p. HP.			200 T. × 17·62 EU. / 45 HP. = 78·3		160 T. × 12·24 EU. / 95 HP. = 20·6		200 T. × 26·09 EU. / 140 HP. = 37·3			200 T. × 25·3 EU / 153 HP. = 33·1		1320 T. × 24·19 EU. / 814 HP. = 39·2	

From these figures it will be noted :—

1. That the relative efficiency of stamps falls off rapidly from 78·3 for 9 mesh screen (0·27 in.) to 39·2 for 1,200 mesh (·0177 in.) and to 33·1 or 1,600 m. (·015 in.).

2. The relative mechanical efficiency of combined stamps and tube mills, under the conditions of the above example, is 37·3, against 33·1 for single stamp crushing through 1,600 mesh (·015 in.), and 39·2 through 1,200 mesh (·0177 in.). Single stamp crushing through 1,200 mesh is therefore somewhat more efficient than double stage crushing but at the expense of a coarser final pulp. On the other hand, stage crushing is more efficient if compared with the stamp crushing through the 1,600 mesh (·015 in.).

3. The relative mechanical efficiency of the tube mill is about one quarter that of the coarse-crushing stamp, but, compared with the stamp crushing through the fine screen (1,600 mesh) the ratio increases in favour of the tube mill to about two-thirds of the fine crushing stamp.

On January 19th, 1910, a report had already been submitted to the Mines Trials Committee, in which it was clearly laid down and shown graphically, that single stamp crushing appears to reach the limit of economy at about 1,200 to 1,400 mesh, corresponding to a final pulp of about 24·5 EU. (say 16% of the +60 grade). For the production of finer final pulp stage crushing proves to be more efficient, provided that the additional expenditure is paid for by the higher extraction to be expected. The economic advantages of regrinding by tube mills are only pronounced when the very coarsest battery screens are used, whilst the intermediate stages all show—if any—small advantages or, putting this statement in the words of one of my conclusions: "Not only the mechanical, but also the economic efficiency increases with the coarseness of the battery screen, and, even a 4 mesh screen (0·404 in.) appears not yet to mark the limit of the highest efficiency."

Economic Considerations.—To deal with economic efficiency and capital expenditure would lead us still further afield, and even the approximate figures would require lengthy comment. Accounts and elaborate statistics, although admirably done on all mines from a professional point of view, entirely fail to give to the engineer any grasp of the working costs of individual units or to inform him of the economical merits of intended innovations and improvements. Separate accounts are kept for all the principal units, but they are spoilt by a more or less arbitrary monthly distribution of general expenses, or working expenses which should more correctly be regarded as such. Items, such as mill transport, water service, assaying and sampling, pulp elevation, etc., are generally charged as a whole to the mill account and the tube mills live as parasites at the expense of the stamps. As long as they were few, it was admissible to consider them as simple auxiliary machines, but now, with wholesale tube-milling, they should share in fair proportion in these general expenses, incurred during the travelling of the ore through the plant from the crusher station to the cyanide works. The difficulty of distributing these costs, owing to the variety of methods of account, renders it practically impossible either to make accurate comparisons of, or to form correct opinions on the capital and running costs of stamps and tube mills: the consequence being that the opinions on these costs are still as divergent as at the time of the Mining Industry Commission enquiry (1908). In calculating capital cost allowance must be made for additional plant required in the tube mill circuit, such as launders, dewatering cones, pumps for relifting pulp, motors, buildings, etc., and in the case of weighting up existing stamps, it must be remembered that the economic advantages gained by overloading an existing plant have no necessary connections with the merits of the new crushing policy.

It is hardly necessary to mention that the sensational periodical publications of daily increasing stamp duties have never had any effect on the man in the street, who thanks to his common sense has always been aware that correspondingly more work has to be done by the secondary crushing appliances.

By taking all these points into consideration the dreams of hidden treasure vanish in the light of logical reasoning and correct figures. It will be found that the relative merits of the two systems are so close that the decision depends in each individual case upon the nature of the ore or upon other local conditions.

There can be no doubt that the decided advantages shown by the theoretical calculations for the very coarsest battery meshes will be realised in new plants, but when it comes to hanging up of stamps, such marvels as "to crush with 440 stamps more than we do now with the whole of our 820 stamps with a consequently much decreased cost"* have first to be proved. Financial considerations, and not the mere pride of revolutionising Rand milling practice have to justify not only the considerable capital expenditure for the new plant, but also redemption and interest charges on the book value of the 380 idle stamps.

Theory and Practical Experience.—Not even the yearly totals of reduction expenses are fair for comparison purposes, as they also are spoilt by different methods of apportioning "general charges," which, on some mines, for instance, include items such as surface maintenance, whereas on other mines, a portion of this maintenance cost is included in reduction costs. In his memorable and illuminating contribution to Dr. Caldecott's paper on the development of heavy gravitation stamps, Mr. E. J. Way gives a tabular comparison of the costs of 30 mines, where double stage crushing is in vogue, with the costs at the New Kleinfontein mine (which has never adopted tube mills) showing that the reduction expenses are lower than at the New Kleinfontein in only four mines, and that for reasons easily explained; whilst by taking the total reduction costs and general charges (which total alone is a fair basis for comparison), the cost per ton milled at the New Kleinfontein mine (4s. 6·894d., with a monthly output of 38,599 tons) is lower than on any other mine, including those crushing over 100,000 tons per month. (East Rand Proprietary Mines 157,243 tons per month, 5s. 6·731d. per ton. Crown Mines 121,100 tons per month, 5s. 11·625d. per ton.)

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* East Rand Proprietary Mines Meeting, [illegible]

The practical experience obtained from what may be called "year-long" trial runs is in perfect accord with the position as arrived at and laid down by my theoretical conclusions. No attempt has ever been made to contradict Mr. Way's figures, and the only answer forthcoming to his strong plea that more accurate data should be produced in support of the marvels advertised in newspaper articles and chairmen's speeches, was the remark that "the extent to which the capital cost of large modern Rand reduction plants has been reduced in recent years is as yet hardly realised," and the cheap rhetorical gibe: that the practical man who judges by actual facts may consider as a sufficient proof of the soundness of the views expressed by Mr. Way, that on his recommendation, the Benoni Consolidated Gold Mines decided on a crushing plant with heavier stamps than any as yet in operation here and with a higher ratio of tube mills to stamps than has yet been adopted in regular practice on the Rand. This proves, in any case, that Mr. Way is unprejudiced. By skipping the intermediate stages and at once adopting the coarsest possible crushing, he is only drawing correct conclusions from the trials and research work executed by the Mines Trials Committee. Theory proves that decided advantages are to be expected only by the adoption of the heaviest stamps and coarsest screens and if they are realised the onus of proof of the "absurdity and untrustworthiness" of my method of efficiency computation still lies on the other side.

The good efficiency attributed to stamps by my method of calculation is in practice borne out by the fact that all attempts to replace them by something better have hitherto failed. In any case, they will not be superseded by our *present tube mills*. With the coarsest meshes the stamps nearly encroach on the domain of the ore breakers, and even the most rabid tube-mill-progressists are afraid to consider the possibility of feeding the tube mills directly from the crushers, probably because in this case the tube mills would have to be self-supporting and could no longer live as parasites at the expense of the stamps.

By submitting the method to the hard test of applying it to ore-breakers the relative mechanical efficiency obtained is, as a rough average, about 50. (Stamps vary comparatively, according to mesh, from 30 to 85, and tube mills from 13 to 25.) This figure appears reasonable, seeing that the efficiency of ore-breakers is being probably over-estimated as up to 40% of the rock hauled is already broken to the size of the discharge. All methods of efficiency computation by "surfaces" fail completely in this case, giving such low efficiencies for ore breakers that it would not be worth while to have them running.

The conclusions arrived at with regard to the varying efficiency of stamps and tube-mills individually under different working conditions met with little or no contradiction, and some of them are now successfully acted upon in actual practice.

Final Remarks.—From the great number of tests executed on behalf of the Mines Trials Committee, especially from those executed in series, it has been ascertained that the proposed method, if judiciously applied, is remarkably reliable and trustworthy. The fact that its application must be limited to materials of equal technological state does not seriously diminish its practical value, as the industry of grinding has become so specialised that the possibility of establishing the relative merits of crushing appliances in dealing with any type of raw materials is already a valuable first step.

Completely independent from the question of the merits of the method of computation are the following two factors which affect the accuracy of the computation to a certain extent:—

1. The difficulty of obtaining representative average samples, particularly in the case of battery feeds, as these samples are subject to considerable variations in their composition with regard to sizing. By assuming in rough calculations their relative mechanical value in the average to be represented by the 0 grade, *i.e.*, by the size determined by the 1 in. screen, we are not erring as much as Mr. Johnson would make us believe when he states that the stamps receive credit for the reduction of 10% of the ore which was already finer than 60

mesh before it reached the battery. He overlooks the fact that the 67·7% of the 6 mesh contain a large amount of rock remaining on the 1 in. screen, the reduction of which to the 0 grade compensates for the work already done.

2. The more or less arbitrary assumption of an average size for the – 200 grade. It is quite apparent that the average size of a final pulp leaving a tube-mill with 64·7% – 200 mesh isbound to be much finer than that of a battery pulp with only 21·2% of this grade. In taking however, as I have done, all pulp of the mechanical value of 28 E.U. into account, I have been guided by the consideration that for Rand ore a further reduction of the – 200 sands simply means mere waste of energy, and that from the point of view of practical efficiency even this figure is too high. The large amount of work overdone in tube-mills is chiefly responsible for their inefficiency, and in my opinion conical tube-mills mark an improvement in this direction.

In conclusion, I wish to record my hearty thanks to the many contributors for the attention they gave to the paper, especially to Mr. MacDermott, who, in introducing it with kind and indulgent remarks, thought it worthy of the hospitality of the Institution of Mining and Metallurgy. The recognition and the appreciation of the importance of my introducing the energy factor into grading analyses, kindly expressed by Mr. H. L. Sulman and others, gives us hope that they will not lose sight of it and that they will find some further time to spare for its closer investigation. Those who admit the practical value of systematically executed research work, will join me in my appreciation of the liberality of the Mines Trials Committee in permitting the publication of data obtained in the course of investigation work executed on its behalf and of its broadmindedness in allowing the free expression of quite personal views and opinions, which, if scientific problems were to be settled in conclave, would, in the present stage, perhaps be those of a minority.

NOTE.—Please add in the formulae given in Table B the figures and letters printed below in heavy type, an omission which was unfortunately overlooked.

$$\log F = \frac{N}{10} - 0{\cdot}477 \text{ and } N = \frac{3 \log F - \log 27}{\log 2}$$

ADDENDUM.

My reply was already written when I was favoured by Mr. E. H. Johnson with an advance copy of his second contribution,* giving the experimental results of separating the finest portion of the battery feed from the coarse rock and taking it direct to the tube mill.

Compared with the example of double-stage crushing given in his previous contribution, the mechanical value of the final pulp, under the new working conditions has increased from 26·09 EU to 27·27 EU and the relative mechanical efficiency now works out at:

$$\frac{261 \text{ tons} \times 27{\cdot}27 \text{ EU}}{150 \text{ h.p.}} = 47{\cdot}4,$$

as against 37·3 of the previous example.

The data available are not sufficient to determine the proportion in which the single units contribute to this improvement. In my opinion the credit is entirely due to the increased efficiency of the tube mill in consequence of the increased coarseness of its feed. The same results would have been obtained—and by experimental tests they have been proved obtainable—by using a very coarse battery mesh. The great capacity and efficiency of tube mills with coarse feed has been proved more than a year ago, and the results of the first practical application of this fact, by Mr. Johnson at the East Rand and by Mr. Way, at the Benoni Consolidated, will be awaited with great interest.

Mr. Johnson's remark, "that we are only just beginning to realize the possibilities of the tube mill" is gratifying to me, as it proves that I have not been hammering away quite in vain for two years at a point admittedly capable of improvement.

An account of the laws of ore crushing and the experiments executed in that connection is contained in a work by F. Kick:—"Das Gesetz der proportionalen Widerstande und seine Anwendung," Leipzig, 1885.

The President: I have pleasure in asking you to accord a very hearty vote of thanks to Mr. Stadler for his interesting reply. I am sure it is quite in the best controversial fashion of this Society and I fancy the subject will require to be reopened again.

NOTES ON PRECIPITATION.

(*Read at July Meeting, 1910.*)

By F. D. PHILLIPS (Member).

REPLY TO DISCUSSION.

Mr F. D. Phillips (*Member*): The interest taken in this paper is gratifying, and my thanks are due to those members who have contributed their—not adverse—but very instructive criticisms. It convinces one that precipitation as conducted on these fields is still open to improvement, more especially if particular attention be paid to the minimum amount of zinc requisite for a first-class result.

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* See this Journal, October, 1910.

Mr. Brazier's curves are interesting, but would have been more so had he given another, showing what happened a week later, making it, say, the 15th or 16th day after the clean up. I quite agree with him that it is folly to keep the zinc in the store if it can be of any use in the boxes, but I am still open to conviction that eight or more compartments should be used when exactly the same results can be obtained with six. Admitting that it is only a question of labour, surely that alone is a point worth considering in these days of record-breaking in working costs.

With reference to his paragraph on temperatures, Mr. Brazier's experience appears to be in direct contradiction to that witnessed here, and I append some figures extending over a period of three months showing comparison between the results previous to installation of heat and the corresponding winter months in the following year after installation. It will be seen that on the average, the temperature was increased by 20° F. The remaining figures speak for themselves.

Before Utilizing Waste Heat.

	KCN.	NaOH.	Assay Value entering Boxes dwt.	Assay Value leaving Boxes dwt.	Tons Solution per cub. ft. of Zinc per 24 hours	Temperature at Head of Boxes F.
June '08	0·010	0·0140	0·331	0·033	1·58	56·4
July -	·010	·0150	·366	·041	1·42	53·8
August-	·011	·0140	·357	·033	1·59	57·2
Averages	·010	·0140	·351	·036	1·53	55·8

After Installation.

June '09	0·008	0·0140	0·412	0·010	2·08	73·8
July -	·006	·0157	·381	·015	2·17	76·3
August-	·006	·0156	·362	·013	2·16	77·4
Averages	·007	·0151	·385	·013	2·14	75·8

One very important point not to be overlooked is the increased amount of solution it was possible to precipitate. To sum it up shortly, there is less consumption of cyanide, better results and greater flow. Mr. White's statement on temperatures also verifies the above inasmuch as he says he found ·004 % "free cyanide" sufficient with hotter solution up to 28° C. Heated solution is certainly responsible for the comparative absence of white precipitate in the boxes here, and it is somewhat peculiar why the reverse should be the case at the Crown Mines.

Mr. Fraser Alexander advocates the use of extra compartments of zinc, "if only for filtering purposes." Can not a substitute just as efficient and much cheaper be found—coir for instance, which if placed in the last compartment of an extractor, could be taken out at each clean-up, washed, and replaced?

In reply to the point raised by Mr. White as to the accuracy of assays, I may say that for all the solutions quoted in my paper, our assayer worked with 40 assay tons in duplicate. When there was a discrepancy between the two results, a "mean" was taken which gave a value on 80 assay tons. I hope Mr. White will admit that this is working to a very fine point, and accurate enough for any experiments in connection with solutions.

Attention is drawn by Mr. Hayward Johnson to the fact that on some plants the flow of solution is not regular. This often happens here when the rate varies from 150 to 200 tons per box per 24 hours, but precipitation does not appear to be affected thereby. It is unfortunate that Mr. Johnson was unable to get an analysis of the coating on the zinc, as it is difficult to make out from his remarks whether it was white precipitate or a deposit caused by the stope-sweeping.

Below is an analysis of a substance which had accumulated in an empty compartment at the bottom of a slime box. It took the form of a thick white slime, and was pumped through a filter press. When calcined it was very light in weight, and of a powdery nature.

	Per cent.
Silica	6·94
Iron and alumina ...	4·30
Lime	6·08
Zinc oxide	62·23
Carbon dioxide ...	8·19
Loss on ignition ...	11·57
	99·31

Calculated as	Carbonate of lime ...	10·82
	Carbonate of zinc ...	9·70
	Hydrate of zinc ...	55·72

The remainder of the zinc had most probably become dehydrated in drying on the hot-plate. A trace of ferro-cyanide was also present.

In conclusion, the writer can hardly believe that the information desirable on this subject has been exhausted, and trusts that it will again be brought forward at some future date.

The President: We thank Mr. Phillips very much for his reply on the great subject of precipitation, which you see is by no means exhausted. I should like to ask about the substance analysed. What preliminary treatment was given to it before analysis?

Mr. F. D. Phillips (*Member*): It received no preliminary treatment before analysis.

NOTES ON BATTERY PRACTICE.

(*Read at September Meeting, 1910.*)

By A. R. Stacpoole (Member).

DISCUSSION.

Mr. D. J. Pepler (*Member*): The author in his paper made sweeping accusations against mill managers and amalgamators on the Rand for not taking an interest in the extraction of gold in the mills, for their careless way of recovering black sand and the handling of same.

After having read the paper over carefully, I could come to no other conclusion than that the author must be an absolute stranger on the Rand, for he seems not to have the remotest idea of the methods in vogue in mills here. Some of his recommendations are obsolete, and others so unnecessary and cumbersome, that in my opinion they are not worth considering. I presume the author must have seen these practices somewhere in the outside districts, and if he will visit some of our large mills here and see the means applied to recover all the gold possible from the ore by amalgamation, and how black sand is really recovered and handled, he will be sure to form quite a different opinion. Personally, I shall be quite pleased to show the author over the Ferreira Deep battery, and I am sure that other mill managers will be willing to do the same.

The author asserts that mill managers and amalgamators are rapidly having their power and responsibility usurped by the reduction officer and cyanide manager, but have themselves to blame for it; the reason he gives is that the mill men take no interest in the extraction of gold, but simply rush as much tonnage through the mill as possible. This statement I consider to be misleading and unkind to mill men on the Rand. I wonder what people in other countries will think of us here as mill men when reading such a statement in the *Journal*, notwithstanding that our mills have always been able to produce the greatest proportion of the total gold recovered.

With respect to the usurping of the mill manager's and amalgamator's responsibility by the cyanide manager as stated, supposing that the cyanide manager usurps the responsibility of the mill manager, will it be conducive to better results in the mill if he has no practical mill experience? Personally, I have failed up to the present to detect any improvement in the few cases where such appointments have been made, either in gold extraction, crushing or costs; therefore I fail to see where the cyanide manager has the advantage over a mill manager. I can quite understand a good and practical cyanide manager being appointed to the charge of several cyanide plants, a business with which he is thoroughly conversant, and a good and practical mill man being appointed to the charge of a like number of mills, a business with which he also is thoroughly conversant. Such appointments will naturally be conducive to good results.

It may be said the crushing can be watched, the assays, the percentages of gold recovered; this sounds very well, but what about that very important factor, the upkeep and maintenance of the mill, a factor which is so sadly overlooked in many cases and costing thousands of pounds which could have been saved. I do not claim for one moment that there is not room for improvement in our Rand mills, for there is always room for improvement in all classes of business, neither will I deny that there may be a certain amount of slackness in some instances, but you find exceptional cases in all classes of business, which only require to be put in order. In my opinion it is to the best advantage for many reasons that these two departments should be kept separate, for a one-man government has always been a hopeless failure, whereas party government is the great success of the day.

One great drawback which mill men are suffering under is, that the mills have no one at head quarters to represent that department in its true form. Personally, I take a pride in mill work, and if I can keep all the gold in the mill and let the cyanide works have none, I will be more than pleased. The maintenance of a mill should be constantly watched, for once a structure is shaken there will be no end to trouble and expense.

With regard to the recovery of black sand, it is not necessary when dressing plates to shut off the water immediately the stamps are hung up, for that will mean that 75 of tailing will be collected with the black sand. Neither do I see any advantage in hosing the sand down the plate into a trough. The water should be allowed to flow over the plate after the stamps are up for about 8 seconds, and then shut off; by that time most of the tailing will have disappeared while the black sand will remain on the plate, which should then be brushed up with a soft hair spoke brush and the black sand placed in a locked dish. This should be done again after the plate is dressed, there need be no fear that the black sand will be dislodged by the steady flow of clean water for a few seconds, since it remained on the plate when the stamps were in motion and pulp running over it. At the Ferreira Deep the black sand recovered by the above method gave the following results: of total amalgam recovered at tube mill plate house, black sand gave 59%, scrapes and other sources 41%; in the battery black sand 42%, scrapes and other sources 58%, which should be considered satisfactory. The plates are brushed every four hours, and with regard to the author's truck for carrying black sand and a trolley for carrying amalgam and brushes, I do not think it necessary to comment on them, as Mr. Alexander has already remarked that the author might have supplied a horse to pull the truck. An amalgam safe which is locked, and a stamped-steel dish 18 in. wide and 8 in. deep for brushes and scrapers which the amalgamator carries with him from plate to plate serve the purpose quite satisfactorily.

When changing screens, the author recommends the use of an apron, for what purpose I am unable to ascertain, as the methods in vogue up to the present have been found satisfactory to every one concerned. If the author were fully conversant with the subject upon which he has written, he would find in actual practice a different procedure altogether, and I would rather leave it for others, who have practical experience, to enlighten him as to how the changing of screens can be effected.

The idea of putting black sand or barrel residue through the mortars is in my opinion wrong, for on high grade mines the barrel residue is still of high value and is likely to contaminate tailing samples, especially when such residue runs up to 4 oz. to 6 oz. per ton. Sometimes the barrel residue should be treated by itself at the cyanide works and a return made to the mill manager of gold so recovered.

Screen sampling seems to be a vexed question; mill yield plus pulp I think leaves mill men absolutely in the dark. In my opinion the mill manager should work out his results on the actual screen value, and if the extraction called for is not what is produced he should go into the matter to find out where the discrepancy arises and have it remedied.

I work out my gold recovery on the assay value every day, and as soon as I find a discrepancy I go into the matter right away and find out where the discrepancy is and have it put right. I am glad to say that actual recovery and assays agree very closely.

I agree with Mr. Pyles that launders need not be lined with amalgam if plates are properly looked after. Prof. Stanley remarked that mill men are not taking part in the discussions. In my case I must admit the fact, for I very seldom attend the Society meetings. The reason is that I cannot hear when any interesting remarks are being made, and therefore cannot take part in the discussions. Having been in the mill for 23 years, my hearing is badly affected, and I think the same applies to a good many other mill men. At the same time it must be borne in mind that the man who shouts most is not always the man most to be relied on.

With regard to the disposal of amalgamating plates in the batteries, I rather prefer to reserve my opinion for a future occasion. Two new reduction plants have been started up lately where the new scheme has been adopted. Another one will be started soon in which a keen interest is being taken by everyone connected with the gold industry, and it is to be hoped that the returns will show a great success.

In conclusion, I may say that the author's paper has not shown anything new in mill work, nor has it enlightened any practical mill man whatsoever, and as the Prof. Stanley remarked "that mill men are not taking part in the discussion," this shows that the paper is of no importance, and should have been criticised by the committee before placing it before the members.

Mr. G. A. Robertson (*Member*): The main questions at issue in the author's interesting contribution are to show some weak points in the construction and methods of handling mill clean up plants. The illustrations and some of the methods given are good, and with the exception of the overflow from the "collecting trough," the "storage tank" and the "trolleys," there is no really new departure from representative Rand battery practice.

The author first tells us that the power of the mill manager and amalgamator is being usurped by the cyanide manager and reduction officers, and "have only themselves to blame for it." When first I read this I thought the author would be able to bring forward some tangible evidence to show that millmen had been guilty of gross negligence or incapacity to conduct their departmental affairs, but instead of so doing the author proceeds to make a sweeping statement to the effect that millmen paid "little or no attention to the extraction of gold, and looked upon it as his first and almost only duty to crush." It is the millman's first duty to crush I should say, but the author in laying down the duties of amalgamators would appear to put the cart before the horse, and want the free gold, amalgam and mercury looked after as the "first and principal duty." Instead of the author then showing us some method whereby a saving might be made in the free gold, mercury, and amalgam when they find their way into the cyanide works, he proceeds to tell us how to collect sand from plates which even though a part of it were lost by the "clumsy method," which I think died out on the Rand years ago, it would only be a mere bagatelle to what is lost incidental to the production of amalgam in the ordinary course. According to H. H. Johnson and F. H. Roche "15% of the entire mercury fed is lost . . . and it is well known that the amount of gold carried away and totally lost by this means is enormous."* Another authority in commenting on the quicksilver situation states that the "ordinary stamp mill loses 20% or more."† It would therefore appear to be duties extraordinary "to see that no free gold, amalgam or mercury goes over the end of the amalgam plate." Whether the author meant that the words "to see" is visible and external means in the collection of sand, etc., and invisible in the cases just cited is perhaps not altogether void of logic. At the same time if the cyanide manager usurps the power and responsibility of the millman he wants to be a little more careful in defining the duties of amalgamators.

I do not see any advantage to be gained by the design of collecting trough Fig. I. (p. 113) and overflow into a bucket. Collecting troughs generally in use are made to fit the launder with square edges and as a rule catch all the sand desired, also allowing for decanting off a portion of the water before emptying the contents into a bucket. Launders, like everything else, are not always designed in accordance with the needs of practical experience, so no doubt in that case the author's overflow would allow a factor of safety to work upon with shallow launders. I might here point out that the plate tail launder sloping towards the centre as shown by the author, though almost generally adopted, is not required and should be made perfectly horizontal at least 6 in. deep by 6 in. wide. The author's brackets to hold the trough on are certainly original. Riffles are generally to be found in plate launders which being cleaned out alternate days are responsible for over 80% of the silver recovered which escaped from the amalgamated plate. These riffles meet the requirements of the author's brackets. I am of opinion that instead of the drop pipe into a mercury trap the fall could be put to more profitable use by allowing the pulp to pass down a flight of step riffles not unlike those used by the Egyptians and illustrated in Johnson's "Getting Gold." I agree with the author about the cleaning out of the troughs in a cylindrical tank set on a carriage because sand cannot be properly tipped out of the trough into a bucket. The author must have seen some careless amalgamators at work carrying brushes around by hand "full of amalgam." I agree with the author that "scratching and tearing up of the plate and amalgam" should not be allowed. No doubt the use of the rubber apron as suggested by the author would obviate this trouble, caused by "throwing tools, screen pins, pulling coarse gravel out of mortar box, &c., on to the plate." The storage vat, as illustrated in Fig. 3, p. 113, is the proper method for storing concentrate for amalgam barrel treatment. Several new milling plants have the amalgam barrel and accessories placed in the following order:—Amalgam barrel, chute, Batea, amalgamated plate, settling boxes, the overflow from the settling boxes going into the tailing launder. If the tailing launder is at mill floor level, such generally being the case where the amalgam room is at the end of the mill, it will necessitate the axle of the amalgam barrel being 6 ft. above floor level. A storage vat of 5 ton capacity is not uncommon; the difficulty however arises in getting sanction for the rearrangement of the storage vat in the method shown by the author, which in many cases would, I am afraid, necessitate an extension of the building. Simplifying matters further would be to have an inclined stage to haul up a concentrate collecting trolley and tip direct into the storage vat. Such an arrangement would very soon pay for itself because it is not a difficult matter to collect concentrate to the assay value of 100 dwt. from traps, &c., in the tube mill circuit from a milling ore of 6 dwt. With modification in the handling then the tonnage dealt with could be trebled without materially affecting the assay value of the concentrate. One objection to amalgam barrel treatment of concentrate is that it slimes all the silica where with tube mill treatment this portion passes off readily, allowing the pyrite to be more finely comminuted for subsequent treatment by cyanide.

A simplification in the methods adopted in treating "magnets" or "battery chips" would in the writer's opinion be effected by treatment in a miniature tube mill. "Weathering" of these chips is generally practised before treatment, but experiments went to show that fine particles of amalgam adhering to iron or steel can be separated by a tap from a hammer: this led the writer to the conclusion that an abrasive action to the "chips" would wear off the amalgam, which would then be released by passing through a fine meshing on the discharge of a tube mill.

The sketch represents a model which was partially made some years ago and consisted of a short piece of an 18 in. pipe which when found on the scrap heap had conveniently flanges on each end. By charging up this tube

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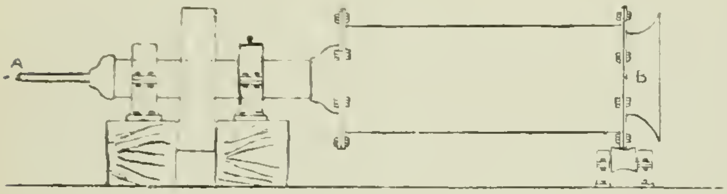
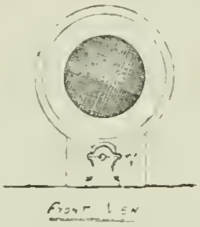
* *Journal S.A.A.E.*, Suggestions for Deep Level Mining.

† See *Mining and Scientific Press*, May 28, 1910.





with magnets and steel balls a gentle stream of water at A would undoubtedly free the amalgam which when ground fine enough would pass through the fine meshing B. The proper diameter of meshing and aperture would easily be determined by experiment. This tube mill possesses the advantage that when not used for the purpose stated it can be converted into an amalgam barrel by the bolting of a plate on the discharge.

Mr. W. C. Brown dealt with the arrestment of iron from the tube mill circuit,* and Mr. G. O. Smart emphasised the fact that a magnetic separator would be of considerable advantage. With the tendency there is towards coarse crushing which releases increased quantities of coarse iron from the mortar box, thereby increasing the quantity of this objectionable product in the tube mill circuit, the use of a magnetic separator will soon become essential, likewise a quick method of treating the increased quantity of these by-products.

I agree with the author that mill men should join more generally in the discussions on tube mills and milling in general, a subject in which we have all got much to learn. When our chiefs, however, in their periodical visits round the mines, give vent to free expression of opinion, the result is that millmen have perhaps foolishly converted it into the commandment "Thou shalt not adversely criticise my ideas," however the recent speech of Mr. Lionel Phillips, who pointed out before a kindred society that Consulting Engineers and their chiefs are only too pleased to have their faults pointed out, ought to stimulate free expression of speech, without which we are absolutely in the dark. We have heard of the good points of various processes and devices ; we now want to know of their bad points (if any). These are more likely to come from those whose daily and observant attention allows them to make a more careful scrutiny of varying working conditions in the ordinary course of routine work.

I am afraid many of us do not agree with the author that free gold should not get into the cyanide works. Mr. David Gilmour pointed out* that sufficient amalgamated plate area was required for the tube outflow to prevent enrichment of the circuit, the overflow from the hydraulic classifiers also passing over plates. On the other hand, the tendency is to have all the plates in the tube mill circuit, consequently if the pulp from the stamp mortar is not amalgamated, nor the overflow from the hydraulic classifiers, fine free gold is getting into the sand charges, and the first stage towards the elimination of gold amalgamation on the Rand has been reached. If the final gold extraction is as good by this method as the former one, does it not seem probable that a portion of the gold in the tube outflow is sufficiently fine to be also amenable to cyanide treatment. Direct reclassification of the tube outflows without amalgamation would also add to the quantity of fine free gold in these overflows, the underflows passing through another tube mill. By this method to reclassify the outflows from three tube mills getting in all, say, 300 tons to pass through the final tube mill would reduce the number of plates required from say 24 to 8. Maximum mechanical efficiency will not be got from every fourth tube mill but that is not of much consequence in a large plant, the utility of the idea and saving in capital expenditure having also to be taken into account.

Metallurgists on the Rand have long pinned their faith on the use of mercury, although this silvery liquid has been dispensed with in many cases in the United States of America and elsewhere. This, of course, does not justify its abolition here, and no doubt the failure of the Meyer and Charlton mine† to get a satisfactory gold extraction by the elimination of amalgamation will no doubt strengthen the use of mercury for a further period.

Though I have gone a little outside some of the points dealt with in the author's paper, all of it, however, can be considered battery practice.

It would take much space to go into all the details of the author's paper, and I think I have let him off lightly on one or two points. He is methodical in details, adopts a system, and by his words he will see that it is carried out. A few more papers before the Society on battery practice, a wide subject, would be to the benefit of all, as much yet remains to be done in that direction

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* Trans. S.A. *Institute of Electrical Engineers*, Vol. I., May, [illegible] p. 122-125.

* See this *Journal*, Sept., 1910, p. 134.

† See Chairman's speech, Report of the Company—*South African Mining Journal*, March 12, 1910.

NOTES ON THE OCCURRENCE AND CONCENTRATION OF TIN ORES.

(Read at October Meeting, 1910.)

By C. Fred. Thomas (Member).

DISCUSSION.

Mr. E. M. Weston (*Member*): I am afraid I cannot altogether congratulate Mr. Thomas on his paper. He has attempted to cover far too wide a field, and so has had to commit himself to many vague and wide generalizations to which numerous important exceptions have to be taken. He does not also appear to be acquainted with many of the latest developments in the practice of tin ore concentration.

In his remarks under mispickel we first meet with a somewhat vague statement. "The extraction of mispickel is low by wet concentration and selection previous to milling is advisable." Does he mean that it is very much more difficult to concentrate a mineral with a hardness of 5·5—6, a specific gravity of 5·76—6·3, which has not a very distinct cleavage, than for instance chalcocite with a hardness of only 2·5—3 and a specific gravity 5·5—5·8 from which a 75% extraction from disseminated ores is being made in modern milling plants after stamping with Nissen stamps! Arsenopyrite is being concentrated in many gold milling plants to-day, without great difficulty, as experience in Deloro, Canada, and Lucknow, N.S.W., has proved. Several of the Rhodesian mines would obtain I think better extraction if they took the trouble to concentrate both their sand and slime before cyaniding. I am amazed to learn that (apparently *semper ubique ab omnibus*) wet concentration for the extraction of the sulphides of copper is unsatisfactory. Of course this depends on dozens of conditions, and I will not waste the time of this Society by quoting many instances. I need only mention that in the Tasmania mine, Beaconsfield, Tasmania, a gold bearing quartz ore containing a large percentage of sulphides of copper and iron is treated in stamp mills. The sand and slime are successfully concentrated, yielding a high grade concentrate for roasting, while the extraction of the copper renders the sand amenable to cyaniding. The author really should try and realize that Cornwall is not the universe—or even the hub of it.

The author again shows that he is not aware that in some parts of the world stannite is by no means rare. He is unaware that the Conah Mines, Zeahan, Tasmania, depends on this ore exclusively, and that a special smelting plant is successfully at work, designed for the extraction of copper, silver and tin with an output of some £1,000 per month. In the Tewlong district, New South Wales, an extensive bedded deposit is now being mined. It is not always found in slate near granite.

Under the head of production of tin the author appears ignorant of the fact that very extensive deposits of alluvial tin have lately been discovered in the Niger Protectorate which bid fair to compensate for the stationary or declining output of the Straits Settlements and Australia.

Under the head of occurrence, what does the author mean by "the fluidity of the tin product in smelting?" I am surprised at the hyper-stylists of the Editorial Committee (with the fear of T. A. Rickard before their eyes) passing such an expression, especially as some of these purists objected to my perfectly legitimate use of the word "slab" as a verb and placed it in inverted commas as if it were a coloured person in white society. Besides, did not they lay impious pens across some of Mr. H. Smith's pet mining terms? I might point out to the author that the tin-bearing pipes of the Transvaal are not unique, as everyone knows similar pipes occur in Inverill, New South Wales. In dealing with concentration the author again commits himself to *ex-cathedra* statements that require modifying. He states that sand from a modern tin-dressing plant should not require further treatment, apparently not considering that this depends entirely on the fineness of the tin ore and the nature of the gangue. At Mount Bischoff, where much fine ore occurs in a hard ironstone, it is necessary to regrind a portion of the sand to get a proper extraction. As most of the tin dressing plants of Cornwall are or were notorious for their slime losses and generally inefficient work, I am not surprised that the author from his experience ventures the opinion that average extractions do not exceed 80%. If the author goes to Australia he will find that better work is required and better means used for obtaining it than he appears to be acquainted with. The Mount Bischoff mine treats 19,000 tons of ore a month containing less than 1% SnO_2, with 90% extraction. It is only fair to admit that the Cornish tin dresser has had on the whole a difficult ore to deal with, and until about 25 years ago, considering the circumstances he worked under, he did really good work. This does not alter the fact that, until the Germans came over to Clitters and gave Cornishmen ocular demonstration in the art of ore dressing under modern conditions, ore dressing practice in Cornwall was about a generation behind the times. This would not matter except to the Cornish tin industry if it were not for the fact that the Cornish tin expert is prone to strut around the world posing as the one and only authority on the ore and its concentration and inflicting out-of-date Cornish dressing devices on young and unsophisticated tin industries in other countries. I have no prejudice against Cornish mine methods. The Cornish miner in his day and generation led the world in the art of practical mining and in the evolution of one type of pumping machinery; but he got wedded to these methods and the refusal to adopt more up-to-date methods because they did not come from Cornwall nearly wrecked the Cornish tin industry. When I state that out-of-date methods and machinery are used on many of the new mining fields, I speak from what I have myself seen here and in Australia and elsewhere, and what is too well known to be denied. Cornish workers have a rooted disbelief in the capacity of any vanner such as the "Lurhig" to make heads concentrates and middlings from slime. Take again the famous rag frame. Here the Cornishman excelled himself in inventing an automatic device to save labour, which he so freely wastes despite its cheapness on his sand or slime buddle, and he is so proud of it that he has overlooked the fact that he has deliberately sacrificed the very quality that is essential in such a device, namely, area of settling surface. The procedure about the famous Red river in Cornwall is somewhat as follows: (I have heard that it derives its name from the fact that the ruby tin escaping from the concentrating plants is in such quantity as to colour its sand red. This may, of course, be a wicked libel). The mine dressing plants pass the slime over a row or two of rag frames, and extract about 10% of the recoverable values and pass the slime to worker No. I. on the Red River. He does the same and passes slime on to No. II., and so on to No. V. or VI., so that about 60% of the recoverable value is finally saved. Does it never strike any of these workers that if the tin is recoverable it is just as easily recoverable at the mines as by six different workers. Australians long ago discarded the rag frame with its 30 sq ft. of settling surface for what the author calls "strips" and does not explain what they are. I suppose some American or Australian came along and gave the designers of the South Crofty plant some hints, as I see they are installed there. These "strips" in Australia consist of frames made of T and G, 6 × ½ in., pine constructed thus:

Each tray consists merely of 6 by ½ in. T. and G. pine boards held together by two 3 by 2 in. pieces to which they are nailed. These 3 by 2 pieces form the sides of a shallow launder of which the boards form the bottom. The canvas is somewhat longer than the tray in order to overlap the next tray and is secured to the tray by light strips of pine at the sides lightly nailed on. This size is about 30 in. wide, and just the width, or little less than the width, of a strip of No. 8 canvas duck, and about 4 ft. long. These are simply stacked in three or four layers one above the other, and the number of separate rows depending on the quantity of slime to be treated, and are given the proper inclination. A spare row is provided. The concentrate is collected by turning the water off one row, and a boy then dips the individual frames in a tub after running fresh water over them, and if necessary gives them a brush over. One boy can look after quite a large plant, and labour expenses are negligible, and capital cost is almost nil. In Bendigo I have seen gold saved on these strips after passing the amalgamating tables, which was so fine that it took over ten minutes to settle in a bottle of water after it was shaken up. I believe rag frames have been installed on some South African mines where the cost of labour to look after strips of this kind would have been a negligible quantity.

In this connection Mr. Edward Walker[*] asked some years ago "why is it that a succession of from a dozen to 20 tin stream works between Dolcoath and the sea can handle the same tailings and all make a profit? I have heard many theories put forward, varying from the natural concentration in river beds to the growth of tin crystals out of nothing."

"To me the explanation seems quite simple. Each dressing works at the mines or at the tin-stream works passes the original slimes only once over the rag frame. The succeeding series of rag frames treat the concentrates from the preceding ones. With such a slippery article as tin oxide it is obvious that the settlement of the particles on the surface is a matter of chance, and the passage of the slime down a mere 6-ft. length is not sufficient to catch 5% of the contents. From this point of view it is easy to see why constant retreatment of the slime pays the various tin streamers, and why the last streamers do very nearly as well as the first."

The author's opinions on the origin of cassiterite in the Zaaiplaats and Groenfontein pipes are open to grave question. I can only refer him to the opinions and arguments put forward by Mr. R. Kecknagel, who has made this subject a special study. As the core of the pipes consists of altered granite, there are great difficulties in imagining them to be geysers in any shape or form, and all the evidence points to them never having been open water channels of any sort. I have never heard responsible geologists declare that simply because the source of the ores is from beneath, it is reasonable to look for large deposits in depth. In most mining *depth* is unfortunately an important factor tending to poverty, and though I have not the slightest reason to suppose that the tin deposits of the Transvaal are only superficial or that prospects discourage mining in depth, yet I object to generalizations of this sort as tending to mislead.

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[*] *Engineering and Mining Journal*, N.Y., May 25, 1907.

Tin Ores.—The author has neglected to mention the unfortunate association of ilmenite or titanic iron with cassiterite in many parts of South Africa. Ilmenite is frequently mistaken for cassiterite and owing to its high specific gravity causes trouble in many places in the separation of the two, and calls for the use of magnetic separation methods. What is one to think of a generalization like this? "Some difficult ores for concentration have been met with, but with depth such difficulties may not be met with." One can only ask when! where! why! And to the next sentence the author might add "in Cornwall," as I know many instances to the contrary elsewhere. His remarks as to mining tin ore may apply to Cornwall but not to everywhere else.

Under *Concentration* the author shows his ignorance of modern practice by stating that "sizing for fine grained cassiterite is unsatisfactory, and that one is dependent on hydraulic concentration." In modern Australian plants great advantages have been gained by the use of Callow screens and similar devices below the battery in place of classifiers. I might mention the plant on the Mount Bischoff Extended T.M. Co. It is not at all necessary to avoid pulp elevation. One of the features of the Mount Bischoff plant is the successful use of small jet elevators, enabling middlings and slime to be subjected to retreatment without trouble.

THE EFFICIENCY OF LABOUR UNDERGROUND.

(*Read at August Meeting, 1910.*)

By Tom Johnson (Member of Council).

REPLY TO DISCUSSION.

Mr. Tom Johnson (*Member of Council*): Judging from the criticism outside the Society my condemnation of placing young men without practical experience in as shift bosses I expected to be well trounced during the discussion and my error pointed out. No champion has been forthcoming, and the remarks of my critics have strayed somewhat from the main point of my paper, viz., on the question of the method of payment. If the present system is not worth defending it should be altered.

One critic remarks that there are not enough technically trained experienced men to fill all billets, therefore I say let those who are only theoretically trained go down the mine and tackle the real work first, instead of trying for staff billets in the sampling and survey departments. One year's work at actual manual practice would make better men of them than three years in a staff billet. It is easier for a technically trained man to be a successful manager than a good shift boss. I maintain that a shift boss's job is the hardest in a mine when well done.

Many of the young college men think I am prejudiced against the technical man. Far from it: I am prejudiced against any man who only knows one side of his business; and in proof of that I may mention that I went to a great deal of trouble to become technically proficient by travelling miles after a hard day's work to attend night classes, and this I did for two years. No one expects a surveyor to handle an instrument underground proficiently without previous practice, so how can a man only technically trained expect to boss up any practical branch of work? Is it done in any other department of the mine? It seems strange that when one dares to mention that the managements of the mines do not take the trouble to see that the fair thing is done by the men, one is at once accused of unfriendliness towards the mines. Records, however correctly kept, are worthless unless they can be made good use of, and that can only be by a practical man capable of understanding them.

Mr. Price is quite right about the price paid per unit of work, the correct price is what a man of average ability can make a fair living wage at; a good man who breaks well above the average amount of ground should be paid for his extra ability, while the poor and indifferent worker should be made to realise that a standard wage will only be paid when the standard amount of work is done; the indifferent should be taught to to better work and the good man should not be "cut," thus making the latter lower his efficiency by just doing sufficient work to be profitable to himself.

The following figures from practice will more or less bear out my contention that the actual price paid to a man does not necessarily determine the true cost to a company

No.	Stoping Width	Fathoms per Machine Shift.	Rate paid per Fathom	Amount Earned per Shift		Cost per Fathom to the Company	
			s.	s.	d.	s.	d.
1	71	·331	45	—		104	5
2	80	·445	42 6	—		80	3
3	74	·673	45	—		79	4
4	49½	·289	45	15 25	4 4	135	2
5	57	·727	50	30	4	67	10
6	69½	·246	45	15	4	127	2
7	86½	·549	45	8 8	9½ 5½	67	6
8	55½	·614	45	25 24	4½ 5½	65	6

Looking over these figures it can be seen that Nos. 4 and 6 did not earn anything on their contract, but were bonused by the manager, but one would have thought Nos. 2 and 3 deserved a bonus if 4 and 6 did. Then note No. 5 with his high price: his wages are fair and his total cost low. These figures are worth studying by those miners who say that they cannot understand how they can come out in debt. In this list Nos. 1, 2, 3, 4, 6 did not earn anything, yet No. 5 comes out with 30s. per shift and a lower total cost to the company. Who is the best man to the company, the man who made the wages, or any of those men who were in debt? Note that No. 8 made wages and has a low total cost. These figures show that there is room for tuition by somebody, and who better than the man who by actual practice has proved his worth!

I would refer Mr. Wiseman to a paper read before this Society by Mr. Weston, in Jan., 1909, for the reasons why machine stoping costs more in explosives than hand-stoping: but there is something more than cost of explosives only to look to for general efficiency: it is the total cost of the particular work that should be looked at; thus in a comment in the *S.A. Mining Journal*, November 19th, on the Goldfield's Report, taking a comparative stoping width of 55·7 in., the cost of breaking rock at the Robinson Deep in August was, per ton:—

For small machines ...	5s. 2·3d.
For large machines ...	5s. 7·3d.
For hammer boys ...	5s. 10·8d.

The point I wish to emphasize is to ignore the increase of individual items, so long as the total is lowered.

I do not think a man can drill as many holes per day, drilling two holes per bench, as when drilling four holes per bench, as it takes time to move a machine from one bar to another. When lighter stoping machines come into general use then we shall get the two hole bench. If holes are drilled too short by hand on one shift let them be deepened to the desired depth on the next shift, instead of their being blasted; in spite of the old arguments of a man wishing to blast his own holes, drills losing gauge, etc., the results would justify the trouble; drills, whether machine or hand, should not lose their gauge if well shaped, sharpened and tempered, and in most ground for 8 ft. or 9 ft. holes two star bits should suffice. Leaving sockets depends a great deal on the man putting in the holes; if the burden is heavy and the explosive is required well down in the bottom, the hole should be "chambered" first to make room for the explosive, and the benches should be kept long.

I am an advocate of long holes, and I do not like Mr. Weston's idea of drilling the four holes "looking out," I would rather use a little more explosive and get the ground with four holes instead of five. The average depth of hole drilled in the stope drill competition will surprise Mr. Weston. It would be much better to lead water on to the face and use a fan to ventilate, or even have a leak in the air pipes than work with such a complicated gear as Petersen's respirator.

I fancy Dr. Aymard's device was anticipated on the second rock drill brought out, but if it succeeds in keeping down the dust, I wish it success.

During my management of the Rose Deep I always advocated the principle of only sending ore to the mill from which profit could be earned, not to keep the mill running at any cost, and in this I was supported by my chiefs.

I cannot agree that if there were no natives on the mines work would stop; we should do the work ourselves, as is done in the Australian and North American gold fields, and as cheaply as the natives do it.

Mr. Ross Browne reported that on two mines in the Western States using white labour under similar mining conditions to our own, the same amount of work was done with as few men as we employ for supervision. I am of the opinion that cheap native labour has been a factor in keeping up costs, and that there has not been sufficient care taken to get the best efficiency out of the native.

There is room for an explosive of lower power than either gelatine or gelignite. I have often felt the want of it for breaking "leaders" in the roof and floor and for light burdens If Mr. Beaton will trouble to read previous papers brought before this society by me he will find I have mentioned most of the things he proposes, viz, longer backs, straight drives, double tracks, etc. I do not agree with stoping breasts of 30 to 60 ft. width on the strike for 300 ft. I have been advocating a more extended use of the stope face and I have not heard or seen anything yet to cause me to alter my opinion. Mr. Beaton claims several advantages for his system of long-wall working. Let us examine them:

1. *Lower developing costs.*—Any system of lengthening backs will do that. Will not stoping costs be higher?

2. *Less shovelling and tramming costs, as all ground would be blasted down to the track and no manual tramming to a greater distance than 300 feet.*—As long ago as 1897 I split up 200 ft. stope faces in the Geldenhuis Deep into three portions with brake roads and side tracks and for many years have advocated locomotives on the drives.

3. *Better supervision work, being more concentrated.*—This is not so, for 200 ft. of face is open and only 30 ft. to 60 ft. of it in use.

4. *Simple ventilation and drainage.*—Breasting dead ends in 300 ft. is not going to simplify ventilation or drainage.

5. *New mines would reach the producing stage in less time than at present.*—I cannot see this, as longer backs will always mean longer time in opening up. 900 ft. of intermediate shaft sinking would be done quicker from three points of attack than from one.

6. *Greater safety, no great stretches of worked out places being left to endanger the whole mine and surface works.*—This is granted, other ways have been proposed.

7. *No pillars of reef need be left and no costs incurred in raising box holes.*—This can be done with our present system of opening out. See 9.

8. *Less timber and timbering.*—This is granted.

9. *Sand would be held safely in position and no mud rushes could take place in a wet stope.*—This is not so as there is always one open side to barricade, and the top must be taken out under wet sand?

10. *The output from any stope would not be limited by the number of shovelling boys it is possible to crowd in a stope.*—The output would be limited by the amount that could be broken on the 30 ft. to 60 ft. face.

Mr. Beaton should embody his ideas with sketches in a paper so that members of this society could criticise and then he would I am sure be forced to modify his views. I also suggest that the Johannesburg correspondent of the *Mining Magazine* (Oct.) would have done better to have attacked me in this hall in open discussion rather than give me a rap in the dark; he has evidently been "hard hit." In conclusion I must thank those who have taken part in the discussion and so extended its scope. And I predict that in the near future we shall have better and cheaper work done and a more satisfied and steady set of men when the work is put up to tender and the men thereby fix their own prices.

VENTILATION AND HEALTH CONDITIONS ON THE MINES OF THE WITWATERSRAND, WITH SPECIAL REFERENCE TO THE VENTILATION SYSTEM OF THE EAST RAND PROPRIETARY MINES.

(*Read at August Meeting, 1910.*)

By S. Penlerick (*Member.*)

REPLY TO DISCUSSION.

Mr. S. Penlerick (*Member*): Several interesting points in connection with ventilation and health conditions in our mines have been brought up in this discussion. I think the most important of these is the question of the *origin of dust* in the mine.

I gave the sources of dust as: (*a*) rock drilling; (*b*) blasting, and (*c*) handling of rock. The judicious use of water and sundry contrivances arrest at least the heavier dust generated by rock drilling and by handling of rock, but there still remains to be dealt with the finer dust which escapes, and also the dust generated as a result of blasting. As blasting operations are distributed throughout the mine and are violent, the amount of dust thus created is enormous, and the area over which it spreads makes it impossible to arrest it all with water, and moreover much of it —the finer portion—does remain in suspension, and must therefore be removed from the mine altogether, and as quickly as possible, and this is effected by an efficient ventilating current carrying it directly into the main upcast. Mr. Stuart Martin does not agree with me on this point: he does, however, admit the existence of the fine dust and at the same time acknowledges the necessity for ventilation, which necessarily entails a current of air in the mine. Therefore, the state of affairs to which he objects will actually prevail, whether he wishes or not. An efficient ventilating current does not create a dust storm, and should not have a greater velocity than will carry off the finer particles which do not readily settle nor can be arrested by any spraying device in the mine. By this means the fine dust will not be allowed to circulate throughout the mine, but will be under control and carried by the shortest route to the main upcast of the mine.

Mr. Stuart Martin rightly points out the necessity for limiting the amount of water used. This leads us to the consideration of a most important point which has to be dealt with, viz., the high percentage of moisture of mine air, and along with this we must deal with the question of reducing temperatures. It may be that we shall have to instal cooling and dewatering apparatus in the workings, to which processes Mr. Johnson has referred, but that is a matter for lengthy consideration. The author does not consider it necessary to place brattices or pipes in all dead ends but stated "where a drive face is being carried far beyond the last connection with an adjacent level, it is necessary to provide for an intake and an outlet." The case instanced was that of a connecting drive 3,000 ft. beyond a connection. In normal development, putting down winzes as soon as the drive face is advanced far enough, working single shift and machines in the face are sufficient. I consider single shift in development faces and stopes is not only practicable but, in every way, most advantageous from several points of view, of which I may mention, efficiency, cost and ventilation. I, therefore, have to differ from Mr. Johnson in this respect.

Referring again to dust, I may mention respirators. I do not agree with Mr. Hildick Smith that a respirator can be worn continuously with "no inconvenience in either breathing or talking," and as he asks the question I may tell him that I have personally experimented with all types, and for many years, and can, therefore, speak with authority.

With regard to vitiation of mine air by compressor air, my experience agrees with the statements of Mr. E. J. Laschinger and Mr. McArthur Johnston to the effect that no carbon monoxide can be detected in the air delivered by the compressors under normal conditions.

Some surprise, and even doubt, appears to have been occasioned by my statement that the temperatures have been reduced by from 2° to 3°. I can only say that my statement is founded on actual results, and the same remarks refer to the figures for CO_2 which our President considers were impossible unless representing samples from absurd places. The samples were taken in every case from actual working places while work was proceeding and special precautions were taken to have correct analyses.

With regard to Mr. Johnson's remarks, I may say that the speed of the "Sirocco" is 185 r.p.m. maximum, and of the "Barclay" 165 r.p.m. The former is driven by a 250 h.p. motor, the latter by a 200 h.p. motor (temporarily).

The water gauges are in the fan drifts about 25 ft. to 30 ft. from the fan. We have two forms of gauge pipe, in one form the pipe is bent at right angles, open at the end and facing in the direction in which the current is travelling; in the other form the pipe is straight across the drift, closed at the end and has about 24 small holes bored on the side facing the direction in which the current flows. I do not intend to follow Mr. Johnson's advice and take down the "Sirocco" chimney and rebuild it, giving it a larger area as the gradually increasing areas of the chimney in its present form are correct as far as it goes. Theoretically, however, it would be better to continue the chimney higher till the area at the top was such as to still further reduce the velocity of the air as it was delivered into the free atmosphere. Probably for our purposes here 144 sq. ft. area at top would be suitable, the 200 sq. ft. mentioned by Mr. Johnson would, we think, reduce the velocity too low.

We should all be obliged to Mr. Cullen for introducing the "health" aspect in his lengthy discourse on ambulance work. Unfortunately the average miner who would benefit most by a general knowledge of "first aid" has generally shown a decided disinclination to undergo any useful training in ambulance work. The necessity of it, however, is only too frequently instanced and I trust they will soon recognise this. At the same time every encouragement must be given them by the mine managers. Drs. Irvine and Macaulay have already laid down the requirements in this department.

With regard to safety helmets, I have only had direct experience of one make, and while it worked satisfactorily on several occasions, it failed on one occasion with, to say the least of it, most uncomfortable results.

The whole ambulance question is certainly one for collective action by mining authorities. As Mr. Cullen says, it entails neither heavy expenditure nor much trouble, while the return is invaluable.

THE SHRINKAGE METHOD OF STOPING AT THE FERREIRA GOLD MINE.

(*Read at November Meeting, 1910.*)

By G. Hildick Smith B.Sc., F.G.S. (Member).

DISCUSSION.

Mr. Tom Johnson (*Member of Council*): The author has certainly made a mistake in describing the operations set forth in his paper as new; still it is a mistake one could easily make seeing how little the method is used. It serves to point out to us older hands how we keep looking for something new instead of using the good old ways where applicable.

I have many times heard men talk of swinging on a chain in their stopes and of the trouble they had to rig their machines, etc. If the system described by the author had been in use in these places, a deal of the trouble would have been obviated. I believe I mentioned this matter of back stoping, working on the broken ore in my paper, "Notes on Mining," (this *Journal*, March, 1908.) Although the method is not new to the Rand, still the author deserves our best thanks and of the younger members especially, for the description and wealth of detail costs given.

In putting in the bearers for the sets of timbers forming the ore passes it is not necessary to be so particular as the author seems to imply. If the hitches are a little wrong, causing the bearers to be out of perpendicular, levelling up where necessary before placing the first set will overcome the trouble.

In examining the results of working I think the cost of 3s. 1d. per ton to be a little low, as this only leaves about 9s. per machine shift for air, steel, sharpening, machine upkeep, etc., which is not enough. If this figure is correct it serves to strengthen my opinion that the work done in breaking ground shows up badly, ·582 of a fathom per machine shift is low considering the facilities for breaking described by the author. He shows a bench 7 ft. × 5 ft. equals 35 sq. ft. area, but only 21 sq. ft. area per shift is broken, a little greater than 5 ft × 4 ft., which is bad with a clay foot and good hanging. I cannot see that there was anything else than poor work to account for this seeing that it only took between 10 lb. and 11 lb. of explosives per fathom

The author speaks of dry holes decreasing the efficiency, where dry holes should not have been necessary: the winze or rise should have been wide enough to rig up in and drill wet holes. If a bench got spoiled it should have been put right with a hand hole or two, which is the cheapest way of doing it.

The author mentions, and also shows in Fig. VIII. of his paper, that when the bar is rigged across from foot to hanging, the machine only gives 2 ft. between the two positions of over and under the bar, for parallel holes. If the machine was given less dip than shown a fairly well balanced bench would be the result, but why does the author not use the arm on the bar and take larger benches?

I take it from the reading of the paper that the benches were broken in rotation from top to bottom, which is a great mistake as I pointed out in 1908. They should be broken in rotation from bottom to top, as there would then be no danger of overrunning the benches or being choked by the rocks lodging against the points of benches, as the machines would be travelling away from the rock. I think the manner of breaking the benches is responsible for the low fathomage.

The amount of rock taken out of the stope should be 40%, and with a combination of overhand and underhand stoping should be about 70%, two-thirds being overhand and one-third underhand. The faces would have been better if carried at such an angle as to just save dry holes, or, say, about half the angle shown by the author, the rock would not roll so much the standing would be much better, and it would not introduce any disadvantage.

This closed the meeting, the President wishing all present a merry Xmas and a happy New Year.

Contributions and Correspondence.

STOPE MEASUREMENTS.

"Before discussing Mr. Tonnesen's paper on Stope Measurements, I think it well to arrive at some conclusion as to what the requirements of stope measurements should be. In the field work I consider accuracy of paramount importance. Rapidity and cost are essential factors while the other considerations of plotting the notes and determination of areas in the office come under the head of accuracy and rapidity. Convenience in the use of the working plans resulting therefrom, is by no means unimportant. Accuracy must not be sacrified for speed and also too much time must not be expended for the sake of a slight increase of accuracy. For this reason I have pointed out a method in the sub-joined paper,* securing a high degree of accuracy, in which the determination of dip plays no part. Any method which requires extended calculations is of little practical value. The method for ascertaining the slope area must be rapid and accurate both in the field and office, so that (say, for a mine of 300 stamps) the contracts may be out within three or four days after measurement has begun. Regarding cost, the method should be one that does not require too many assistants in its manipulation. My conclusion, from my own experience, is that, underground, the method of radiation with a transit or theodolite, reading slope distances, vertical angles, horizontal angles, in the office, reducing the slope distances to the the horizontal, the plotting of the reduced distances, the determination of the angle of dip, the determination of the horizontal area by means of the planimeter, the reduction of the horizontal area to the slope area by multiplying by the secant of the angle of dip, is the most advantageous one for all classes of stopes, flat, vertical, inclined and warped surfaces or any combination of them. In the case of vertical or nearly vertical stopes the area is projected on the vertical plane through line of strike, and the projected area reduced to the slope area by multiplying by the secant of the complement of the angle of dip. The plotting of the notes on the horizontal requires no further reductions, as would be necessary if the stope were plotted on the plane of the reef; and hence, as regards the working plan, is the most advantageous method. My experience is that the errors in the use of this method are due, more than to any other cause, to the faulty methods commonly used in the determination of the angle of dip. The methods I have indicated in the accompanying manuscript give the dip accurately, hence the slope area far more accurately than before and as accurately as any other method consistent with rapidity and the data at hand. Any one who has surveyed a stope knows how irregular the stoping face is, and, hence, the difficulty of judging just where to select the point in the face at which to hold the tape; if held too high or too low the variation in the slope distance may be half a foot and, hence, when introduced in any method of determining the slope area, will cause error. The most refined mathematical method will not compensate an error of this kind. In using this method only one assistant is absolutely necessary, although rapidity is enhanced by having two; if they are natives the cost is a very small item.

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* See this *Journal*, "Devices for Plotting Co-ordinates, Drives and Stopes," pp. [illegible].

Taking up Mr. Tonnesen's paper, which I have read very carefully and compared with my own experience, I may state that as regards paragraph 1, I concur with his views entirely. Regarding paragraph 2, I disagree most emphatically with his main contentions. I take exception to his statement that in a steep stope the setting up of a theodolite is a very lengthy and difficult (conceding that it is sometimes difficult) operation. I have always been able to survey the face of a steep stope by setting up in the upper drive or, if the face could not be seen from the upper drive, in the intermediate drive at the top of the stope, surveying part of the face from there and, then, setting up above a pillar in the stope or above the lower drive pillar and surveying the remainder of the stope from either one or both of these points. In connection with this point I may state that Mr. Brewer, of the Witwatersrand Deep Mine, has overcome this difficulty by devising an extensible bar carrying an adjustable arm to which the transit is clamped. It is true that at times I have had to keep myself in position by a rope fastened around my waist and fastened to a point above me while setting up and taking sights to the face of the stope. This has been the exception rather than the rule. In vertical stopes I have set up on one of the benches, sighted to one or more stations, thus locating the transit, and then sighted to the benches of the stope. For example see Fig. I., which represents a vertical section of the stope through the line of strike. In case a point cannot be seen with the instrument such as *c*, *ac* and *bc* were recorded, thus locating *c* from the observed points *a* and *b*. After taking the necessary sights I sighted to the backsight to see if the horizontal angle agreed with the first reading on the backsight. In no case do I recall any discrepancy in the two readings, thus showing that the instrument had not been shifted.

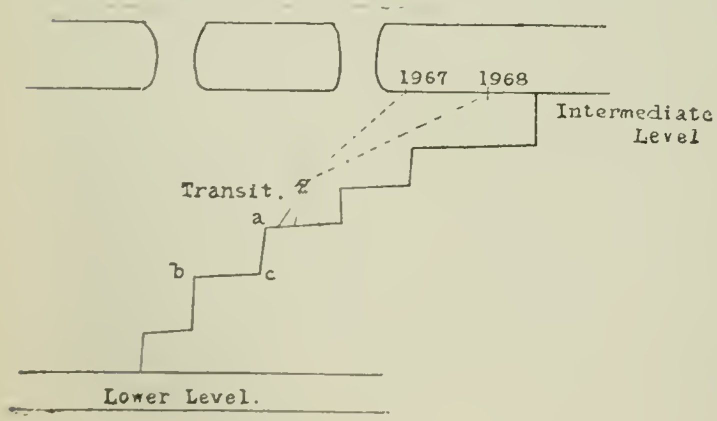

Fig. I.*

Mr. Tonnesen states that the method requires a large number of angles, and that the recording thereof is inconvenient. True, the method does require many angles, but if good judgment be used in sighting to the most prominent points of projection and recession of the face of the stope, their number need not be excessive, and, when one considers that the time used in recording them is small, the inconvenience is of no moment. The method of procedure that I use is to sight to the face, read the tape (always a steel tape, and reading to tenths of feet) record, and then read the horizontal and vertical angles (to 10 min. by estimation) record, and then unclamp instrument and sight to the next point. Almost invariably I have recorded the angles and been ready to sight to the next point by the time my assistant had arrived there, so that I cannot conceive of any time being lost. I must admit that the surveyor's position is somewhat trying at times, but by transiting the telescope and reading the other verniers he need not shift from his original position after beginning the sighting. That the office work is slow, is true by the methods usually employed—reducing the slope distances to the horizontal by means of tables, plotting the reduced distances with a scale and ordinary protractor. With the protractor I have devised, which was used on the Simmer and Jack Proprietary Mines, Ltd., for several months, this procedure is dispensed with, the slope distances being reduced to the horizontal mechanically on the instrument and the position of the point on the paper indicated almost simultaneously. One can easily plot thirty stopes in about eight hours. Stopes containing numerous pillars require a longer time, but even here, and more so, I claim that the method is the most rapid and accurate. I agree with Mr. Tonnesen that there is a liability to error in converting the horizontal area to the slope area. This error is due entirely to the method usually used in making the determination of the angle of dip. If the dip is ascertained by the methods I have indicated in the subjoined paper there should exist no cause for doubt as to the accuracy of the area thus determined. An average dip should never be used unless the entire area is equally distributed over the entire dip line, and not then if the dip line is such that its radius of curvature is a variable quantity. In a case of this nature the best plan is to find the average width of the horizontal area and multiply this by the length of the dip line. The equation of this line may be found by a method indicated in Wright's "Method of Least Squares," and thence the length found by the integral calculus. The most practical way, however, is as I have indicated in the following paper. In using the methods indicated good judgment must be used in the selection of the points to be sighted at. As an illustration of the difficulty encountered in the selection of the points, suppose the contour of the face is as represented in Fig. II., which is a section of the stope through the line of strike and at right angles to the line of dip at that point. Suppose (*a*) the leader or reef *c* has been selected as being the true direction of the plane of the stope, then if A is a depression in the face of the stope, it would not be correct to place the end of the tape therein. If an average of the section of the face were made, as indicated by the line (*hk*), then B would be the point to be sighted to ; but on the plan *c′* is the greatest recession of the face on plane of reef *c*, and point B lies within the face. In (*b*), (*hk*) is the average line and point B the point to be sighted to ; but on the plan *c′* is the greatest point of projection, and the point *b* lies within the face. The question is, how are the points *c* in (*a*) and *c* in (*b*) to be chosen so as to reconcile the vertical contour of the face and the horizontal contour, and have the points selected in the plane of the reef so as to arrive at an accurate value of the slope area. My plan has been to select the most prominent points of projection and recession of the face on a leader whose continuity is preserved throughout the stope, or, if this is not feasible, to assume points about halfway between the hangingwall and the footwall; but here again if the two walls are very irregular the plan is not feasible, and if this be the case, to judge where the points should be selected so that they may be on the average line of trend of the face of the stope, and to ignore the vertical line of contour of the face. (I write of the vertical line of contour as though the plane of the stope were revolved to coincide with the horizontal plane.) If poor judgment is used then errors are the rule and the resulting line of dip will be undulating with perhaps incongruous points. These incongruous points may be rejected in as far as the determination of dip is concerned, just as one would in a series of observations reject an observation at total variance with the remaining ones of the set, although retained for the horizontal plan. The waving curve may be averaged and the resulting line of dip will be the most probable one, and hence as accurate determination of the true dip as possible. A point to be taken notice of here is that, if the contractor works the stope for a long period of time the errors discussed above are of little moment (except as regards the determination of the line of dip), as what is lost one month is gained the next. If the contractor be released from his contract then greater care should be used in averaging the face of the stope.

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* Figs. I., II., VI. and VII. in this article are not to any particular scale, and are in some instances, slightly out of proportion; this has been done to show more clearly the idea to be conveyed.

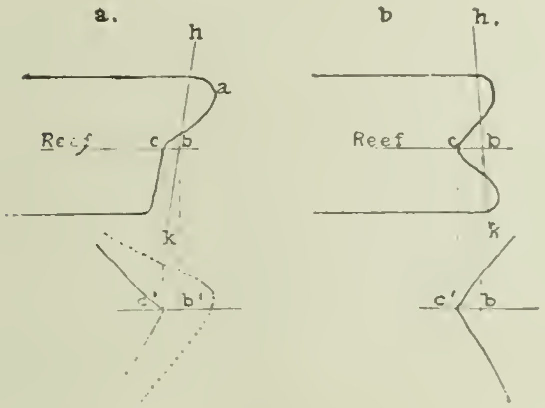


Fig. II.

In paragraph 3, without going into detail, I agree with Mr. Tonnesen's conclusions regarding the tape triangulation method of measuring stopes. Although he is in error (as Dr. J. Moir has pointed out) in regarding the curve of a sagging tape as being a parabola, the very slight error in so doing does not invalidate his conclusions. My objections to the tape triangulation method are the amount of time consumed in surveying the stopes, the method of triangulating in the stations, and the number of assistants required to hold the tapes. Accuracy is impaired in the establishment of the stations owing to the irregular nature of the hanging wall, and in the surveying of the stopes by the sagging of the tape and to offsets not being at right angles to the tape.

Although I have not had the opportunity of using Mr. Tonnesen's instrument, and am therefore unable to offer an absolute opinion of its merits, it would seem to me that the conditions essential for its application to secure accurate results would have to be ideal ; that is, the hanging wall would have to be regular, and hence the dip uniform. If such were the conditions, I can understand the device lessening the labours of observation and the subsequent reduction of notes. It would seem that a great deal of time would be required in bringing the three points into position, unless the stations were very favourably placed, especially if the backsight were behind one. In the case of stopes of varying dip it would appear that the instrument would be absolutely inapplicable, both in surveying the stope and establishing the stations therein : for the reason that a common reference plane for all points of the face does not exist, and hence if it were attempted to plot on the plane of the reef the areas would consist of areas that over lapped or contained hiatuses, depending upon whether the radius of curvature were positive, or negative as well as the position of the stations, and hence causing error and confusion. Fig III. and the following discussion thereof will suffice to illustrate my contention. Suppose a is the backsight station, and b the station under which the instrument is placed: say, under b the same distance as the point on the plumb line suspended from a is below a, and that d is the foresight at same distance below hanging wall. Then the angle measured is the angle abd. The next point sighted to is c, and the angle measured abc. Now the part of the stope in which dc lies is at a different slope from ba as may be seen by projecting dc to d_2c_2 and drawing line through d_2c_2, hence if abc is the plane upon which the area is represented then d cannot be in this plane, and if represented as if it were in this plane an error is perpetuated. If $d'c'$ represent two other points the error is more noticeable as $d'c'$ is at a greater inclination than dc. Now if it became necessary to use two new stations, as fg, for the next survey, for purposes of checking it would be necessary to sight to c of the previous survey, and, since the plane containing cfg is less inclined than abc, if the points were plotted as though all were in the one plane they would not coincide and confusion and errors would arise.

To show the error that would arise in the angles determined by Mr. Tonnesen's device if the plane of the reef was slightly misjudged, suppose the co-ordinates (found with transit and tape and reduced to a as the origin) of abc are :—

a : $x=0$	b : $x=10$	c : $x=20$
$y=0$	$y=15$	$y=2{\cdot}5$
$z=0$	$z=15$	$z=3$

Now suppose a second survey were made and a' were 1 ft. lower than a, or if a were sighted at directly it would amount to the same thing if the second surveyor held the instrument 1 ft. higher, at the same time making the same error at the face; then the co-ordinates of a' in the first instance would then be, $a' : x=0, y=0, z=1$. Now find the equation of the line through ab and through bd and then find the angle between these lines: then find the equation of the line through $a'b$ and then the angle between $a'b$ and bd.

If $x'y'z'$ and $x''y''z''$ are the co-ordinates of two points the equation of the line through them is $\frac{(x-x')}{(x''-x')} = \frac{(y-y')}{(y''-y')} = \frac{(z-z')}{(z''-z')}$, and if the equations of two lines are $\frac{(x-x')}{\mathrm{L}} = \frac{(y-y')}{\mathrm{M}} = \frac{(z-z')}{\mathrm{N}}$ and $\frac{(x-x')}{\mathrm{L}'} = \frac{(y-y')}{\mathrm{M}'} = \frac{(z-z')}{\mathrm{N}'}$ and θ the angle be-

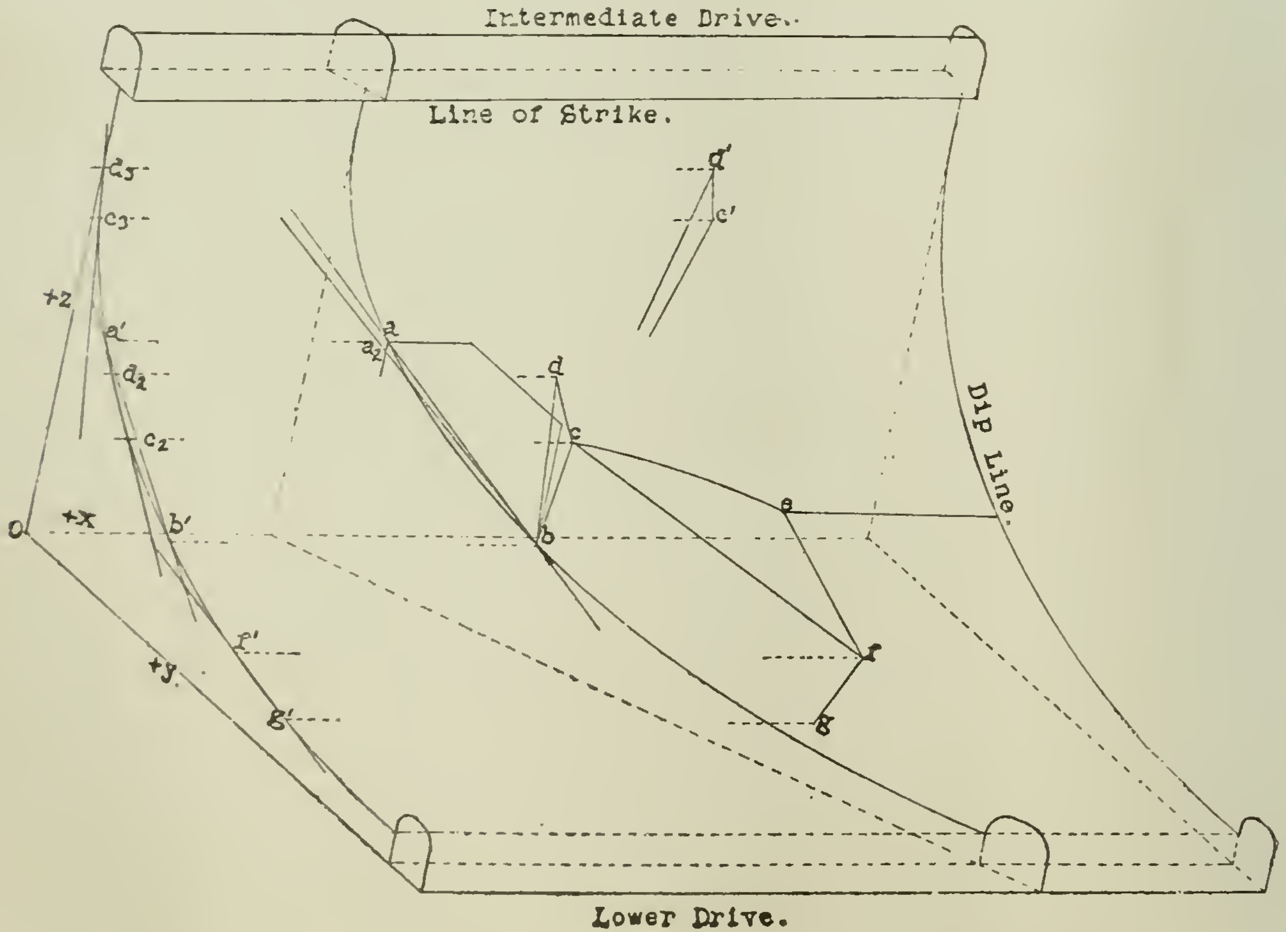


Fig. III.

tween them, then θ is found from

$$cos\theta = \frac{LL' + MM' + NN'}{\sqrt{L^2 + M^2 + N^2}\ \sqrt{L'^2 + M'^2 + N'^2}}$$

Hence equation of the line *ab* is $\frac{x}{10} = \frac{y}{15} = \frac{z}{15}$ and of the line *bd* is $\frac{x-20}{-10} = \frac{y-2{\cdot}5}{12{\cdot}5} = \frac{z-3}{12}$ and

$$cos\theta = \frac{-100 + 187{\cdot}5 + 180}{\sqrt{100 + 225 + 225}\ \sqrt{100 + 156{\cdot}25 + 144}}$$

$$= \frac{267{\cdot}5}{\sqrt{550}\ \sqrt{400{\cdot}25}} \text{ and } \theta = 55^\circ\ 14'\ 26''.$$

The equation of the line *a′b* is $\frac{x}{10} = \frac{y}{15} = \frac{z-1}{14}$

and of *bd* $\frac{x-20}{-10} = \frac{y-2{\cdot}5}{12{\cdot}5} = \frac{z-3}{12}$ hence

$$cos\theta = \frac{-100 + 187{\cdot}5 + 168}{\sqrt{100 + 225 + 196}\ \sqrt{100 + 156{\cdot}25 + 144}}$$

$$= \frac{255{\cdot}5}{\sqrt{521}\ \sqrt{400{\cdot}25}} \text{ and } \theta = 55^\circ\ 58'\ 13''$$

and the difference is 44′ 17″, which would certainly show in the plotting and give rise to an overlap. This is not an extreme result, as in a stope with a very irregular hanging wall an error of a foot in judging the plane of the stope can easily be incurred. Owing to the irregularities of the hanging wall of stopes, I am unwilling to believe that stations may be accurately established therein with Mr. Tonnesen's device. For accuracy, this necessarily involves the reading of a vertical angle, and hence, to my mind, requires the use of a transit or theodolite. Figs. IV and V. represent a vertical section at right angles to the line of strike showing the line of dip of two stopes (now worked out) on the Simmer and Jack Proprietary Mines, Ltd., which, I venture to state, would offer many difficulties in the application of Mr. Tonnesen's instrument. With a transit, and a determination of the line of dip, the area is very readily and most accurately ascertained.

An error that sometimes occurs, especially in steep wide stopes is illustrated in Fig. VI., which is a vertical section of a stope at right angles to the strike. Accurately, the horizontal projection of the length of the stope is *a′b′*, but if the assistant were to hold the tape at *c* the horizontal projection would be *a′c′*, and the contractor would gain an amount in width equal to

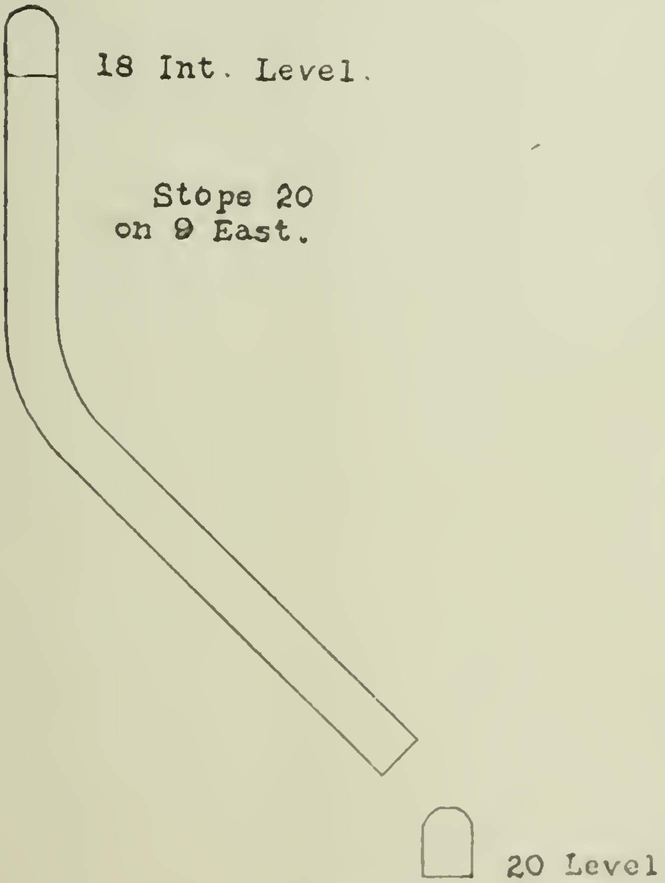


Fig. IV.

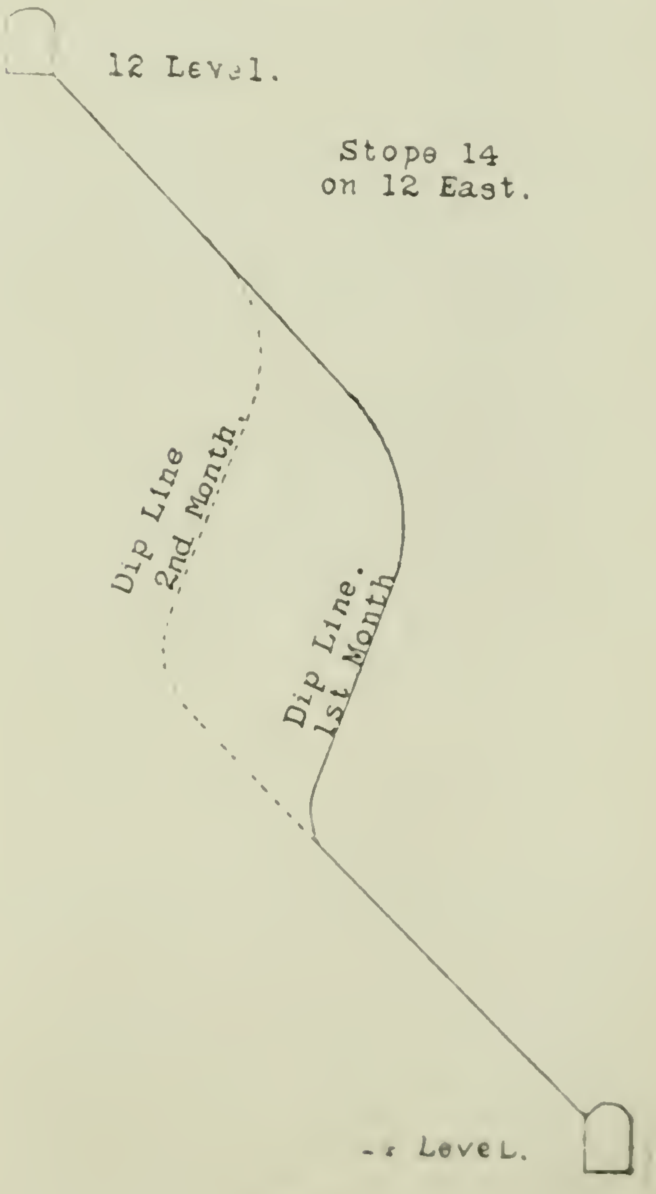


Fig. V.

$b'c'$ on the horizontal, and on the stope slope bh, and in length equal to the bottom width of the stope. This would also happen if the tape were held at d. This gives rise to some trouble with the contractor, as when he notices the tape being placed at a he at once concludes that he is being "shortened on the tape," as his conception is to hold the end of the tape as far from the transit as possible. And also the contractor may, contrary to orders, carry the reef in a position such as ab Fig. VI., so that the horizontal area of the stope should be multiplied by the secant of θ. If the contractor has been ordered to carry the reef as indicated then the area should be calculated using the angle.

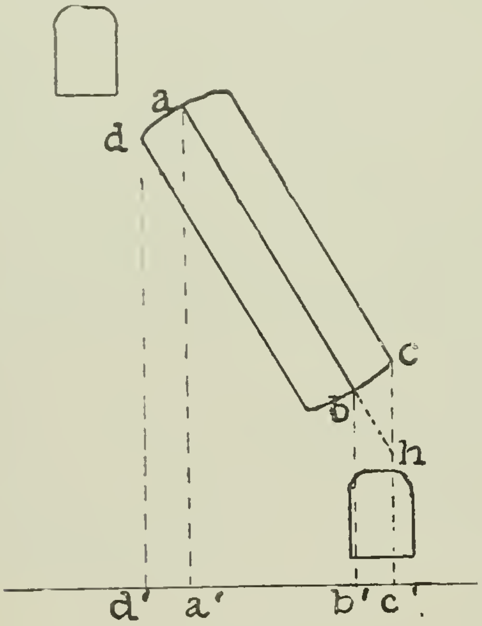


Fig. VI.

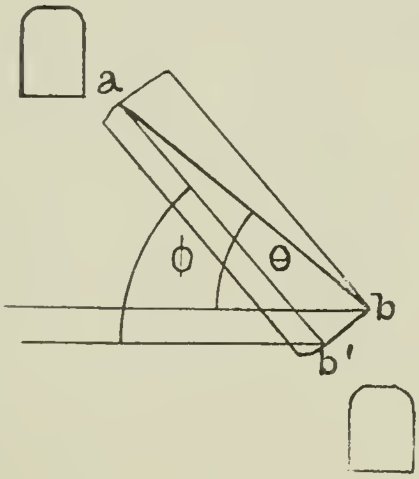


Fig VII.

Another source of error results from the careless use of the planimeter. With the rolling planimeter, unit of vernier reading 0·0005 sq. in., the error is for an area of 7·75 sq. in., 1/3000 of the whole; the error for the polar planimeter, vernier reading to 0·015 sq. in. is 1/355 of the whole, so that it is far more preferable to use the rolling planimeter where a high degree of accuracy is desired. A constant for the planimeter should be found depending on the character of the paper used for the stope sheets, and should be checked occasionally because of the contraction and expansion of the paper.

In conclusion, I venture the opinion that, if more care were used in the surveying of stopes with the transit, in the selection of the points sighted at, and in the determination of the angle of dip, the errors of measurement would be extremely small.

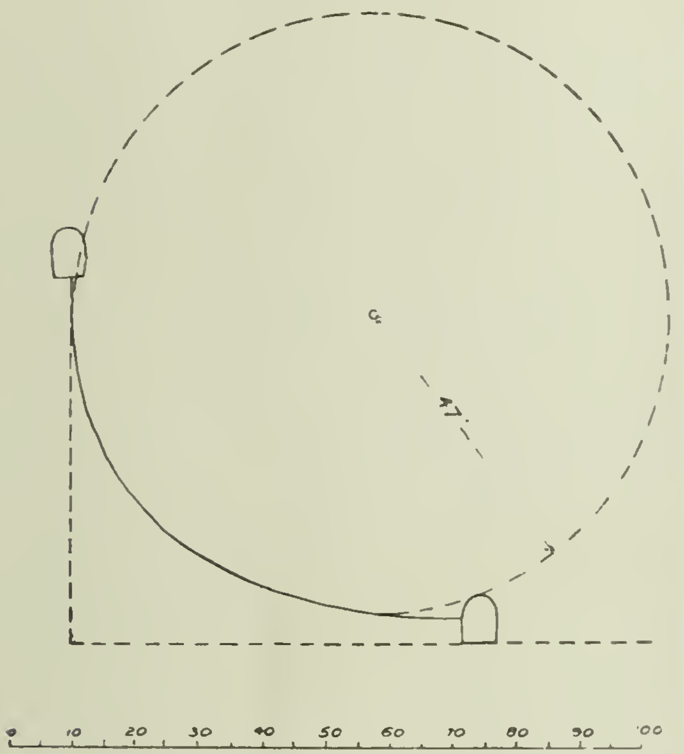
J. J. Bristol.

Reno, Nevada.

I fully endorse Mr. Bristol's introductory remarks, "A method which requires extended calculations is of little practical value." A decided advantage with the inclined method is that no calculations are required. The author proceeds: "I have always been enabled to survey the face of a steep stope by setting up in the upper drive, or if the face could not be seen from the upper drive then in the intermediate drive at the top of the stope, surveying part of the face from there and then setting up above a pillar in the stope or above the lower drive pillar and surveying the remainder of the stope from either one or both of these points." Mr. Bristol is most fortunate in the many exceptions from his "always" to have intermediate drives to cling to and convenient and accessible pillars to jump on and glue his theodolite to. But if this method is so easy and the setting up of a theodolite is neither lengthy nor difficult, why does the author recommend the use of "an extensible bar carrying an adjustable arm to which the transit is clamped"? I have also constructed an extensible bar but found it impractical for stope measuring owing to the varying stope widths and the weight and inconvenience of the bar. I should be able to measure the stope shown in Fig. 1 far quicker and cheaper than by setting up a transit.

"If good judgment be used in sighting to the most prominent points of projection and recession of the face of the stope their number need not be excessive." True, and equally true for any method.

Mr. Bristol's protractor and method I have already criticised elsewhere. To use methods of least-square and integral calculus for stope-measuring is absurd. Such calculations are only justified with elaborate triangulations, where the highest degree of accuracy is needed and obtainable. The difficulties illustrated in Fig. 2 applies to the theodolite method only, as in inclined methods the surveyor himself will be close enough to the face to direct the boys where to hold the tape. The author bases a proof against my method on an example illustrated in his Fig. 3. Such a stope would present equally great difficulties to the theodolite method. To show the absurdity of his example I have in Fig. I constructed a vertical section from Mr. Bristol's Fig. 3. The stope-dips vary from vertical to horizontal connected by a circular curve of 47 ft. radius. What an ideal place for shooting the shoot! If Mr. Bristol will take the trouble to re-read my paper he will find that it is based on the reef of the Witwatersrand. The Main Reefs of the Rand form to an astonishing degree true planes when considered in blocks. When faults occur the reef is broken up into smaller planes to be considered separately or developed on to one plane as described in the May *Journal.* The author's example is not applicable to our reef and the proofs derived therefrom thus void of practical merits. If the angle abd was measured *for radiation purposes* then an error of 44 min. in such a short line as bd = 20·01 ft. is of small account being 3 in. only, and if the line bd was longer then the resulting angular error would be less. If *traversing*, such an acute angle as 55° would never be needed for the traverse framework as one rarely would "go backwards to come forwards," but naturally choose lines in the main direction of the faceline. The greater the angle the smaller the error caused by 1 ft. drop of backsight disappearing entirely at 180°. The angles required in practice lie somewhere between 130° and 230°, where consequently the result of this error is negligible. The author has chosen his backsight line as short as 20 ft. in order to exaggerate the effect of the 1 ft. drop. In practice there is not much danger of such short lines being used.



Let us now for argument's sake for the time being *imagine* that in a stope for some unaccountable reason Mr. Bristol's error of 44 min. crept in.

Angle a b c (see Fig. 2) is measured = 150°14′

Correct angle s b c, = 150°58′

The points a, b, c are plotted on a tracing paper and points a and c located on to the known points a and c of the stope-plan.

Co-ordinates :

a	b	c	c'
$x = 0$	$x = 0$	$x = 49\cdot65$	$x = 48\cdot53$
$y = 0$	$y = 100$	$y = 186\cdot81$	$y = 187\cdot43$

Length $ac = 193\ 3$

" $ac' = 193\cdot6$

Equation of line ac : $y = \frac{186\cdot81}{49\cdot65}x$

" " ac' : $y = \frac{187\cdot433}{48\cdot53}x$

Distance from b to line $ac = \frac{-100}{\pm\sqrt{1+3\cdot807^2}} = 25\cdot41$

Distance from b to line $ac' = \frac{-100}{\pm\sqrt{1+3\cdot862^2}} = 25\cdot07$

The error at b = 25·41 – 25·07 = 4 in. only showing how this method of plotting reduces the angle-error.

Figs. 4 and 5 in Mr. Bristol's paper will cause little difficulty when using tape-triangulation and do not "involve reading of a vertical angle and hence require the use of a transit." The author's Figs. 6 and 7 are arguments weighing very heavily against his own method and prove the enlargement of error likely to occur when working from less to greater. Such causes of errors are entirely avoided in tape-triangulation and all inclined methods, where the offsets are plotted on the plane of the reef and therefore no horizontal errors come into account.

O. Tonnesen.

East Rand.

DEVICES FOR PLOTTING CO-ORDINATES.

I have read with great interest Mr. J. J. Bristol's paper on "Devices for Plotting Co-ordinates, Drives and Stopes.*

1. *Devices for Plotting Co-ordinates.* — The author claims the following advantages : "The triangles save considerable time in plotting, are accurate, and do not soil the plans as there are no erasures to be made." I had some triangles made and experimented with them, in order to prove the merits of the system. The experiments did

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* See this *Journal*, pp. [illegible].

not show any noticeable saving of time. Neither did the triangles improve the accuracy in plotting co-ordinates. The distances between the axes of the co-ordinate framework will generally vary somewhat, no matter how carefully this is drawn up, owing to contraction and expansion of the drawing paper. Such errors are generally interpolated when plotting co-ordinates with ordinary scale, the zero points of the scale being placed so as to practically mean the error. This adjustment can not be done with Mr. Bristol's triangles. Further, the rectangularity of the co-ordinate framework is often somewhat distorted owing to *uneven* contraction of the paper or inaccurate draughtsmanship. The ordinary plotting method by scales allows also for this error, plotting parallel to the frame-ruling, thus giving a *relatively* correct position of the co-ordinates. But the set-square method does not allow for this error. Mr. Bristol's triangles do save some erasure. A disadvantage of the system is the complication and the great number of set-squares to be kept in stock, one for each scale required.

To transfer a plan to a different scale, I should prefer the old method of using two sets of scales and offsets, because this method has the advantage, that the base-line for offsetting can be chosen in any direction convenient to the main strike of the plot, whereby more convenient and accurate intersections are obtained and greater simplicity.

2. *Device for Plotting Drives.*—Disadvantages are: There are two scales on each protractor leading to confusion. The protractor is graduated the reverse way. The protractor-readings must be located over the back-sight line, and when plotting many points from the same datum-line, the use of the protractor must be rather slow, as one has constantly to change its position for every point plotted. This continual turning about the centrepoint has also a tendency to spoil the plan with big centre-holes, a far greater disadvantage than erasure of pencil lines.

3. *Device for Plotting Stopes, "The Plotting Protractor."*—This protractor seems rather complicated, consisting of so many moving parts, which may easily get out of adjustment. For each determination of horizontal and vertical distances one has to make four readings and three movements on Mr. Bristol's protractor, while a slide-rule will give the same result in three readings and one slide movement. In section 26 the author divides the horizontal area with the horizontal length W and multiplies this result by the length of the dip-line (*vh*). He has thereby distributed the several *uneven* parts of horizontal areas *evenly* and *uniformly* along the total dip-line instead of weighting the small horizontal areas according to the length of their corresponding functions of the dip-line, which error may in cases of irregular stoping be considerable.

Mr. Bristol determines in an example the dip *θ* to 35° 17' and later to 36° 24', *an error of 1° 7'*. He points out "that the points sighted to must, to ensure accuracy, be chosen halfway between hanging-wall and foot-wall or on the line of trend of the stoping-face." However, in using the method of *inclined* stope measuring the above mentioned area error is avoided and it does not greatly matter, whether the off-set points measured are accurately halfway between hanging and foot, as the *inclined* error thus caused is negligible compared to the *vertical* error. The inclined method also takes into due consideration "warping and change of dip" in the natural unfolding process involved in the system.

The author even goes so far in theorising as to consider "conical stopes"!

To digest the last part of Mr. Bristol's paper is a rather severe test to practical men, however interesting the calculations may be. I consider simplicity to be the main aim in avoiding errors. In spite of much excellent work, I do not think that the author's methods are clear and simple enough for general practice.

In conclusion I wish to express thanks to Mr. Bristol for the keen interest he is taking in the subject.

O. Tonnesen.

East Rand.

SAND FILLING ON THE WITWATERSRAND.

Mr. Edgar Pam's paper on the above subject and the discussion thereon has afforded myself—and doubtless numerous other members—in outside districts very interesting and instructive reading.

Without wishing to enter into the discussion of this system of stope filling, which I think I mentioned some time ago as being adopted, in a rather primitive method, on the Jubilee Mine in 1889 or 1890, I would like to give expression to an idea which has struck me, and doubtless other members, in connection therewith, viz., the possible utilization of the power which could be generated by the fall of the sands in the shaft pipe lines and which is at present running to waste. Surely some of our mechanical or electrical engineering members can devise some scheme whereby the power at present lost can be utilized for, say, ventilating purposes or for raising the waste water to surface again.

An improved Pelton wheel or turbine giving ample clearance for the discharge of the pulp and which would not allow of any accumulation of the sands in the pipes could surely be devised which would answer the purpose.

Without going further into the subject at present, I would leave the suggestion for the consideration of my fellow members in the hope that it may bear fruit in the shape of a "paper" at some future date.

Jno. Moyle-Phillips.

New Lisbon Berlyn, Ltd.,
Pilgrims Rest.

THE EFFICIENCY OF LABOUR UNDERGROUND.

I am pleased to see that the few remarks I made on Mr. Tom Johnson's paper, on "The Efficiency of Labour Underground," and in which I drew comparisons between the efficiency obtained from hand and from machine stoping, (results which worry our mine managers) have been deemed worthy of criticism and enlargement by Mr. Weston.

My excuse for referring again to the subject is the fact that Mr. Weston has placed a wrong construction on what I said. I certainly made no reference to the law of blasting, which states that the quantity of explosives necessary to break a hole, varies as the square of the burden on it, which is one reason why hand stoping should be more economical than machine stoping in explosives used. This law is known by all mining students, and is hammered into the heads of students attending the evening classes at the S.A. School of Mines here. I did *not* advocate drilling *one hole* benches with our heavy machines, as Mr. Weston states I did. No practical machine stoper would even dream of trying such an idea. I suggested trying *two hole* benches, instead of the four to six hole benches, and blasting only one of these holes on every bench at a time.

My objection to the narrow gauge chisel bit, is construed by Mr. Weston to mean that I consider it a mistake to drill these long holes. Such is not the case. I advocated the use of a wider gauge star bit, in place of the narrow chisel bit, and of the same length, not shorter. The narrow gauge chisel should answer well for holes pitched, as Mr. Weston suggests, to look out, but of course good judgment must be exercised to avoid "bull ringing." In my limited experience of Rand mining, I have noticed that the majority of machine stopers make an invariable practice of keeping their holes parallel to the face, or inclined to look in, with the object, so they say, of avoiding "bull ringing." If they use the narrow gauge chisel, a deep socket is the natural result after blasting, and this was my reason for advocating a *long* star bit, in place of the *narrow* chisel. Besides this, why resort to the practice and expense of chambering the bottoms of these narrow holes, when it can easily be avoided by providing wider gauge drilling steel !

In conclusion, I must apologise for taking up your time and space, and for my previous remarks being so obscurely expressed as to be misunderstood.

E. J. Wiseman.

Johannesburg.

Standardization of Metallurgical and Mining Terms.

The Council has adopted the following report of its Sub-Committee on this subject, and authors and contributors are requested to comply with its recommendations when submitting papers and contributions to the Society.

This Sub-Committee was appointed by the Council to consider the advisability of furthering the Standardization of Metallurgical and Mining Terms and Phrases.

After considerable discussion, it was decided that where variations in spelling and in application exist in the use of such terms, one form should be recommended and adopted in all papers, etc., published by the Society, even although alternative forms be not inaccurate.

It will be evident that many of the words have reference *mainly to practice on the Rand.* Where the meaning of any word is not clear, an explanatory note is attached.

1.—Except when otherwise necessary, the following words shall be treated as collective nouns, and therefore used in the singular :—

Sand.
Tailing.
Fine.
Black Sand.
Residue.
Ash.
Pyrite (FeS_2).
Concentrate.
Slag.
Sulphide.
Telluride.
Slime.
Chloride.

2.—The following terms shall be used in the plural form :—

Shavings.
Skimmings.
Steamings.
Sweepings.
Drillings.
Reserves.

3.—Preference should be given to the singular form in .—

Precipitate. Bin. Cost. Profit. Value.

4.—Such abbreviations as the following shall not be printed in the plural :—

Oz. lb. dwt. gr. cc. cm
kg. in. gal. m. gm. mm.

5.—As far as practicable the following terms shall be avoided :—

Dirt.
Mullock or Muck. (Broken ore or waste.)
Kokopan. (Truck.)
Cheesa Stick. (Firing Stick.)
Lasher. (Shoveller)
Lashing. (Shovelling.)

6.—The following spelling shall be used consistently :—

By-product.
Shoot (the geological term).
By-pass.
Hydraulicking.
Siphon.
Chute (passage or opening through which material is guided).
Kafir.
Grate.
Dike.
Rise.

7.—"Vat" shall be used in place of "Tank."
"Cyaniding" shall be used in place of "Cyanidation."
"Breaking and Sorting" shall be used in place of "Crushing and Picking."
(Note: "Crushing" equals "Stamp milling" plus Tube milling.")
"Ore Slime" to distinguish from "Gold Slime."
"Extraction" shall refer to metal extracted from ore.
"Precipitation" shall refer to metal precipitated from solution.
"Recovery" shall refer to metal won in a form suitable for the market.

8.—In referring to screening, it should be stated whether number of holes refers to the square or linear inch, and the diameter of the aperture in decimals of an inch should be given; battery screening should have the gauge of wire in decimals of an inch and the percentage of discharge area mentioned, wherever possible.

Temperatures shall be expressed in degrees Centigrade (with the equivalent in Fahrenheit in parenthesis).

Cyanide and alkali strength shall be expressed in terms of specified reagents, thus:—

0·05% KCN; 0·02% NaOH; 0·02% CaO.

9.—A ton shall be 2,000 lb.

A fluid ton shall mean 32 cub. ft.

10.—The following scheme illustrates the nomenclature of Rand Reduction Works:—

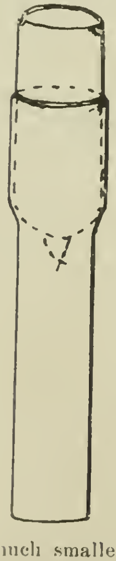
Reduction Works
- Breaking Plant
- Crushing Plant
  - Stamp Mill — Sets of Stamp Batt'ries
  - Tubemills or other secondary Crushing Machines
- Cyanide Plant
  - Sand Plant
  - Slime Plant

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

SENSITIVE REAGENTS FOR PHOSPHORIC ACID AND PHOSPHATES.—"The author has prepared a series of reagents for phosphoric acid and phosphates. 0·5 grm. of quinine sulphate is shaken with 50 cc. of water, 10 cc. of nitric acid of sp. gr. 1·39 are added and when solution is complete, 10 cc. of ammonium nitromolybdate reagent are introduced and the whole made up to 100 cc. (The nitro-molybdic reagent is made by dissolving 15 grm. of ammonium molybdate in luke warm water, and making up the volume, after cooling, to 100 cc. with water, then pouring this solution into a mixture of 50 cc. of pure nitric acid, sp. gr. 1·390, and 50 cc. of water.) Similar reagents may be prepared either with 0·5 grm. of antipyrine, or 0·5 grm. of strychnine sulphate, instead of quinine sulphate. That prepared with antipyrine becomes rose coloured, then red in about a day's time, tending to become yellow finally. That with strychnine becomes gradually of a picric yellow colour. These changes of colour, however, do not affect the delicacy of the two reagents. In preparing the antipyrine reagent, the antipyrine is dissolved in about 50 cc. of water before adding the nitric acid and the solution of nitro-molybdate, while in the case of strychnine sulphate, the salt (0·5 grm.) is dissolved in 20 cc. of water by heat, adding, immediately after, 50 cc. of cold water, then 10 cc. of nitric acid and 10 cc. of the nitro-molybdate solution, finally making up to 100 cc. with water. A few cc. of these reagents precipitate abundantly with a few drops of very dilute phosphoric acid, free or combined. A few hundredths of a milligram of phosphorus, in the state of phosphoric acid, per litre may thus be detected by the strychno-molybdic reagent, which is the most delicate of the three. In extreme dilutions only a turbidity occurs. These reagents are precipitated by tannin and salts of mercury, but both may be got rid of by calcination before testing for phosphates. Arsenic acid and the arsenates likewise precipitate the reagents, but in a less degree than phosphates, but the arsenical precipitate obtained with the quino-molybdic reagent is white, while that with phosphates becomes yellowish very quickly."—G. DENIGES, *Bull. Pharm. Bordeaux*, May, 1910, 195; *Pharm. J.*, 1910, 85, 321.—*Journal of the Society of Chemical Industry*, Sept. 30, 1910, p. 1136. (J. A. W.)

A CENTRIFUGAL PRECIPITATING AND FILTERING VESSEL.—"Some of the most irritatingly tedious of chemical operations, the settling of precipitates and the filtering of fluids containing suspended matters, also such processes as the draining of crystals, may be enormously hastened by the use of centrifugal appliances, and it was with the object of taking advantage of this fact and applying it more widely in laboratories where no more elaborate centrifugal machine than a simple Leffmann-Beam milk centrifugal is available, that the little vessel shown in the accompanying sketch was devised. It is composed of an outer tube into the enlarged upper part of which an inner cup slides, the lower part of the cup resting on the lower and narrower portion of the tube. The inner cup, which acts as a funnel, is drawn out at the bottom into a short neck which is cut off at an angle. A channel at one side of the outer tube facilitates escape of air. The size of the appliance as a whole is governed by the dimensions of the centrifugal receptacles. The tube without the cup may be used for tests both qualitative and quantitative; substances may be weighed, dissolved, and treated in it, with the appropriate reagent; on then whirling it in the centrifuge for a few minutes the precipitate will often settle at the bottom into so compact a cake that decantation may replace filtration. Washings, too, may be carried out with much smaller volumes of liquid in this way than when conducted under ordinary conditions. Where



filtration is necessary, and it can be hardly avoided in quantitative work, the supernatant fluid is transferred to a filtering cup suitably provided with a filtering medium, such as a plug of asbestos, a filtering thimble, or a 'plaited' filter, preferably of hardened paper. As long as filtration proceeds rapidly there will be but little gained by attempting to hurry it by centrifuging, but as soon as it slackens and in any case before adding a fresh wash water, the cup should be slipped into a filtering tube and whirled for a minute or so, whereby much more perfect draining is ensured, and as a consequence, more rapid removal of soluble matter results. To lessen chance of breakage, actual contact between the cup and the tube at the shoulder of the latter should be prevented by placing between them two small strips of hardened paper, or by cutting a piece of such paper into the shape of a star with central hole through which to pass the neck of the cup.

While no great originality is claimed for this little appliance the writer, having found it of frequent use in the laboratory, feels justified in commending it to the notice of other chemists. It may be obtained from Mr. Riley Robson, 52 Fore Street, London, E.C."—*Chemical Trade Journal*, August 13, 1910, p. 145. (T. D.)

Hydrolysis of Cyanogen.—" Electrical conductivity measurements lead to the conclusion that the primary reaction between cyanogen and aqueous potassium hydroxide proceeds completely according to the equation, $2KOH + C_2N_2 = KCN + KCNO + H_2O$. The subsequent reaction forming oxalic acid, azulmic acid, etc., proceeds much more rapidly as the concentration of the hydroxide is increased, so that for concentrations greater than normal the primary reaction can no longer be quantitatively recognised. The reaction of cyanogen with water is primarily $C_2N_2 + H_2O = HCN + HCNO$. The degree of hydrolysis of a saturated cyanogen solution at 0° C. is about 0·02%, the corresponding value for chlorine being 0·16%. Determination of the dissociation constants leads to the conclusion that cyanic acid ($K \times 10^9 = 120{,}000$) is many times stronger than hydrocyanic acid ($K \times 10^9 = 1{\cdot}3$). The mobility of the CNO' ion (56·5) at 18° C., calculated from conductivity measurements corresponds very closely with the previously determined mobility of the CNS' ion (56·6)."—R Naumann, *Z. Elektrochem*, 1910, *16*, 772-777.—*Journal of the Society of Chemical Industry*, Oct. 15, 1910, p. 1156. (J. A. W.)

Titanium in Ores.—" The method as outlined for the determination of titanium in rutile and iron ores is based on the volatilisation of the silica by hydrofluoric acid in the presence of sulphuric acid, evaporation to dryness and fusion with sodium carbonate and a little sodium nitrate to convert the iron and titanium to insoluble ferric oxide and sodium titanate, extraction with hot water to remove the soluble phosphates, sulphates and aluminates; solution of the ferric oxide and sodium titanate in hydrochloric acid, extraction of ferric chloride with ether; and then either reduction of slight traces of iron with sulphur dioxide, precipitation of the titanic acid by boiling in acetic acid solution, filtration and ignition to titanium oxide; or a colorimetric determination by means of hydrogen peroxide. The authors find that the volumetric method based on the reduction of titanium to the trivalent state and oxidation with permanganate gives low results. With the sulphur dioxide reduction when considerable iron is present, the titanium product is usually contaminated with some iron. The ether separation removes practically all iron from the titanium very quickly. Re-fusion of the ignited product with sodium carbonate for purification, which requires considerable time, is never necessary by use of this method. The final titanium oxide is pure, unless zirconium is present, in which case this element is removed by the usual phosphate precipitation. This method combines the colorimetric and gravimetric determinations and the colour comparison is made in solutions always totally free from iron. For convenience hydrochloric rather than sulphuric acid solutions of titanic acid are used. The method is accurate and not long."—O. L. Barnebey and R. M. Isham.—*J. Amer. Chem. Soc.*, 1910, *32*, p 957—962. *Journal of the Society of Chemical Industry*, p. 1061. (J. A. W.)

Methylene Blue, indicator in Iodometry.—" A 0·005% solution of methylene blue may be used instead of a solution of starch in iodometric titrations; when a solution of iodine in potassium iodide is added to a solution containing methylene blue, an iodo compound of the colouring matter is precipitated, but in dilute solution this reaction is indicated by a change from blue to yellowish-green, and finally a yellowish-brown colour is developed. One cc. of the above-mentioned solution is a suitable quantity to employ as an indicator in a titration, and this quantity requires only 0·2 cc. of $N/1000$ solution of iodine to effect the colour change."—F. S. Sinnatt, *Analyst*, 1910, *35*, 299—310. *Journal of the Society of Chemical Industry*, August 15th, 1910, p. 979. (A. W.)

Electrolytic Determination of Copper.—" The electrolytic method is considered by many who have used it as the most accurate for the determination of copper, provided that care be taken that the metals, which are deposited under the same conditions as copper, are, previously, carefully removed. The great drawback is the consumption of a large amount of time in making the deposition, for an accurate result is often desired in the shortest possible period. When the deposition is allowed to run over night and none of the chemist's time is taken, the time element can not always be considered an inconvenience. Still, be one ever so careful, accidents are likely to occur, and if it so happens, with the method in ordinary use it means a wait of at least a number of hours before it is possible to obtain a result. Therefore, one can see at a glance the value of a rapid method of deposition.

Within the last few years there have been worked out, among many others, three rapid electrolytic methods, each of which, when properly operated, will, in a few minutes, give good results. Each involves a principle, long used commercially by men doing electroplating, *i.e.*, that agitation of the electrolyte, by keeping the metallic ions in contact with the pole on which the deposition is to be made, will greatly shorten the time in which one can make a deposition in a good, adherent form. Herein lies the possibility of the utilization of much higher currents, a fact which has only within the last few years been used by chemists making the deposition of different metals for the purpose of quantitative determination.

The oldest successful method for accomplishing this agitation, was, the rotation of either pole, *i.e.*, the anode or the cathode, or mechanical stirring or by other means.

The use of the rotating anode for chemically pure solutions, has been worked out, on not only nearly all of the elements which can be electrolytically deposited quantitatively but, as well, in case of many separations and for some ores, as for copper. In scientific laboratories, where the comparatively complex piece of apparatus necessary for this operation can be skilfully enough handled that it does not readily get out of order and the inconvenience is not great should something go awry in the mechanism, this method can be called a perfect success, for, by its agency, it is possible to deposit all of the copper which the analyst will find in a one gram sample, in from five to ten minutes. For routine work, in commercial laboratories, complexity is a point to be done away with.

A simpler form of apparatus is the solenoid. This causes the rotation of the electrolyte by means of magnetic stress set up in the solution by the electric current passing through it. But here, again, though in a lesser degree, occurs the disadvantage of complexity, and, furthermore, with this, one can not deposit the copper with the same rapidity as with the rotating anode or cathode.

In either of the above cases the results are, however, as accurate as with the older methods and practically the same electrolytes can be used.

Of rapid methods, the latest published is probably the best for the practical workman, combining, as it does, the simplicity of the older with much of the rapidity of the newer methods. In this, use is made, in the agitation of the solution by:

(1) The passage of high currents, causing (a) the evolution of large amounts of gas; (b) heating of the solution to such an extent that it boils.

(2) The free circulation of the electrolyte, allowed by the use of the gauze electrode.

Its application to the precipitation of chemically pure copper solutions has recently been experimented with by several chemists, including the author, who has also worked out a means for the rapid analysis of copper ores by this method. It is as follows:—

Process for Ores.—From 0·5 to 1·0 gram of ore, which contains none of the interfering elements, is dissolved in nitric or nitric and hydrochloric acids. When aqua regia is used, or if lead is present, sulphuric acid is added and evaporation continued until heavy white fumes are evolved. The excess of sulphuric acid is neutralized with ammonia and three cc. of concentrated nitric acid added. Provided sulphuric is not added it is only necessary to concentrate to three cc. In either case the solution is diluted to 75 cc., transferred to a tall battery beaker, without being filtered (if, however, the residue does not settle it is better to filter) and the copper deposited by means of a current of eight to ten amperes and three to four volts. If the deposit is black or dark colored it is dissolved in nitric acid and re-deposited in the same way. This re-deposition is, as a rule, easier than filtering. The following results show the accuracy of the method on ores, as compared with those obtained by the old method:

Result obtained with the Gauze Electrode.

No. of Sample.	Amount of Ore taken.	Per cent. Copper in ore.	Copper found.
4	0·5003	16·49	0·0825
4	0·5003	16·47	0·0825
6	0·5003	15·39	0·0770
6	0·5003	15·37	0·0769
Results obtained by the Ordinary Methods.			
4	0·5000	16·50	0·0825
4	0·5000	16·50	0·0825
6	0 5000	15·40	0·0770
6	0 5000	15·38	0·0769

Process for Slags.—In case of slags and other low grade material, it is necessary to select a method of disintegration suited to the particular material in hand. Slags, which always give large quantities of gelatinous silica and contain low percentages of copper, give the most accurate results when disintegrated by means of nitric acid followed by hydrofluoric and sulphuric acids. After evaporation until heavy white fumes are evolved, the analysis is finished, as in the process for ores. Re-deposition is more frequently necessary with low grade material than with high, but in all cases is more rapid, as well as more accurate, than filtering. The following will illustrate the accuracy of the procedure:

Determinations made by means of the Gauze Electrode.

No. of Sample.	Ore taken.	Found.	Copper in ore.
13	1·0000	0·0060	0·60
13	1·0000	0·0059	0·59
13	2·0000	0 0123	0·61
11	2·0000	0 0103	0·51
11	2·0000	0·0101	0·50
Determination made by means of the Ordinary Method.			
13	2·0000	0·0121	0·60
13	2·0000	0·0121	0·60
11	1·0000	0·0051	0·51
11	1·0000	0·0050	0·50

Elements which interfere, and, at the same time, are likely to occur in copper ores, are arsenic, antimony, lead, and bismuth. The separation of antimony and bismuth from copper was tried under the preceding conditions, but without success. Lead, in such quantities that it will deposit with the copper, is best removed as the sulphate. Arsenic, in small quantities, will not deposit with the copper, but in quantities about equivalent to the copper present begins to be deposited with the last of this. The following series of experiments, in which copper was deposited in the presence of varying amounts of arsenic, illustrates the point:

Arsenic taken.	Copper taken.	Copper found.
0·0605	0·2652	0·2648
0·1210	0·2751	0·2747
0·1815	0·2183	0·2183
0·1210	0·3038	0·3035
0·2420	0·2686	0·3106
——	0·2028	0·2026

All these deposits were made under the usual conditions. From these results it is seen that arsenic, when present in amounts large enough to contaminate the copper, can be easily removed by dissolving the deposited copper and re-precipitating under the same conditions as before. The removal of the interfering elements by chemical means can be accomplished by the following method:

The ore is disintegrated with nitric or nitric and hydrochloric acids. Five or six cubic centimetres of sulphuric acid are added and it is then evaporated until heavy white fumes are given off. It is diluted to one hundred and fifty cubic centimetres with water, boiled to insure complete solution of the basic sulphate of iron (a few drops of hydrochloric acid are added, if silver is present), filtered and washed with water containing a little sulphuric acid. The lead and silver remain on the filter paper. The filtrate is now heated to boiling and saturated with hydrogen sulphide and allowed to cool during the process. The precipitated copper sulphide is filtered off and washed with water containing hydrogen sulphide. The precipitated sulphides are rinsed from the filter paper into a beaker using as little water as possible, warmed with a little colorless sodium sulphide to remove the antimony and arsenic and again passed through the same filter, being washed with some water containing sodium sulphide. Bismuth is now removed from the copper by warming the mixed sulphides with a solution of potassium cyanide alkaline with ammonia. This dissolves the sulphide of copper leaving that of the bismuth insoluble. The solution of the copper is boiled with nitric acid to destroy the potassium cyanide and electrolyzed in the usual manner."—PROF. R. C. BENNER, *Los Angeles Mining Review*, September 17, 1910, p. 12. (H. A. W.)

THE QUANTITATIVE DETERMINATION OF BARIUM SULPHATE IN THE PRESENCE OF SUBSTANCES WHICH INFLUENCE THE RESULT.—"The author details his work and contends to have established the following points:—

1. The presence of potassium, sodium, ammonium, calcium, magnesium, iron and cobalt work disturbingly on the estimation of sulphate as $BaSO_4$ as they contaminate the precipitate and cannot be washed out in the usual manner.

2. The impurities appear in the precipitate as sulphates, and they go so slowly into solution by washing with water that it is impracticable to try and get rid of them in this way.

3. Calcium sulphate is the principal impurity, and if the solution to be examined contains a sufficient amount of this base, then even in the presence of the other bases, $CaSO_4$ will be the only impurity found in the precipitated barium sulphate.

4. Calcium sulphate can be dissolved out of the $BaSO_4$ precipitate by boiling for several hours in strong HCl.

5. Other salts, however, are not removed in this manner.

6. For an exact estimation of SO_4 ions in solutions containing other salts it is advisable to make sure that an excess of calcium is present. After the precipitation the solution is boiled with sufficient HCl to make 20% thereof, whereby all the $CaSO_4$ is converted into $BaSO_4$ by the $BaCl_2$ present.

7. A mixture of HCl and HNO_3 may be used instead of HCl alone, in which case 10% is enough.

8. If the $BaSO_4$ is contaminated with $BaCl_2$ the latter cannot be got rid of by boiling with HCl.

9. The precipitate will not contain $BaCl_2$ if the latter is used in a 10% or still more dilute solution, and dropped drop by drop into the boiling solution under examination, shaking all the while. There must not, however, be much HCl or HNO_3 present.

10. If the solution to be examined contains either of these acids, the excess must be removed by evaporation.

11. To estimate barium in a solution containing foreign salts, it should be brought slowly into an excess of boiling dilute sulphuric acid.

As a general prescription for an SO_4 ion estimation in a solution containing also Am, K, Na, Mg, Ca, Fe or Co, the author recommends the following procedure:—

An amount of pure calcium chloride is added to the solution which may not contain much HCl or HNO_3, at least double the equivalent of the probable SO_4 ions present. A precipitate of $CaSO_4$ may come down but does not interfere. (If the solution contains much HCl or HNO_3 it is best removed by evaporation on a water bath.)

After the addition of the $CaCl_2$ the whole is boiled and a solution of not stronger than 10% $BaCl_2$ added drop by drop with continued shaking. The amount of $BaCl_2$ added should be about double that theoretically necessary. After the precipitation the solution is heated to boiling, and enough HCl of 38% added to make 20% of the whole, or one may add a mixture of HCl (25%) and HNO_3 (50%) 3 : 1 sufficient to form 10% of the total solution. The mixture is then boiled for three hours on a small flame. When a mixture of HCl and HNO_3 is used for this purification it is necessary during boiling to now and again add a few drops HNO_3. The precipitate is allowed to settle and the solution decanted off as completely as possible into a glass basin. The precipitate is then boiled for a further period of three hours in 150 c.c. of at least 20% HCl or HCl and HNO_3 with the addition of a few cc. $BaCl_2$ solution. After settling this solution is also decanted into the basin and the precipitate treated a third time in the same manner. The three solutions are evaporated nearly to dryness, taken up with water, heated a little, and carefully added to the precipitate in the beaker or Erlenmeyer flask. The whole is then boiled for several minutes in 100 c.c. distilled water, and at once filtered. If the original solution contained iron salts it is necessary to add to the evaporated mass after taking up with water about 5 cc. HCl and a few drops HNO_3 in order to prevent the separation of iron. This will only be necessary if the iron chloride is decomposed by carrying the evaporation too far. When the method is carried out exactly as here stated, not only are all the sulphate ions precipitated as $BaSO_4$ but the precipitate will contain no impurities of any importance whatever, the method being therefore not dependent upon any compensation between the desired precipitate and its impurities. One must be careful that the reagents contain no sulphuric acid. $CaCl_2$ particularly often contains this acid. When the reagents are not quite pure a blank test is done along with the sample."—*Zeitschrift fur Analytische Chemie*, 1910, p. 393. (D. G.)

GAS ANALYSIS BY CONDENSATION.—"Two gases whose melting points are widely apart can be quantitatively separated by bringing the one with the higher liquifying point into the liquid state by means of a suitable cooling medium. The apparatus required consists mainly of a measuring tube and gas pipette between which is interposed a tube with a relatively small inlet, and cooled by liquid air, liquid oxygen, melting ether, solid carbon dioxide or some such cooling medium. The dried and measured gas mixture is by the usual means made to pass backwards and forwards through the cooling tube between the pipette and the measuring tube. After measuring the non-condensable portion of the gas mixture, it is removed from the apparatus. The cooling medium is then taken away and the condensed portion re-gasified and also measured."—*Zeitschrift fur Angewandte Chemie*, 19th Aug., 1910, p. 1580. (T. D.)

ESTIMATION OF SULPHUR IN PYRITES, SLAGS AND SULPHATES.—"A method proposed by A. Zehemayr, in the *Zeitsch. angew. Chem.*, is:—The substance under examination (pyrites, slags, sulphates, etc.), is heated with metallic iron, the sulphur present is thus converted into ferrous sulphide, sulphates also being reduced to sulphide, and the ferrous sulphide is then decomposed with hydrochloric acid, the liberated hydrogen sulphide being collected in potassium hydroxide solution. The details of the method are as follows:—0·5 grm. of the finely powdered sample are placed in a hard-glass tube, closed at one end, together with about 1 grm. of ferrum redactum, and after being mixed the contents of the tube are heated to dull redness for a short time. While still hot, but not glowing, the mixture is shaken into a flask containing about 25 cc. of water, and the flask is closed with a stopper carrying a safety funnel and a delivery tube. The latter is provided with a bulb filled with glass beads to prevent any solution in the flask splashing over into the receiver; above the bulb the delivery tube is bent downwards, and is surrounded by a condenser, the lower end of the tube dipping into the potassium hydroxide solution in the receiver. Concentrated hydrochloric acid is added through the safety funnel to the contents of the flask. and when the evolution of gas has nearly ceased heat is applied, the solution being finally boiled until all the hydrogen sulphide has been driven over into the receiver. Towards the end of the distillation a drop of the distillate may be tested with lead acetate to ascertain whether all the hydrogen sulphide has been expelled. The alkaline distillate is then poured into a known quantity of $N/10$ iodine solution, acidified with acetic acid, and the excess of the iodine is titrated back with thiosulphate solution. One cc. of $N/10$ iodine solution is equivalent to 0·0016 grm. of sulphur. The ferrum redactum used must be free from sulphur. Results of analysis of pyrites, slags and ferrous sulphate are given, the figures obtained by the above described method agreeing closely with those yielded by Lunge's method.—*Zeit für Angewandte Chemie. Chemical Trade Journal*, September 17, 1910, p. 262. (T. D.)

HYDROFERRO- AND FERRICYANIC ACID, DETERMINATION OF.—"The titration of hydroferrocyanic acid with permanganate by De Haen's method is found to give accurate results when carried out as follows:—20 cc. of sulphuric acid (1 mol. of acid to 4 mols. of water) are added to 150—200 cc. of the solution (containing about 1 grm. of ferrocyanide), and the mixture is titrated with $N/20$ permanganate, until the colour changes from yellowish-green to yellowish-red. Hydroferricyanic acid also may be determined by permanganate, after reduction with ferrous sulphate; but this acid is preferably estimated by Mohr's method, which is accurate when carried out in the following manner:—0·7 grm. of ferricyanide is dissolved in about 50 cc. of water, and to the neutral solution are added 3 grm. of potassium iodide and 1·5 grm. of zinc sulphate (free from iron); the solution is then shaken and titrated with $N/20$ thiosulphate. The determination of hydroferrocyanic acid by oxidation with iodine and titration with thiosulphate, in the presence of alkali bicarbonate (Rupp and Schiedt; this J., 1902, 1099), is found to be inaccurate. Small quantities of ferricyanide were detected in all the samples of potassium ferrocyanide examined."—E. MÜLLER AND O. DIEFENTHALER, *Z. anorg. Chem.*, 1910, *67*, 418—436. *Journal of the Society of Chemical Industry*, August 31, 1910, p. 1007. (A. W.)

DETERMINATION OF FERRO- AND FERRICYANIDE.—"*Potassium ferricyanide.*—"The salt is dissolved in 500—750 cc. of water, and 10—15 cc. of concentrated hydrochloric acid (sp. gr., 1·19), 10—15 grm. of potassium chloride, 10—20 cc. of a 10% solution of potassium iodide, and 10 cc. of zinc sulphate solution (approximately molecular) are added. The turbid mixture is allowed to stand for 3 minutes in a stoppered flask, after which some starch solution is introduced, and $N/10$ thiosulphate is added, until nearly all the iodine has disappeared; the titration is then finished after waiting again for 3 minutes. 1 cc. of $N/10$ thiosulphate = 32·921 mgrms. of potassium ferricyanide. *Potassium ferrocyanide.*—To the aqueous solution (600—800 cc.) 10—15 cc. of concentrated hydrochloric acid and 40—60 cc. of sodium acetate solution (250 grm. of crystallised salt in 675 cc.) are added, and then $N/10$ permanganate, with shaking, until the solution is quite clear and distinctly red; then 10 cc. of 10% potassium iodide solution are added, and, after 3 minutes the separated iodine is titrated with $N/10$ thiosulphate (best standardised against permanganate solution). 1 cc. of $N/10$ permanganate = 42·236 mgrms. of crystallised potassium ferrocyanide. An approximately standard solution of permanganate may also be used, and the ferricyanide produced determined as above, thus giving the ferrocyanide indirectly. To determine *potassium ferrocyanide and ferricyanide*, the above methods are combined, the solution (700—800 cc.) being mixed with 15 cc. of concentrated hydrochloric acid and 60 cc. of the above sodium acetate solution and the ferrocyanide determined directly, and then a further 10 cc. of hydrochloric acid, 10 cc. of 10% potassium iodide, and 10 cc. of (molecular) zinc solution being added and the total ferricyanide determined."—W. MECKLENBURG, *Z. anorg. Chem.*, 1910, *67*, 322—338. *Journal of the Society of Chemical Industry*, August 15, 1910, p. 946. (A. W.)

METALLURGY.

AUTOMATIC SAMPLING.—"'Lamb's Tailings Sampler' is one that may be recommended. This sampling machine has been designed to obtain an accurate sample of the tailings discharged from the mill—something that would be reliable in operation, automatic, simple in construction, and accurate in results—the main idea being to introduce a device to obviate the hand sampling by mill men or foremen, who might have reasons for favouring results in sampling, and also to take a sample regularly at such stated intervals as may be desired, which latter is seldom attained in mill work, for many well known reasons. With this sampler placed below the final discharge of tailings sluice, and enclosed in a small building under lock and key, it can be made accessible to only one person, and accurate samples are bound to follow. As the whole discharge of tailings from the mill does not pass through the sampler, there is not the wear on it that there is on machines or devices that have to carry the whole stream. This sampler has been in operation in one of the largest and most complete concentrating mills in the West, and, during a period of twelve months, many comparative tests with hand sampling have been carefully made, and the results have proven that this device is reliable and accurate in its work. Its simplicity of construction, small liability of getting out of repair, together with its low cost, I beleive, makes it the most desirable appliance for this work that has been placed on the market.

Adaptability.—This sampler can be applied to any mill regardless of the process employed for treating the ore, with the single proviso that the tailings from the mill are discharged by means of a sluice box, to which the sampler could be attached.

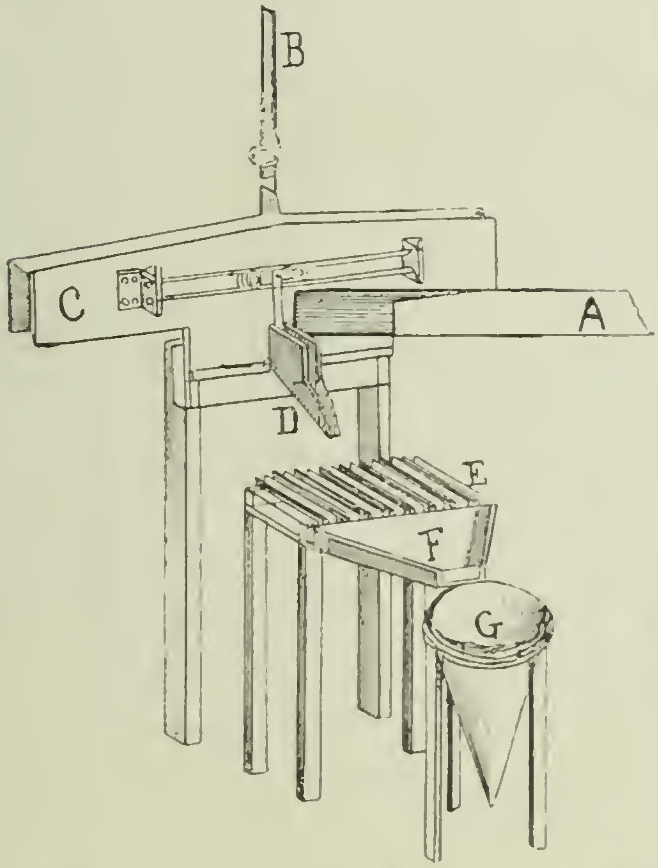
Details of Construction.—By referring to sketched drawing, in conjunction with the following description, the details of construction will be more readily understood:—A represents a 'tailings sluice' running out of the mill; B, 'water pipe' for filling ballance box; C, balance (or tilting) water box. This is a double compartment box, arranged with a partition in the centre, so that when one side has filled and is discharging, the other side is being filled.

Attached to the front of the balance box is the track for carrying the sampling spout (D).

(D) *Sampling Device.*—This consists of a narrow spout or trough, suspended from a hanger or carriage, having two grooved pulleys which run on the track attached to the balance box. This spout and carriage works by gravity, and, as the balance box discharges at either end, the spout travels across the face of the stream. This gives an accurate sample of the tailings, as it takes an equal amount right across the sluice, and also includes a proper proportion of the slimes, as well as the fine and coarse sands.

(E) *Divider or Reducer.*—This consists of a number of small troughs, made preferably with a V bottom and vertical sides, the sides being high enough to prevent the slimes or lighter material from slopping out. These troughs rest loosely in a notched rack, and are set at a pitch that will give a free discharge on the apron below. By varying the number of troughs employed the volume of the sample can be regulated as desired.

(F) Apron on to which the dividers (E) discharge, and by means of which the sample is conveyed to the filter sack.



Filter Sack.—This is simply a canvas duck sack, which permits the surplus water to be drained off. In place of the filter sack, a series of overflow tanks may be used.

The water from the feed pipe discharges into the upper half of the balance or tip box. When the weight of water accumulated in this compartment equals or slightly exceeds the weight of the travelling trough, the balance box tips and the travelling trough moves along the inclined rods, sweeping across the stream of tailings at right angles to its course, cutting out an equal proportion of the full width of the stream, and taking an actual proportionate amount of coarse, fine and slimes tailings. The travelling trough moves over and across a series of V troughs, with open spaces between, discharging the sample cut into these troughs, the excess falling through the open spaces between them. The V troughs discharge on to an apron, which in turn discharges into the final receptacle, the filter sack. Beneath the filter sack is placed a box for receiving the filtered water from sampler. This is a guard against imperfect filtering or leakage, as well as means for testing such waters for any values that may be soluble. With this sampler the feed water to balance or tip box can be regulated by a valve, so that a sample can be taken at any interval of time desired, or, for a twenty four hours' run (if only a daily sample is to be taken), can be regulated by increasing or decreasing the V troughs, into which the travelling trough discharges its contents. If the machine is kept well oiled it will do its work faithfully."—*Australian Mining and Engineering Review,* Oct. 5, p. 23—24. (C. B. S.)

Charge and Series Systems of Cyaniding Slime.—"The various methods of treating pulp in air-agitation tanks offer problems for experiment and study which are fascinating as well as practical. The usual method heretofore has been to fill each tank in turn, agitate the mixture the required period and then discharge the treated pulp into a storage tank, from which it is drawn to the filter as required.

A later method is to run the pulp from the tube mill, classifier, thickener, or settler into the first of a set of tanks, and thence continuously, or in series through the remaining tanks, finally drawing off the pulp to a continuous filter, or to a pulp storage tank if the filter is of the intermittent type. It is this method which I propose to discuss in a preliminary way, with the idea of going exhaustively into the examination of the results of a series of experiments in a later paper.

It may be of interest to note that the design of a cyanide plant has recently been completed in which the estimated cost of the agitation tanks, storage tanks, and pumps was 30% higher for the charge agitation arrangement then for the series plan.

The first question in studying the series method is, 'What is the actual period of treatment undergone by the pulp?' One first supposes that by chance some of the pulp may get through with little or no treatment, while other portions may stay in the tanks a needlessly long period. The former is not probable or even possible, and the latter is, practically, true to only a limited extent. The greater the number of tanks the greater is the percentage of chance that all the pulp will be treated equally. Without going into the mathematical analysis of the problem its important terms may be clearly grasped by thinking of, or 'visualising,' the conditions when, say, 50 tons of dry slime is treated in 24 hours. The thickened pulp representing this tonnage is a sluggish, 2 in. stream flowing 25 in. per second. Taking the flow of this stream for any given period, say 5 min., 16 cub. ft. will flow into the first of a set of agitation tanks. If the tanks have a working capacity of 1,600 cub. ft., one such tank will contain 17 tons (dry weight) of slime in a 2-to-1* pulp, and three tanks will be required to agitate 50 tons for 24 hours. Assuming an instantaneous mixture of the stream of incoming pulp with that already in the tank (an impossible condition which will be discussed later), then 1% of 16 cub. ft. will be discharged with no treatment. With two tanks the proportion which would be discharged with no treatment is reduced to 0.01%. Of course, other portions would be discharged with varying and increasing periods of treatment.

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* 2 to 1.—Ed. C.

The rate of flow of the pulp from the time it is thrown out of the top of the central tube until it reaches the bottom of the tank varies with the amount of air used and the shape and the size of the tank. The period is from 0·5 to 2 hours. If the incoming stream enters at the side of the tank opposite the outlet, and giving due consideration to this slow movement of the charge, it is a physical impossibility for any particle of the pulp to go through this series of three tanks in less than 1½ hours. In the mathematical analysis this minimum period gives the series method a big handicap, and one which increases directly with the number of tanks used. In fact, this low circulation velocity makes the operation resemble the treatment of pulp in a horizontal tube of the same diameter as the agitation tanks and of a length equal to the combined lengths, having the agitation at right angles instead of parallel to its length.

If any comparisons have been made on a practical scale to show the difference in extraction by the two methods of using the air-agitation tanks, the results, so far as I know, have not been published. Such a test would be of great interest, and the managers of several plants are now in a position to make it. In tests on a small scale, with tanks of a capacity of 100 lb. of dry slime in a 2-to-1 pulp, several interesting things have been noted. With pulp containing coarse sand (and even with fine sand in a thin pulp), care must be exercised in drawing the pulp from one tank to the next in the series, otherwise either the sand or the slime will accumulate. If the drawing-off pipe is merely flanged to the tank, the outflowing pulp will contain a greater proportion of slime to sand than is contained in the average of the pulp, since during agitation by this method the sand has a tendency to sink through the pulp as soon as discharged through the centre air lift, rather than to work out to the sides of the tank with the flocculent slime. This difference in pulp flow is greater with a thin mixture and less with a thick one, and can be utilized practically in a very simple manner if it is desired to agitate the sand longer than the slime. A concentration of sand in the tank will be attained by drawing the slime from a quiet point in the circumference of the surface of the pulp. To do this, a discharge pipe, pivoted at the side of the tank, is arranged to swing in a horizontal plane and to be placed and held at any desired point in its arc between the central pipe and the side of the tank. After the desired initial concentration is attained, this discharge pipe is then placed so as to draw off a mixture of the same proportions of sand and slime as that being fed to the tank, which will maintain the charge in the tank at the higher ratio of sand to slime. Thus the slime will go through faster and the sand slower than the average pulp flow. To illustrate this point, imagine a tank containing 100 tons (dry weight) of pulp consisting of 3 of sand to 1 of slime. Admit a stream of pulp carrying 100 tons in 24 hours, which consists of half sand and half slime, drawing off an exactly similar stream. The slime which was originally in the tank (25 tons) will be replaced twice in 24 hours by the 50 incoming tons, while the original sand (75 tons) will be replaced only two-thirds time. In other words, the slime will receive a 12-hour treatment and the sand will receive a 36. I consider this one of the important advantages of the 'series' system. The regulation of the ratio of sand concentration in the tanks, within the practical limits to be determined in each case, will be controlled by taking a pulp sample of a certain size, washing it through a 200-mesh screen, and setting the discharge pipe as many inches from the rim of the tank as there are ounces of sand remaining on the screen. The proper weight of the sample to give best results can be determined experimentally.

A point of great advantage possessed by the series method in the design of many plants is the saving in difference of level required in the site. The agitation tanks, if used in this manner, need not require more than from 2 to 4 ft. difference in level between the inlet and the outlet, depending to some extent upon the number and size of tanks and the distance apart, but also and mainly upon the margin of capacity required to provide for fluctuations in pulp-feed and discharge. This margin depends largely upon the type of filter used. Ordinarily, the pulp will be drawn from near the top of the last tank, while in the 'charge' system the outlet is at the bottom of the tank. Fig. 1 shows this difference. In the series system no storage-tank is needed below the last agitator tank if a continuous filter, such as

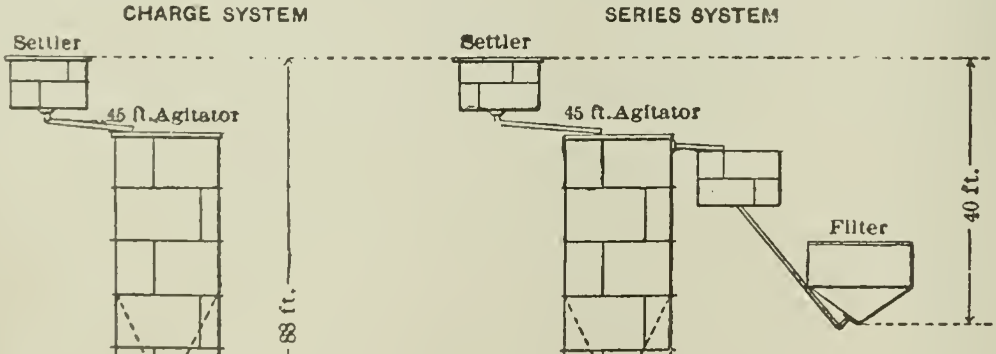


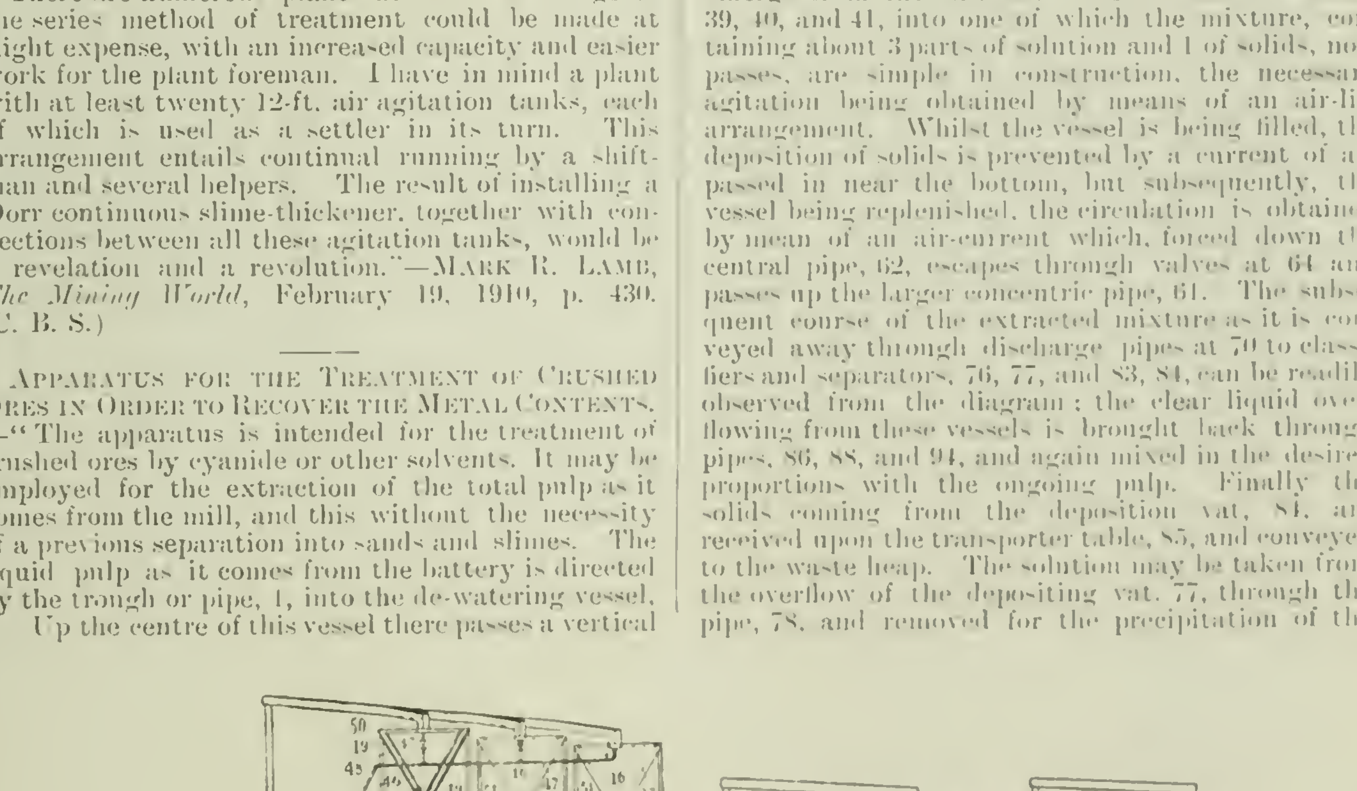
Fig. 1. Charge System and Series System Compared.

Oliver or the Ridgway, is used, only a small tank of a capacity to supply pulp to equal the displacement of a loaded basket of leaves if a Moore filter is used, and a slightly larger tank (sufficient to fill the filter box with the unloaded frames in place) with a Butters filter. Compare this arrangement with the charge system, in which the storage tank, even with continuous filters, must have at least a capacity equal to that of an agitator, and with either the Moore or the Butters filter must have a slightly greater capacity, as is explained above.

The question, what is the least number of tanks which can safely be used in this manner, can only be answered by experiments. Even with the charge system, the occasion will rarely arise in which it will be convenient to use less than two tanks for agitation, and in such a case three smaller tanks with equal average treatment capacity will be so much smaller that their combined cost will be little, if any, more than the cost of the two larger tanks. In a plant having air-agitation operated on the charge system, it is rare that the site will permit all transfers of pulp—from settler through agitators and storage tank to filter—by gravity. Usually the saving of about 45 ft., as shown in Fig. 1, will practically amount to the saving at least of the cost of a 4 or 6in. steel-lined centrifugal pump, with its requirement of from 10 to 20 h.p.

There are numerous plants at which a change to the series method of treatment could be made at slight expense, with an increased capacity and easier work for the plant foreman. I have in mind a plant with at least twenty 12-ft. air agitation tanks, each of which is used as a settler in its turn. This arrangement entails continual running by a shiftman and several helpers. The result of installing a Dorr continuous slime-thickener, together with connections between all these agitation tanks, would be a revolution and a revolution."—MARK R. LAMB, *The Mining World*, February 19, 1910, p. 430. (C. B. S.)

APPARATUS FOR THE TREATMENT OF CRUSHED ORES IN ORDER TO RECOVER THE METAL CONTENTS.—"The apparatus is intended for the treatment of crushed ores by cyanide or other solvents. It may be employed for the extraction of the total pulp as it comes from the mill, and this without the necessity of a previous separation into sands and slimes. The liquid pulp as it comes from the battery is directed by the trough or pipe, 1, into the de-watering vessel, 2. Up the centre of this vessel there passes a vertical shaft, 5, carrying at the top a small cone, 4. Larger inverted cones, 6, 7, 8, with vertical cylindrical prolongations, 11, 12 and 13, are arranged underneath. The liquid pulp being distributed by the pipe, 1, into the space comprised between the cone, 4, and the internal conical ring, 10, the liquid runs downwards and under the lower edge of the cylindrical prolongation, 11, and thence passes in and out between the cones until it rises in the vat and overflows as clear water into the annular discharge, 3. During the circulation the solids are deposited upon the small truncated surfaces, 9, between the cones, 6 and 7, and upon the outer truncated surfaces, 20, whence they afterwards fall to the bottom of the vessel, the operation being assisted by the revolving radial arms, 18, carried by the vertical shaft, 5. The partially drained solids, falling to the bottom of the vat, are thence removed by means of the conveyors, 24 and 29, and the pumps, 36, to one or other of the classifiers, 46, 46, 46, being mixed on the way with a suitable quantity of the extracting solution conveyed from a reservoir by the pipe, 90. The classifiers, in which any desired proportion of the dissolving solution is separated, comprise internal and external cones, 47 and 48, an intermediate conical passage, 49, and an annular overflow, 50, by means of which a certain proportion of solution is returned through the pipe, 51, and again mixed with the solids as they emerge from the drainer. The extraction vessels, 39, 40, and 41, into one of which the mixture, containing about 3 parts of solution and 1 of solids, now passes, are simple in construction, the necessary agitation being obtained by means of an air-lift arrangement. Whilst the vessel is being filled, the deposition of solids is prevented by a current of air passed in near the bottom, but subsequently, the vessel being replenished, the circulation is obtained by mean of an air-current which, forced down the central pipe, 62, escapes through valves at 64 and passes up the larger concentric pipe, 61. The subsequent course of the extracted mixture as it is conveyed away through discharge pipes at 70 to classifiers and separators, 76, 77, and 83, 84, can be readily observed from the diagram: the clear liquid overflowing from these vessels is brought back through pipes, 86, 88, and 94, and again mixed in the desired proportions with the ongoing pulp. Finally the solids coming from the deposition vat, 84, are received upon the transporter table, 85, and conveyed to the waste heap. The solution may be taken from the overflow of the depositing vat, 77, through the pipe, 78, and removed for the precipitation of the



metallic contents." — A. J. ARBUCKLE, *Fr. Pat.* 413,417, March 9, 1910.—*Journal of the Society of Chemical Industry*, Sept. 30, 1910, p. 1115. (J.A.W.)

EFFECT ON THE SOLUBILITY OF GOLD WHEN ORE IS CRUSHED BETWEEN IRON SURFACES.—"The author describes a series of comparative laboratory experiments on the chlorination and cyaniding of gold ores, which had been pulverised respectively in a grinding mill, on a bucking board, and in an improvised wooden tube mill with silicious pebbles. The assay result showed a higher gold extraction in the case of the tube mill, which the author considers is probably due to the fact that when the ore is crushed between iron surfaces the particles of gold become coated with a protective film of metallic iron"—J. M. TIPPET, *Met. and Chem. Eng.*, 1910, 8, 519-520. — *Journal of the Society of Chemical Industry*, Oct. 15, 1910, p. 1161. (J. A. W.)

LINING FOR ORE CHUTES.—"The majority of mining companies usually line the bins and ore shoots with sheet steel. It is seldom that the steel sheets are worn uniformly, and as soon as a hole occurs it is necessary to renew the entire sheet, which often means a waste of material and extra expense. At Mineville, New York, Witherbee, Sherman and Co. found it more economical to use bar steel. At present they use $\frac{3}{4} \times 6$ in. bars. By using bars it is less trouble to do repair work, and it is necessary to remove only the worn out part and put in a new piece. All the steel may thus be used. Even short bars with one end worn thin may be used. Railroad rails of heavy weight are also good for lining the front of ore bins, at a point directly opposite the place where the cars dump. They are occasionally suspended by the upper end, and are free to move at the lower end. They form an excellent buffer for heavy ore as it comes from the mine car or skip."—*Australian Mining and Engineering Review*, Jan. 5, 1910. (C. B. S.)

MINING.

DEVICES FOR PLOTTING CO-ORDINATES, DRIVES AND STOPES.—"1. The device for plotting co-ordinates consists of a 45° triangle graduated (on the two edges at right angles to each other) to any required scale. Fig. 1 represents a triangle graduated

FIG. 1.

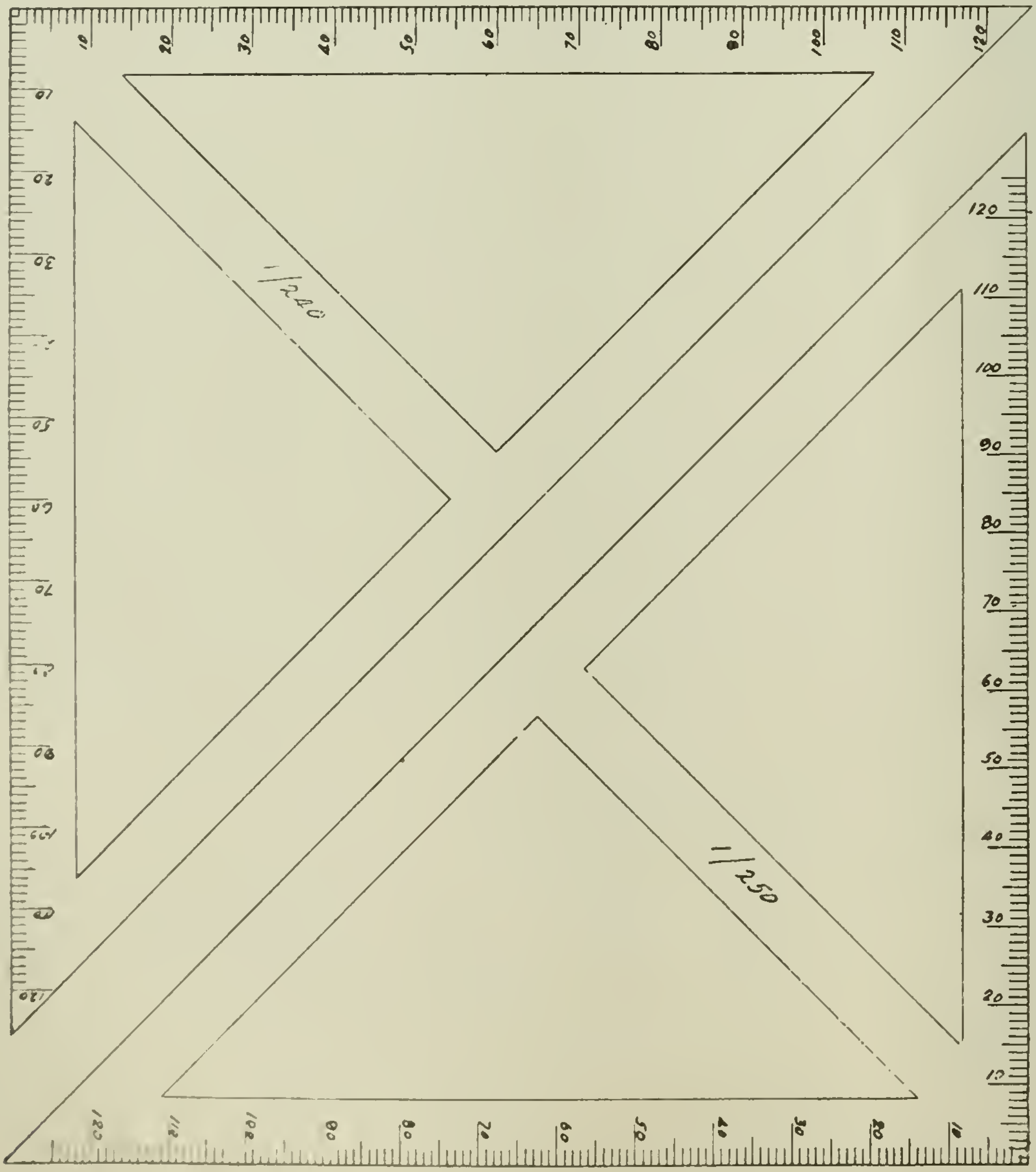


FIG. 2.

so that 1 ft. = 240 ft. : Fig. 2, 1 ft. = 250 ft. : Fig. 3, 1 ft. = 500 ft., and Fig. 4, 1 ft. = 2,500 ft. The graduated edges are about six inches in length.

FIG. 3.

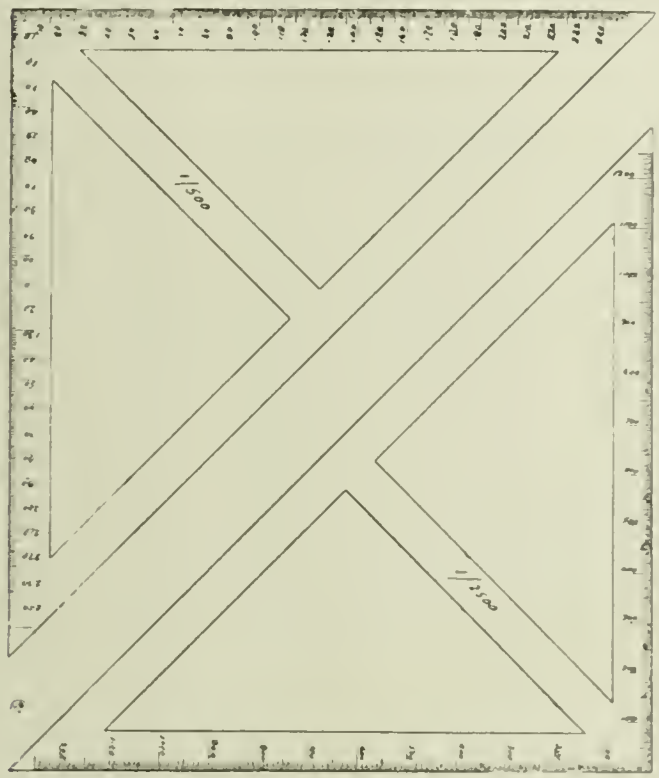


FIG. 4.

2. By a proper assumption of the distance apart of the co-ordinate lines their representation on the plan will be contained within the graduated edges of the triangle. For example, using the 1/240 triangle the co-ordinate lines must be 100 ft. apart and hence represented on the plan 4 4/5 in. apart : 1/250 × 100 ft. - or 4 4/5 in. : 1/500 × 200 ft. - or 4 4/5 in. : 1/2,500 × 1,000 ft. - or 4 4/5 in.

3. In the following discussion the axes of co-ordinates are assumed to be as represented as in Fig. 5.

4. Suppose the plotting is required on the horizontal plan (scale 1 ft. = 500 ft.) of the stations whose co ordinates are

	X.	Y.	Z.
309	6953·30	4995·06	1283·24
310	6978·09	4982·28	1280·64
311	6997·73	4975·30	1281·64
312	7059·39	4959·85	1285·29
310a	6976·75	4969·72	1251·65
310b	6974·15	4961·79	1241·71
310c	6975·36	4955·56	1235·55
210	6963·14	4897·66	1185·99
211	6975·96	4892·15	1185·45
212	7068·83	4866·98	1186·47

5. To plot station 309, place graduated triangle 1/500 on plan (see Fig. 6) with one graduated edge coinciding with the line Y = 4,800, then move so that the 153·30 (6953·30 - 6800·00) division coincides with the line X = 6,800, then at the 195·06 (4995·06 - 4800·00) division on the other edge, prick a point : this will represent on the plan the position of station 309. The other stations are plotted similarly (see Fig. 7).

6. As a check place the graduated triangle as in Fig. 8 so that the 195·06 division on one edge coincides with the line Y = 4,800 and prick a point at the 153·30 division on the other edge. The two points should coincide.

7. On any vertical plane or section the Z ordinate may be plotted similarly.

8. The graduated triangles above described may be employed in various ways. For example, suppose it is required to transfer the lower drive and part of raise represented in Fig. 9 (scale 1/500) to a plan (scale 1/240) (see Fig. 10). In Fig. 9 the 1/500 triangle is placed as indicated ; in this instance with the 160·00 division on one graduated edge coinciding with the line X = 6,800 : the Y ordinates (4,800 + 189·50) and (4,800 + 195·50) of the points of intersection of the contours of the drive with the graduated edge of the triangle are read off : in Fig. 10 the 1/240 triangle is placed as indicated with the 60·00 (160·00 - 100·00) division on one edge coinciding with the line X = 6,900, then points are pricked at divisions 89·50 (4,800 + 189·50 - 4,900) and 95·50 (4,800 + 195·50

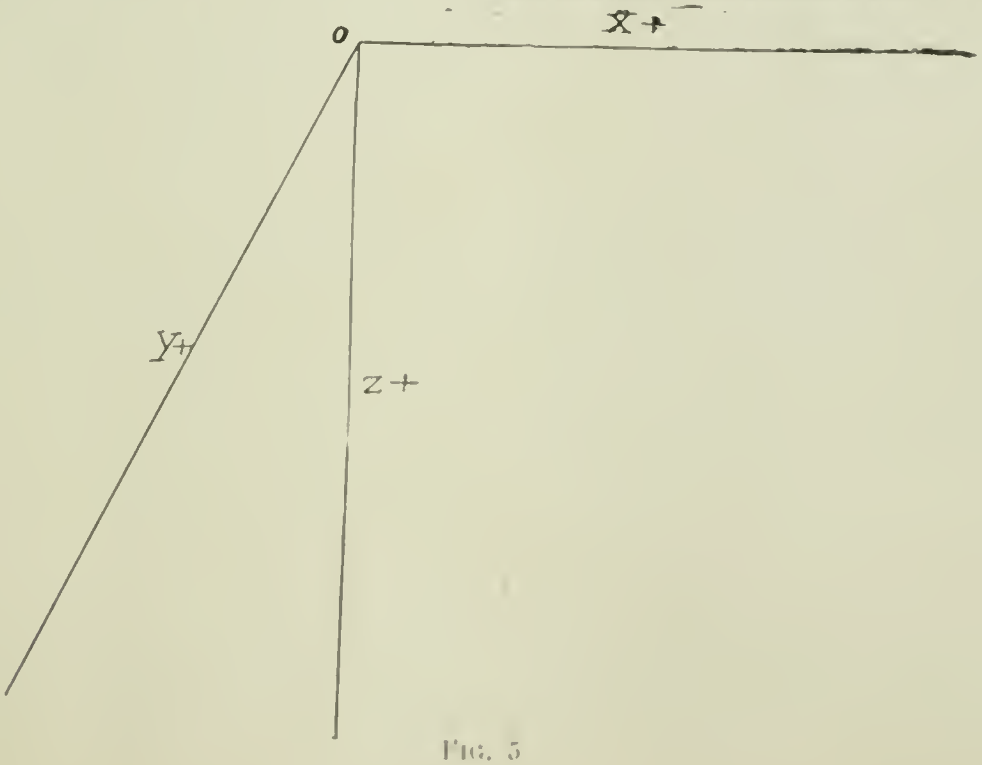


FIG. 5

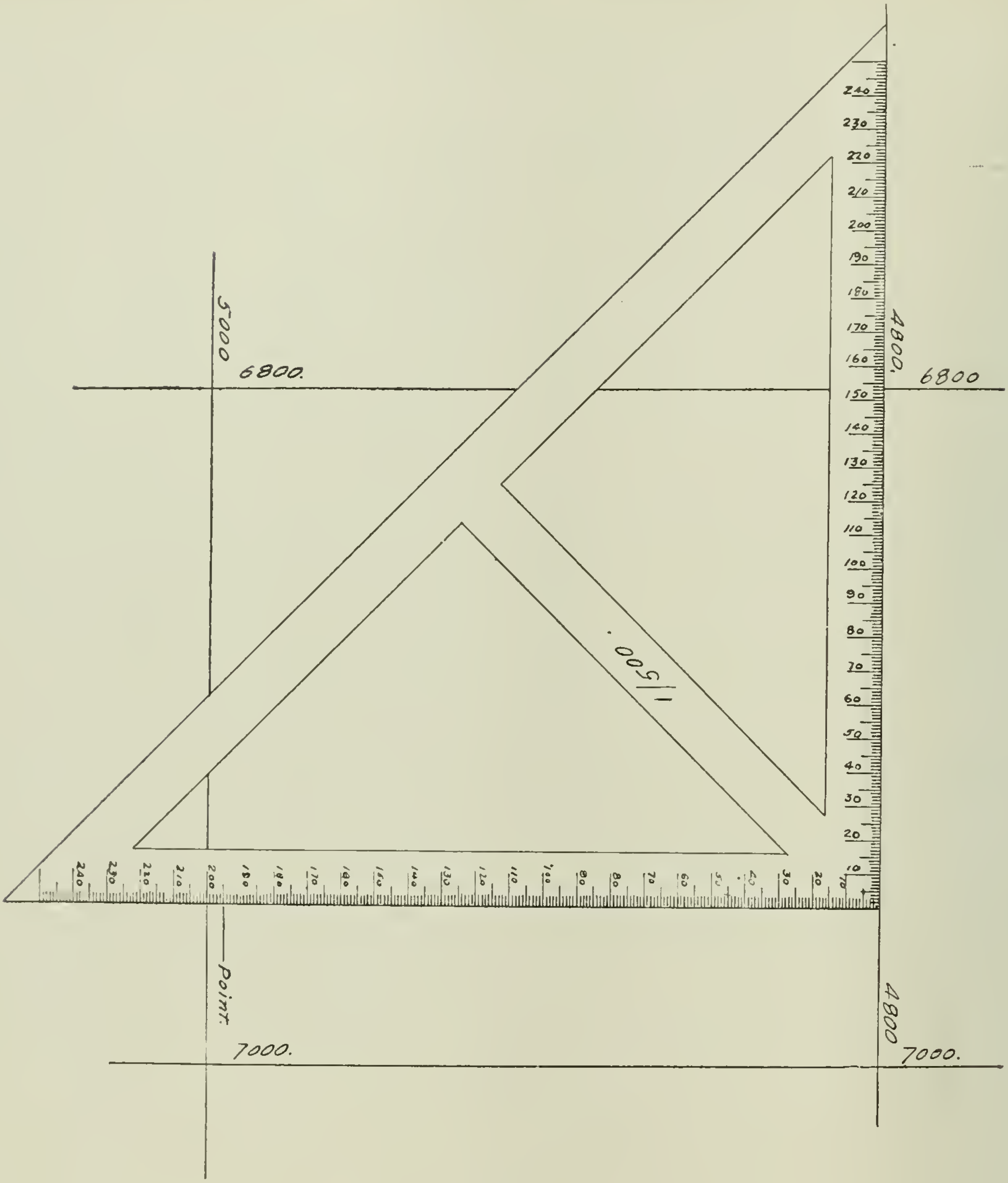


FIG. 6.

– 4,900). The triangles are moved convenient distances of 5 or 10 ft. and the Y ordinates read off and transferred as before. The points thus indicated are joined showing a contour similar to that of Fig. 9.

9. Again if a projection is required on the vertical plane through stations 210, 310, of the raise connecting 210 and 310 (see Fig. 7) (scale 1/240), then assuming in this plane that 210 is at 0 ft. horizontal distance, the other stations are at horizontal distances (measured off from Fig. 7)

210	0·00 ft.	
310c	59·00	(projection on plane 210, 310)
310b	65·00	,, ,,
310a	73·50	,, ,,
310	86·00	,, ,,

and (see Fig. 11) for 310 the graduated triangle 1/240 is placed as indicated, the 86·00 division on one edge coinciding with the line Z – 1,200, a point pricked at division 80·64 (1280·64 – 1200·00) (obtained from station co-ordinates listed above), on other edge. The other stations are plotted similarly. From the notes taken when the transit was set under the several stations the contour is sketched in and the desired section is as shown in Fig. 12.

10. These triangles save considerable time in plotting, are accurate and do not soil the plans, as there are no erasures to be made.

11. The device for plotting drives is represented in Fig. 13. It consists of a protractor about 8 in. in diameter. (A) graduated to 20 minutes and numbered to correspond to those instruments whose numbering of graduations on the horizontal circle is from right to left. (B) is a slot in which the slide (C) can be freely moved. (D) and (E) are scales graduated as desired; in this instance (D) is 1/500 and (E) 1/240. (F) is a hole in slide (C) with centre on line through centre (O) of protractor and 0°. Line (G) passes through (F) and serves to indicate the distance along (D) and (E) of the centre of hole (F) from (O). (H) is an upright, shown in section at (I) through which

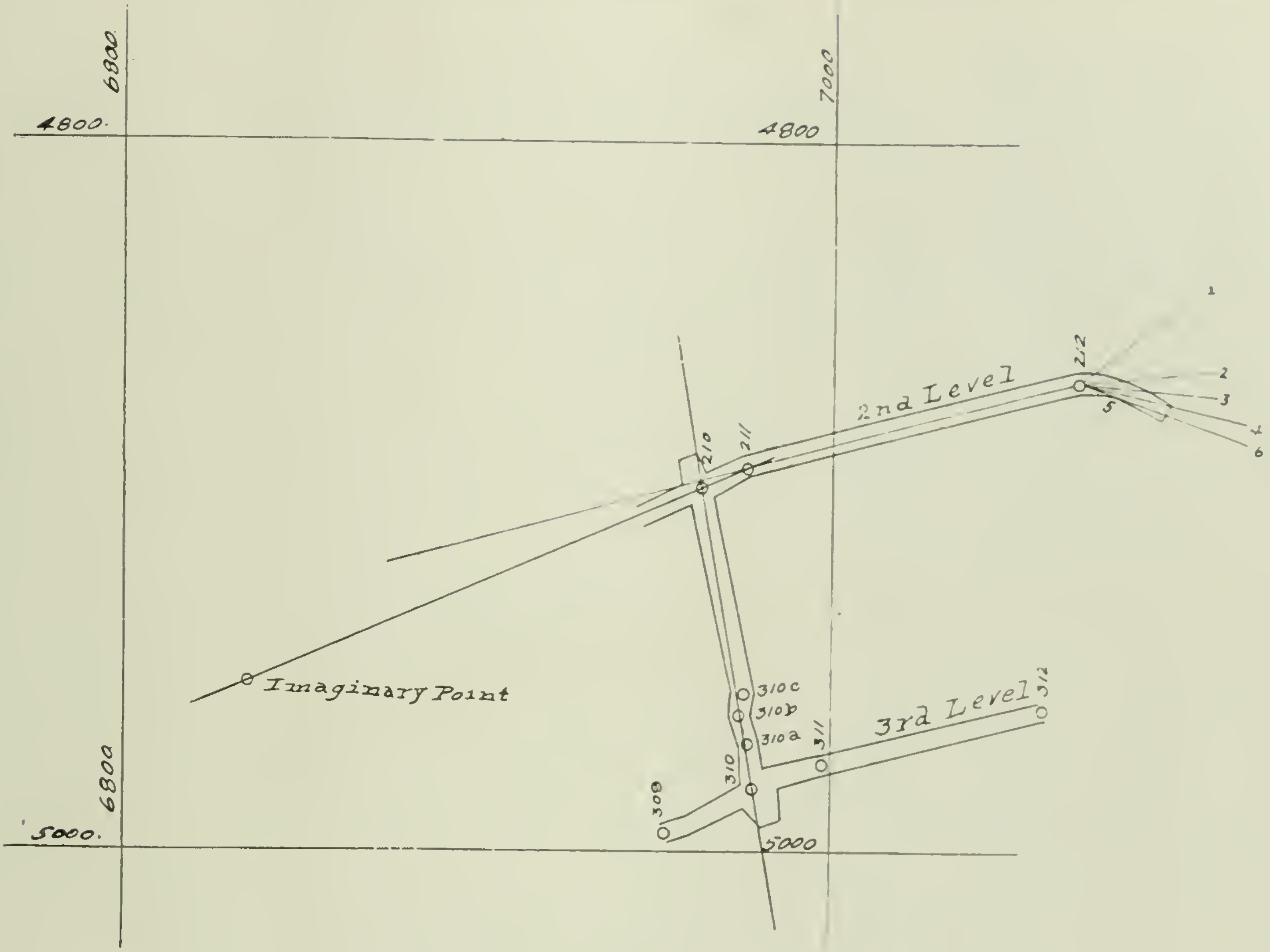


Fig. 7.

passes vertically a fine needle. This needle is exactly over centre of protractor.

12. The application is as follows, supposing the notes for the drive survey are

	Inst. under 212. Horizontal angle.	Backsight 211. Slope distance.	Foresight Drive. Vertical angle.
1.	157° 20′	4·75	90
2.	188° 40′	12·00	90
3.	198° 40′	18·00	90
4.	206° 40′	26·00	90°
5.	213° 30′	10·00	90
6.	213° 30′	25·00	90

Offset right 2 ft.
,, left 3 ft.

A line is drawn through 212 and 211, extending beyond 211; the protractor is then placed on the plan, needle point pressed down through 212, and and then revolved until 157° 20′ is indicated, the slide moved to indicate 4·75 ft. in this instance on scale 1/500. A mark is then made through hole in slide. This mark represents a point in the contour of drive. Similarly for the other points. When point 6 is plotted the offsets indicated are made and the several points joined, thus showing the outline of the drive (see Fig. 7).

13. If when using this protractor the two stations are very near together, to secure accuracy, the best plan is to find the co-ordinates of an imaginary point in the extension of the line through the two stations before drawing the reference line through the two stations.

14. For example suppose the transit were set under 211, backsight 210, then proceed as follows:

	X.	Y.	
211	6975·96	4892·15	
210	6963·14	4897·66	
Diff.	12·82	−5·51	Multiply by 10
	10	10	
	128·20	−55·10	Subtract from
210	6963·14	4897·66	
Co-ordinates of imaginary point	6834·94	4952·76	

Plot this point (see Fig. 7) and then draw a line through it and 210 or 211. To eliminate erasure as much as possible the line should be drawn for a short distance only, near edge of protractor.

15. For distances less than half the length of the slide, that is less than two divisions of 1/500 scale or 2 ft., after the horizontal angle has been indicated, the needle may be removed and the slide moved to the required distance.

16. Perhaps a clearer idea regarding the use of the device for plotting stopes (which I prefer to designate the 'plotting protractor') will be engendered by describing briefly the method of surveying and plotting stopes to which it is applicable.

17. The transit or theodolite is set up under a station, a sight taken to another station—the plates being clamped at 0°—the lower plate is then clamped

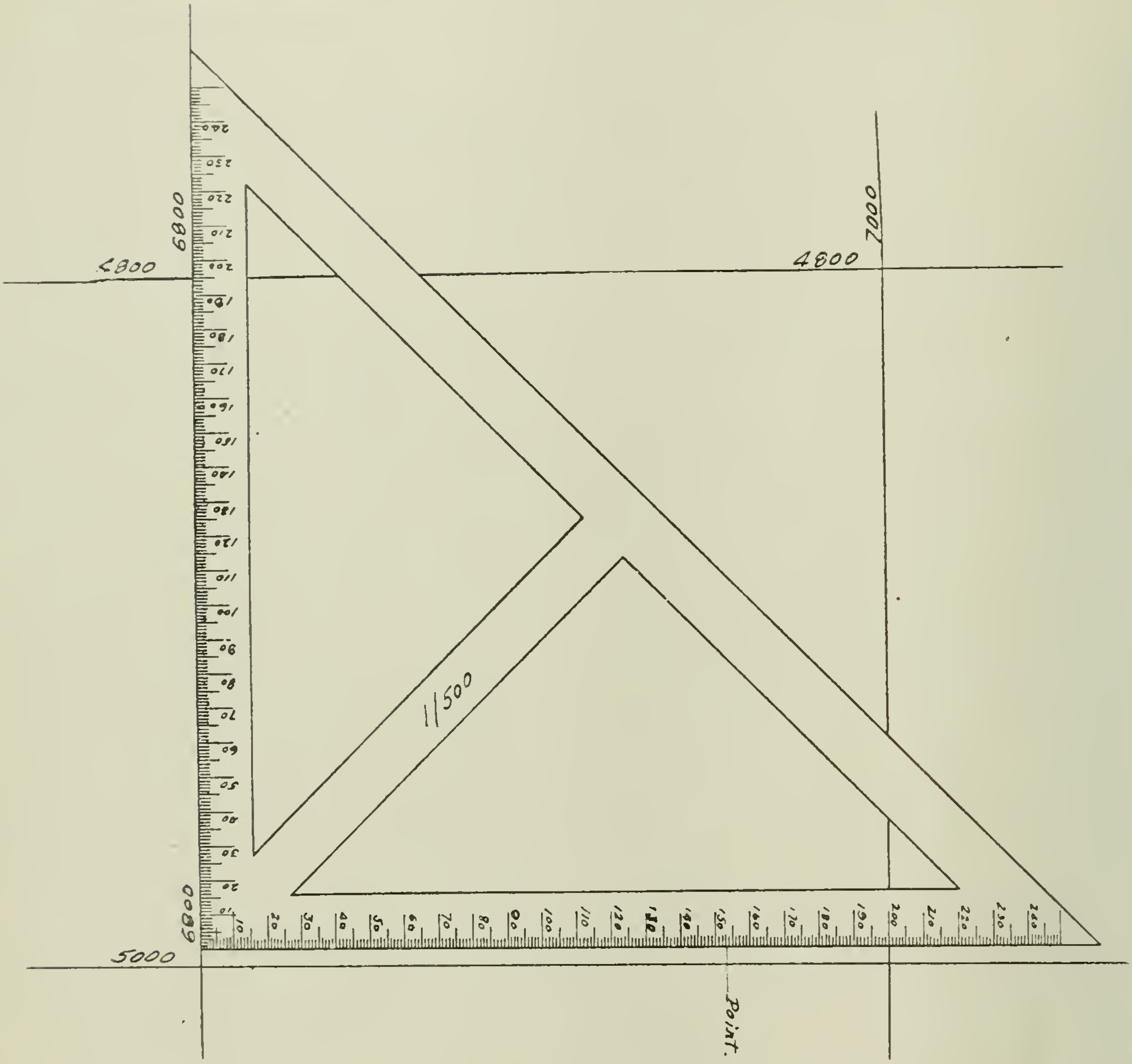


FIG. 8.

and the upper unclamped and sights taken to the most prominent points of projection and recession of the face of the stope; these points being chosen about halfway between the hanging-wall and footwall of the stope, or more preferably on the line of trend of the vein or reef being stoped. The horizontal angle, the slope distance, and the vertical angle for each point is recorded. By means of a table of latitudes and departures, or natural functions, or by logarithms, the slope distances are reduced to the horizontal; then on the plan, usually scale 1 ft. = 240 ft., the horizontal angles are laid off and then the horizontal distance for each angle. The points thus determined are joined, thus forming the horizontal projection of the contour corresponding to that of the face of the stope. This contour with that of the preceding survey, or in conjunction with the contour of the upper drive or that of the lower drive, or of both, determines the horizontal projection of the area stoped. This area multiplied by the secant of the angle of the dip is the slope area corresponding to the area stoped. The dip is usually ascertained by sighting to a point on a rod at the same vertical distance above the footwall as the axis of the instrument, and held at a point in the line of dip, as near as can be judged, and some distance from transit. The vertical angle is read giving the angle of dip. Or the dip may be taken off the vertical section of the raise or winze from which the stope is being developed.

18. The method of surveying stopes above described in conjunction with the 'plotting protractor' and the methods of determining the dip to be described, is a very advantageous one for all classes of stopes, flat, vertical, those with uniform dip and strike, or varying dip and uniform strike, or varying dip and varying strike and those on a conical surface. The advantages lie in increased accuracy, a saving of time—as the notes of a stope survey can be plotted in one fourth the time ordinarily required, at the same time dispensing with all laborious reductions—and the cleanliness of plans, as few, if any, erasures are necessary.

19. The plotting protractor consists of (see Fig. 14) a protractor (A) about 13 in. in diameter, graduated to 20 ft., 10 ft. being easily estimated, numbered from left to right as indicated with two sets of figures, one set black and the other set red, 180° in

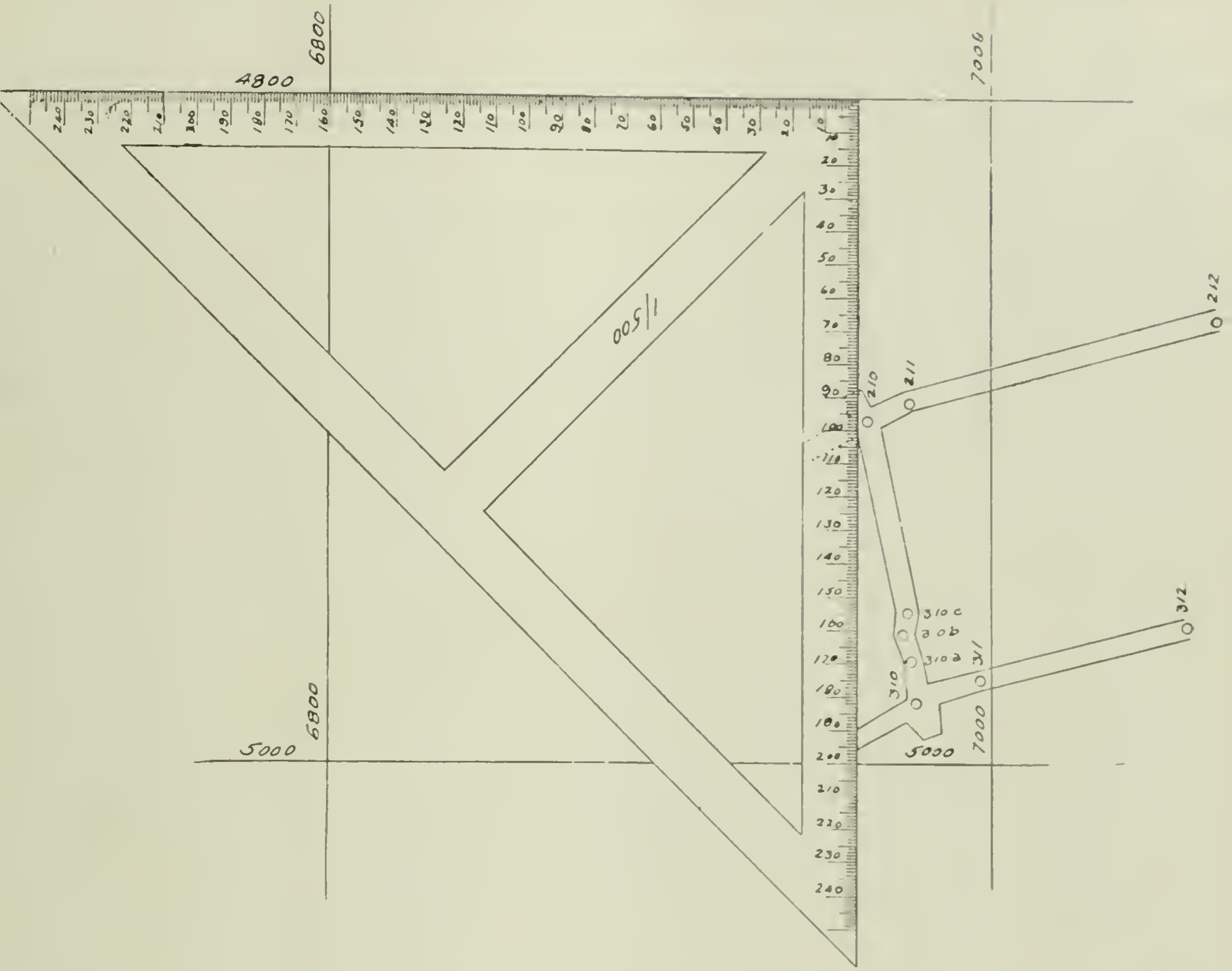


FIG. 9.

advance of the black, the object of which will be explained subsequently. This method of numbering is used in conjunction with those instruments whose graduations are numbered from right to left. (B) is a quadrant—independent of the main graduated circle—graduated to 20 ft. and numbered with two sets of figures, one from 0° to 90° and return to 180°, the 0 division being opposite the 90° red and 270° black of main circle. This set of figures is used in conjunction with those instruments whose vertical circle is numbered 0° to 360° from right to left and whose verniers read 90° and 180° when the telescope is level. The other set of figures is from 0° to 90°, the 0° division being opposite the 0° black and 180° red of main circle. This set of figures is used in conjunction with those instruments whose vertical circle is numbered from 0° to 90° to the right and left from 0° and whose verniers read 0° when the telescope is level. (C) is the graduated scale (in this instance 1/240) and represents the slope distance: (D) is the graduated scale (1/240) placed at right angles to the line passing through (D) the centre of the protractor and 0° black of the main circle and fastened to the slide (E) which is movable along guide (F), so that the 0 division is always on line 0, 0 *b*. (G) is an upright through which passes vertically a fine needle (H) directly over the 0 division on scale (D). This needle upon being pressed down makes a small mark on the surface of the paper and upon the pressure being released is returned to its usual position by a spiral spring. (I) is a fine needle over centre of protractor and when not pressed down is retained free from paper and also from dropping out when protractor is not in use by a small flat spring. (A) corresponds to the horizontal graduated circle of the transit, (B) to the vertical circle, (C) to the measuring tape and C, D, HH, is a right triangle in which C is the hypotenuse or slope distance, D one leg or vertical distance, HH the other leg or horizontal distance and HHC the the angle representing the vertical angle of elevation or depression. By this mechanical arrangement slope distances can readily be reduced to the horizontal and vertical distances determined without recourse to tables. As an illustration, suppose that the slope distance is 100 ft., the vertical angle 30° 00′; move scale (C) until 30° is indicated on (B), then slide (E) until edge of (D) indicates 100 ft. on (C), then read off on scale (E) 50 ft. the vertical distance; (HH) is the horizontal distance and is determined by revolving (C) to line 0, 0; 86·6 ft. is then read off as the horizontal distance.

20. To check (D) that is to ascertain if it is at right angles to line 0, 0, move scale (C) until 60° is indicated on (B), then (E) until edge of (D) indicates 100 ft. on (C) then read off 86·6 on vertical scale, then revolving (C) to line 0, 0, read off 50 ft. If

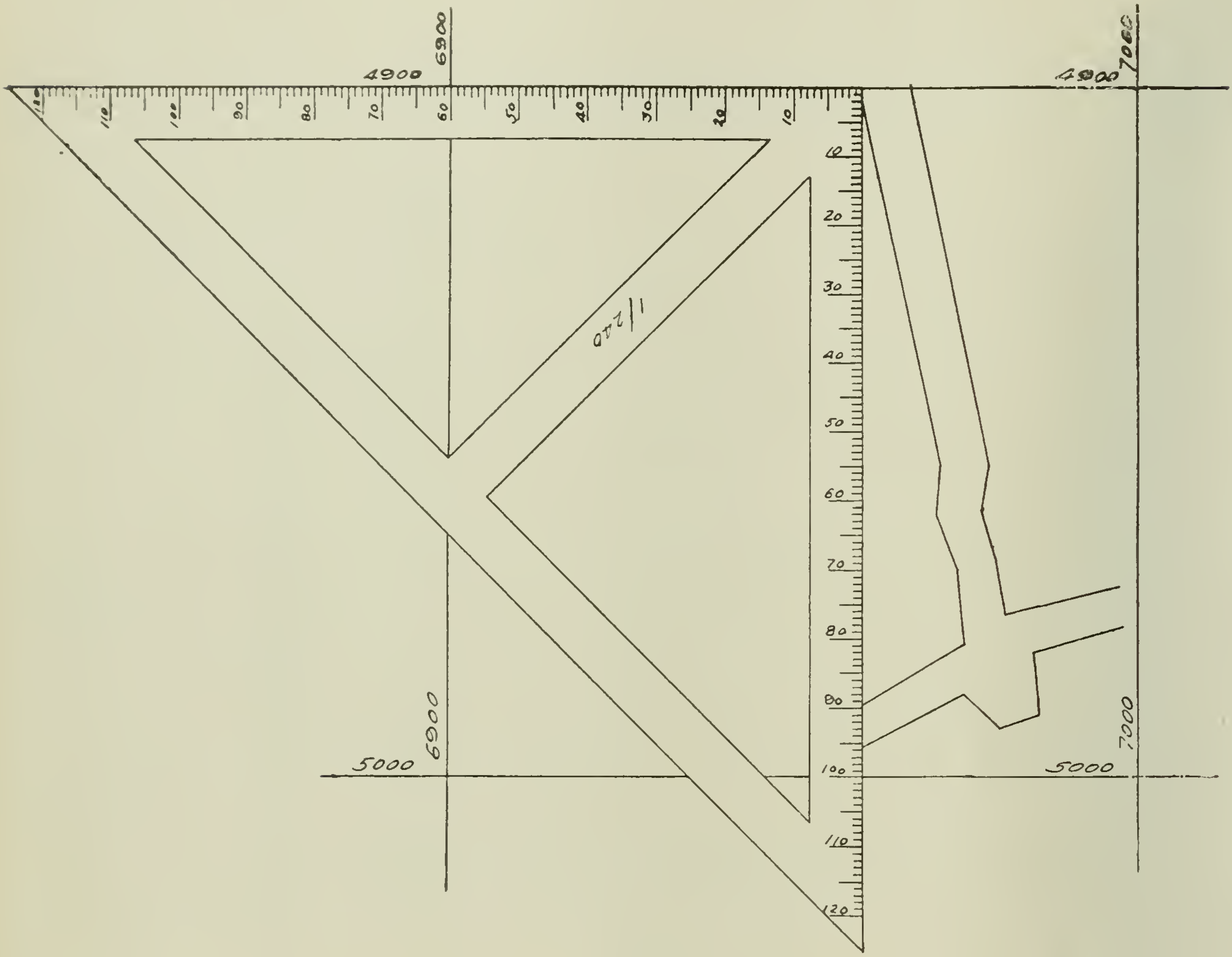


FIG. 10.

there is any difference in the readings obtained by using 30° and 60 (D) may be shifted by means of the screws by which it is fastened to (B) so that the 0 division is on line O, 0° and also so that it is at right angles to line O, 0°. The upright may also be adjusted by means of the screws by which it is fastened to (D) so that the point of the needle when pressed down indicates 0 on scale (D).

21. The application of the protractor is best understood by considering a practical example of a stope survey. In connection with this example the determination of dip and areas will be discussed.

Suppose the notes of two successive surveys of the stope 17 on 1 are as follows :—

Stope 17 on 1 West. Under 1701A. Backsight 1701.

Survey A.

Pt.	H.A.		V.A.		S.D.	V.D.
	deg.	min.	deg.	min.		
1a	204	40	123	00	105·9	57·8
2a	207	40	122	40	100·7	54·4
3a	213	20	121	10	18·8	46·0
4a	218	20	119	30	89·4	44·0
5a	223	20	117	40	84·4	39·4
6a	225	40	116	40	80·2	36·0
7a	229	40	115	00	79·0	33·4
8a	231	40	114	10	74·9	30·7
9a	237	40	111	10	75·6	27·3
10a	240	30	109	10	73·4	24·1
11a	244	10	107	10	69·6	20·6
12a	247	00	105	10	69·5	18·2
13a	250	10	103	30	66·4	15·5
14a	254	20	100	20	64·0	11·5
15a	257	10	98	10	63·6	9·0

Survey B.

Pt.	H.A.		V.A.		S.D.	V.D.
	deg.	min.	deg.	min.		
1b	268	00	90	50	54·6	0·8
2b	264	10	93	50	57·6	3·8
3b	263	50	94	10	62·3	4·5
4b	257	10	98	10	63·6	9·0
5b	254	20	100	20	64·0	11·5
6b	250	10	103	30	66·4	15·5
7b	246	50	105	30	74·9	20·0
8b	239	10	110	00	74·5	25·7
9b	235	40	111	50	76·6	28·5
10b	231	40	114	10	74·9	30·7
11b	224	50	116	10	87·4	39·0
12b	221	50	117	50	94·1	44·0
13b	219	40	119	00	101·4	49·1
14b	218	30	119	10	104·8	51·2
15b	214	20	120	30	107·4	54·8
16b	208	40	122	10	110·8	59·0
17b	204	40	123	00	105·9	57·8

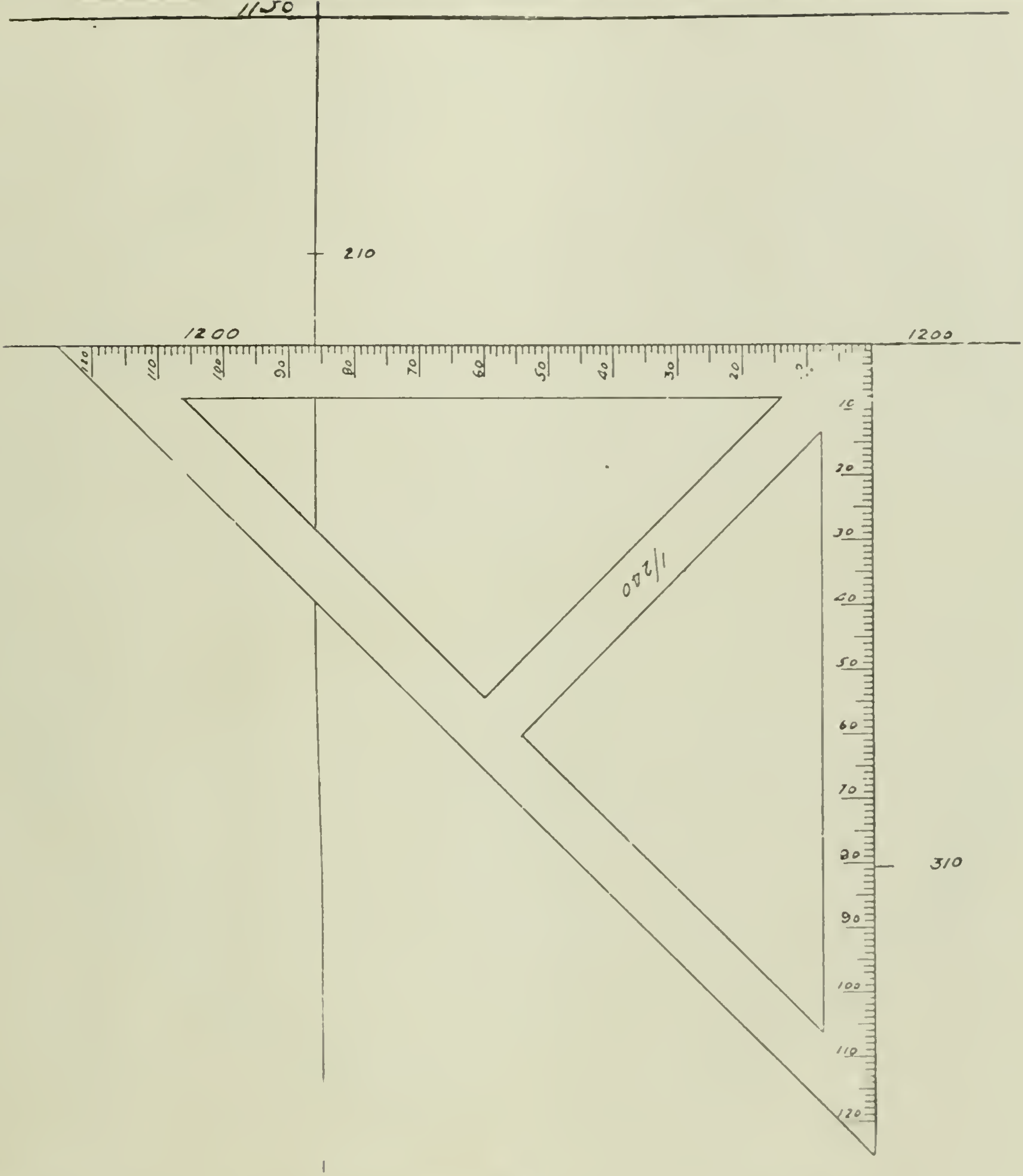


FIG. 11.

Stope 17 on 1 East. Under 1701A. Backsight 1701.

Survey A.

Pt.	H.A. deg.	min.	V.A. deg.	min.	S.D.	V.D.
1a	80	50	93	50	29·0	2·0
2a	84	50	97	10	28·7	3·7
3a	99	10	105	50	31·1	8·5
4a	101	20	105	30	35·5	9·5
5a	111	50	110	20	37·4	13·0
6a	113	50	110	50	41·7	14·8
7a	120	30	113	00	49·1	19·2
8a	124	00	114	10	52·5	21·5
9a	128	10	115	40	53·1	23·0
10a	132	40	117	20	58·4	26·9
11a	138	40	119	50	59·1	29·4
12a	140	10	120	00	63·3	31·7
13a	145	00	121	50	68·6	36·2
14a	153	00	123	50	75·8	42·2
15a	156	40	124	50	83·1	47·5
16a	159	50	125	20	87·3	50·5

Survey B.

Pt.	H.A. deg.	min.	V.A. deg.	min.	S.D.	V.D.
1b	80	50	93	50	29·0	2·0
2b	93	10	101	20	34·6	6·0
3b	101	50	104	50	40·9	10·5
4b	106	20	107	50	46·0	13·9
5b	110	30	109	00	51·5	16·9
6b	115	00	111	10	52·6	19·0
7b	121	10	113	20	60·5	24·0
8b	127	40	116	30	61·6	27·0
9b	132	0	118	10	65·2	30·8
10b	136	40	119	50	72·4	36·0
11b	138	30	121	10	77·6	39·0
12b	144	30	122	20	80·4	43·0
13b	146	30	122	50	84·8	46·0
14b	151	50	124	00	85·8	48·0
15b	157	10	125	20	85·1	49·5

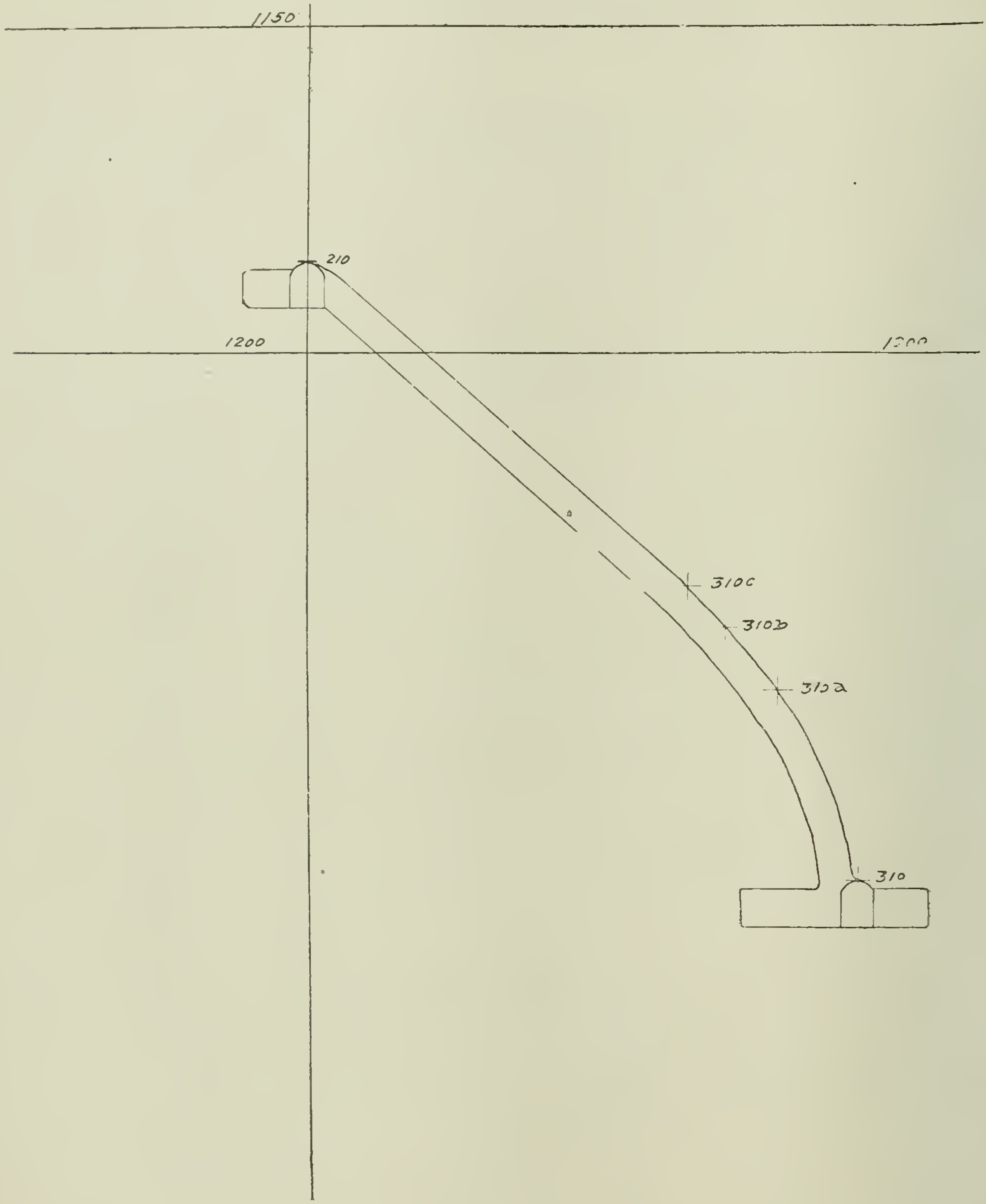


FIG. 12.

22. To plot 17 on 1 West Survey A, draw a line through 1701A and 1701 (as, in this instance, the two stations are very near together it is best to extend the line as previously discussed in the plotting of drives), place the protractor on the plan with needle I pressed down through 1701A and then revolve until 204° 40′ black if line extending through backsight is used or 204° 40′ red if the line extending from backsight, through and beyond 1701A is used, then move scale (C) until 123° 00′ is indicated on (B), then scale (D) until its graduated edge indicates 105·9 on scale (C), the needle (II) is then pressed down and a point marked on the surface of the paper. The vertical distance, to be used later in the determination of the angle of dip, is read off on scale (D) and recorded opposite the station number to which it belongs. All the other points are plotted similarly and the vertical distances recorded. The vertical distances above recorded have been read off from the protractor and, with a few exceptions, will be found to agree with the calculated distances to 0·1 ft.; the exceptions to 0·2 ft. The points thus plotted for each survey are joined by right lines, inked in, thus showing the horizontal projection of the contour of the face of the stope for each survey (see Fig. 15). The freedom of the plan from pencil marks and lines is here noticeable; no erasures being necessary. As the data is not to hand and as the area stoped has no connection with the upper and lower drives they are not indicated on the plan of the stope shown in Fig. 15.

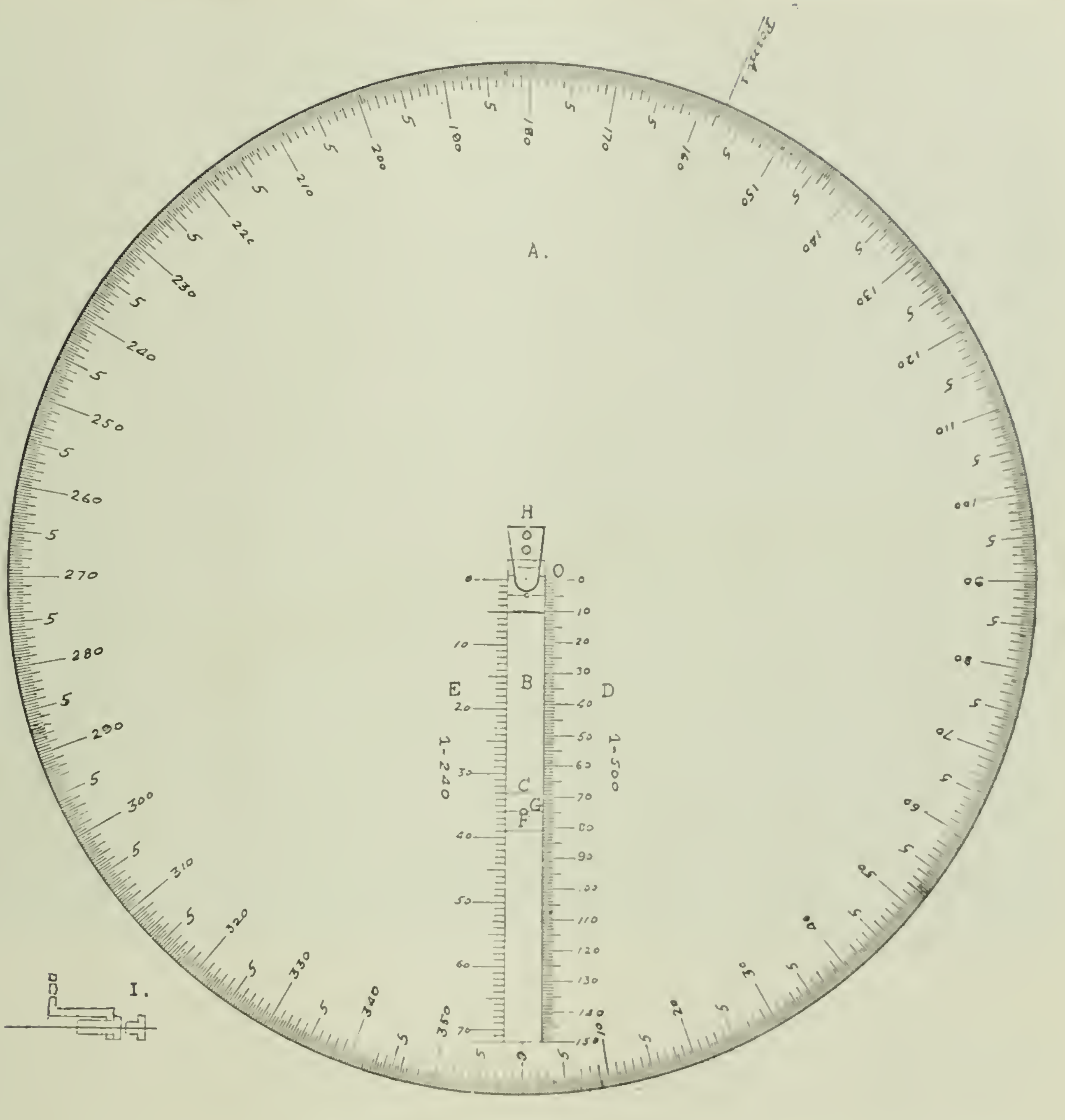


FIG. 15.

23. Before discussing the determination of dip it would perhaps be as well at this point to explain the use of the two sets of figures, red and black on circle (A). If when plotting a stope the line drawn through the backsight would place the protractor in a less convenient position to use, by extending the line in the opposite direction and without altering the horizontal angles as read, the red set of figures would be used and the plotting proceeded with as before. (This has been done in the plotting of the stope above.) Also, when making the underground survey, sometimes owing to the nature of the stope, it is more convenient to read vernier II than vernier I; in this case the angle is recorded as read from vernier II, and when plotting if line through backsight is being used as reference line, the red set of figures is used; if the line extends in the opposite direction, the black figures. And again, it is sometimes necessary to transit the telescope to be enabled to read the vertical circle; in this case of using the line through backsight, vernier II would require the black and vernier I the red, and if using the line extending in the opposite direction, vernier I black and vernier II red. The vertical angles would have to be changed depending on how the vertical circle was numbered and the position of the verniers.

24. Assuming that the strike and the dip for the stope above plotted are uniform, no considerable error, considering the irregularities of the trend of the stope face, will be evidenced if it is assumed that the line of strike passes through a point in 17 on 1 East, Survey B, whose elevation is the same as the elevation of a point in 17 on 1 West, Survey B. For this assumption the point 86 in 17 on 1 West is

[illegible]

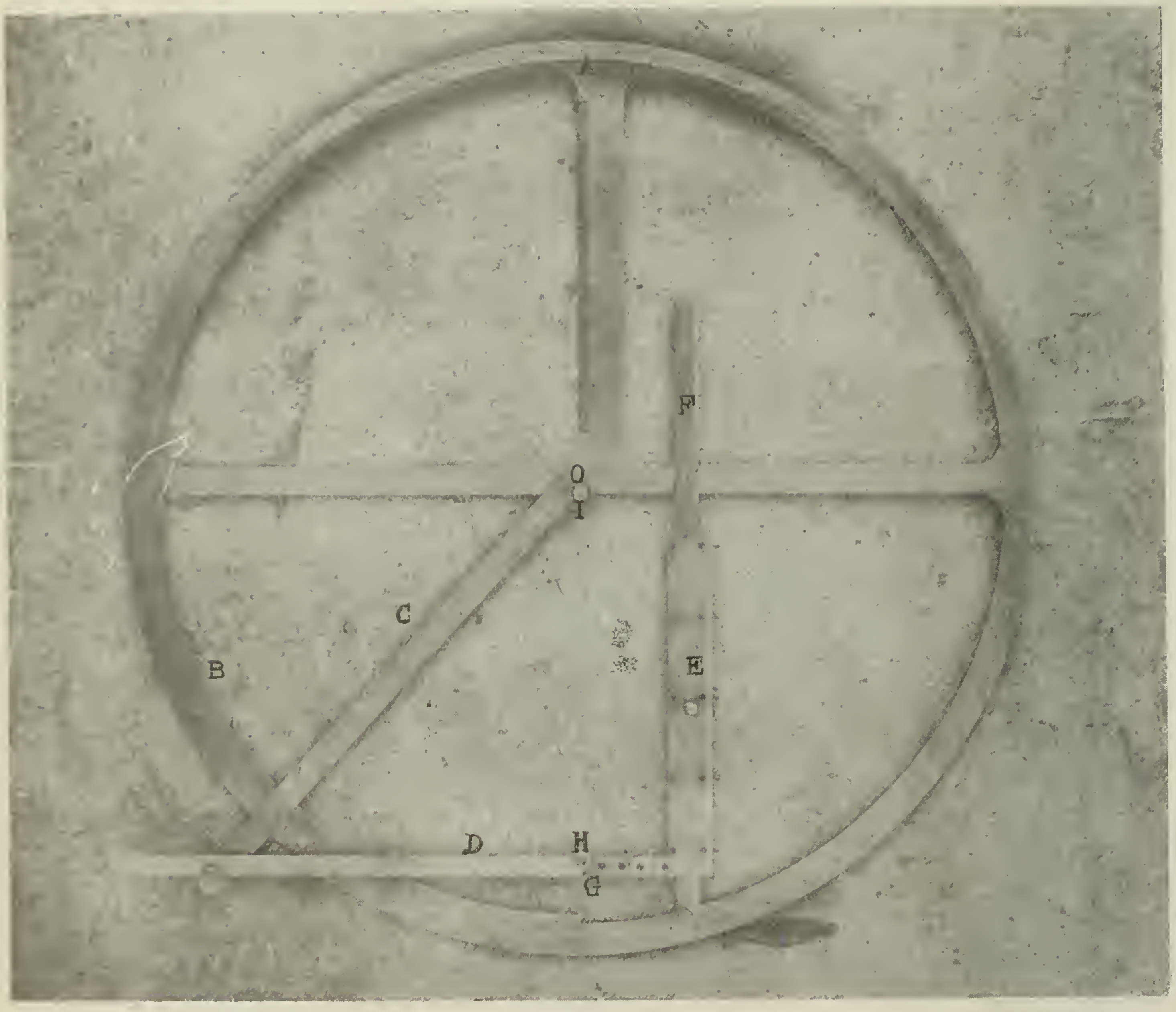


Fig. 14.

vertical distance 25·7 below horizontal axis of instrument and a point in 17 on 1 East half way between points 7*b* and 8*b* the mean of whose vertical distances below horizontal axis of instrument is 25·7 and hence the elevation of the assumed point, will suffice. A line drawn through these two points is the line of strike, which while not the true line of strike, as will develop later, is near enough for practical purposes. At any convenient point draw a line at right angles to the line of strike ; on this line project the points plotted ; then lay off at right angles to this line from each projected point the vertical distance belonging to that point ; join the points thus determined. The contour thus formed is the line of dip (see Fig. 15).

It is preferable to transfer the projected points to a separate paper and lay off the distances thereon. Thus the dip lines for each survey will not be confused with the stope contours.

25. In the dip contour of 17 on 1 East, Survey A, the points from 1*a* to 10*a* are practically on the same line of dip : at 10*a* the dip changes, and from 10*a* to 16*a* the points are on the same dip line. In the dip contour of 17 on 1 East, Survey B, the points from 1*b* to 8*b* are on the same line of dip ; at 8*b* the dip changes, and the points from 8*b* to 15*b* are on the same dip line. The angles of the dip are then taken off with a protractor and, in order mentioned above, are 30° 30′, 36° 30′, 33° 30′, 35° 00′. Draw a line from 10*a* to 8*b* dividing the area into two parts. To determine the slope area of the upper one multiply the horizontal area by the secant of 32° 00′ (the mean of the two dip angles for that part of the stope) ; of the lower one, multiply by the secant of 35° 45′ (the mean of the two dip angles for that part of the stope). The sum is the slope area stoped. In the stope 17 on 1 West, Survey A, the points are practically on the same line of dip, which is determined to be 35° 45′. For Survey B the dip is found to be the same. The area is found as before. As the stoping surface in 17 on 1 East is a warped surface, the more accurate method would be to find points in contour 17 on 1 East Survey B at the same elevation as those in contour 17 on 1 East Survey A, assuming that the change of dip between any two consecutive points is uniform, and then draw lines through points of same elevation. These lines will be the lines of strike at those points : then taking the mean of any two strike lines as the line of strike for the area between those two lines and the mean dip as

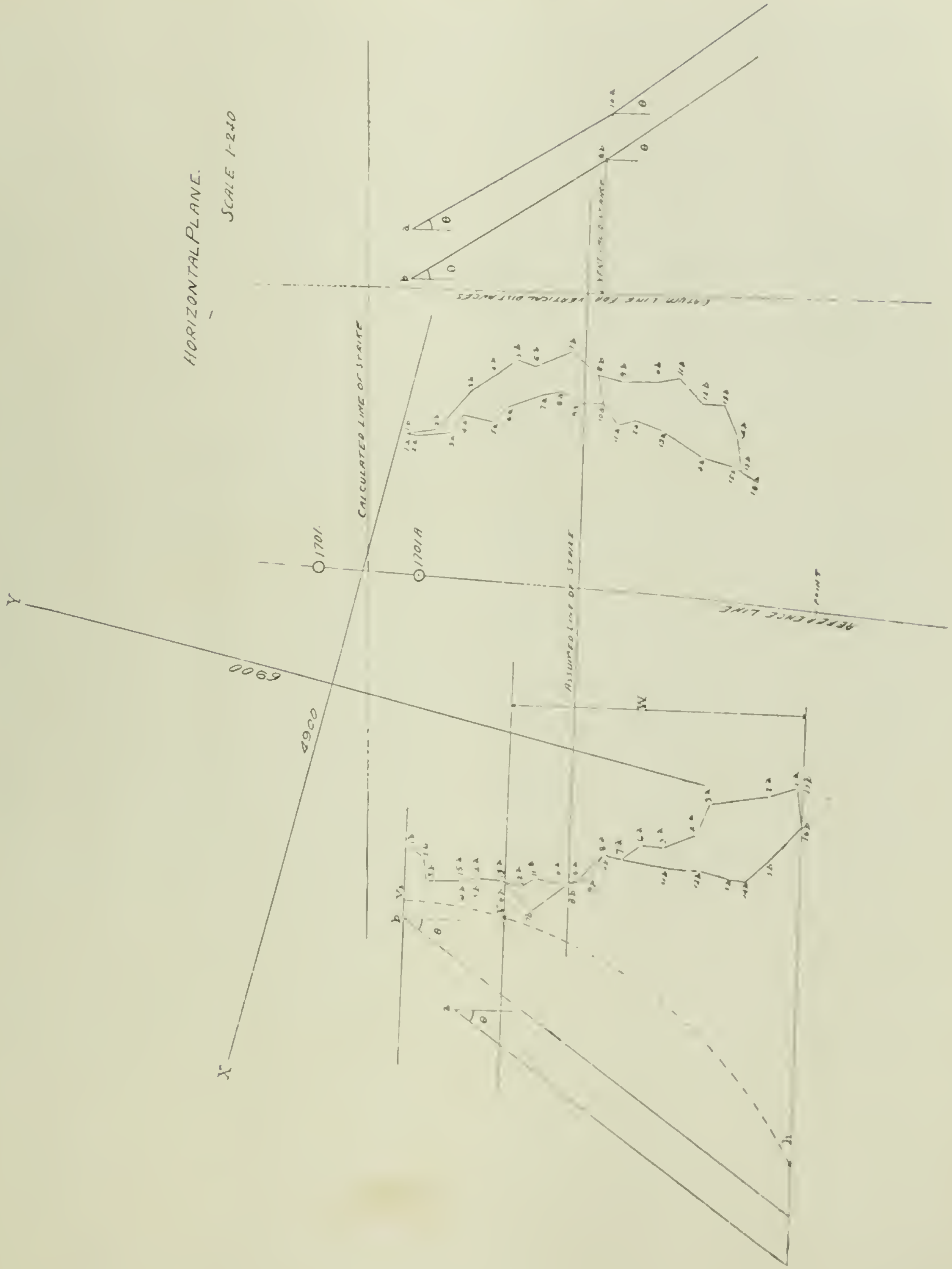

HORIZONTAL PLANE.
SCALE 1-240
CALCULATED LINE OF STRIKE
ASSUMED LINE OF STRIKE
DATUM LINE FOR VERTICAL DISTANCES
REFERENCE LINE
POINT
1701.
1701A
6900
4900
Y
X
W
a
b
θ

FIG. 15.

determined by projecting the points and laying off vertical distances as before, the slope area is determined for that part of the stope. The areas of remainder of stope are determined in the same way and the sum of the whole is the area stoped. For small areas and where the stope is slightly warped the difference of area is inappreciable. If the stoping surface is considerably warped the difference of area is considerable and an application of the preceding method, or one of the following, is necessary for a precise determination of the slope area.

26. In case where the line of strike is constant and the dip variable and hence the contour of the line of dip a curved line, to find the slope area plot the stope as before, and upon determining the line of strike plot the dip line, and either divide the horizontal area into several parts, taking the dip oft the dip line for each part, find the slope area for each part the sum of the several areas being the area of the whole; or find the horizontal area and the horizontal length of this area, measured at right angles to the line of strike, between the line of strike through the uppermost point of horizontal area and the line of strike through the lowest point of area; divide the area by the length giving the average width of horizontal area, multiply this by the length of dip line between the two dip lines, giving the area stoped. For example in 17 on 1 West draw line of strike through 6*b* and 16*b*, measure distance W between them, divide horizontal area by this length giving average width, multiply this by length of dip line (*v h*) giving the slope area (see Fig. 15).

27. The line of strike and the angle of dip may be determined as follows: the co-ordinates of 1701 and 1701A being

	X.	Y.	Z.
1701	6922·16	4891·28	1164·84
1701A	6925·89	4911·44	1178·67

then by calculation the co-ordinates of the points 2*a*, 3*a*, 12*b*, and 15*b* in stope 17 on 1 West, are

2a	6900·88	4992·43	1235·32
3a	6896·41	4981·37	1226·92
12b	6882·62	4982·51	1224·92
15b	6888·40	4996·09	1235·72

6925·89	4911·44	$x=6940{\cdot}81$
6922·16	4901·28	$y=4992{\cdot}08$ co-ordinates of point in extension of line 1701-1701A
3·73	20·16	
4	4	
14·92	80·64	
6925·89	4911·44	
6940·81	4992·08	

Changing the origin of co-ordinates—to facilitate computations—to $x=6880$, $y=4980$ and $z=1220$ the co-ordinates of the above points become

2a	20·88	12·43	15·32
3a	16·41	1·37	6·92
12b	2·62	2·51	4·92
15b	8·40	16·09	15·72

Find the equation of the plane containing any three of the above points, then the intersection of this plane and any horizontal plane will be the line of strike, and the angle between the two planes the angle of dip. The general equation of the plane is $Ax+By+Cz=D$. First taking the points 3*a*, 12*b* and 15*b*, substituting the values of *x*, *y* and *z* in $Ax+By+Cz=D$, solving for B, C, and D in terms of A, substituting the values thus found in $Ax+By+Cz=D$, the equation of the plane is found to be $x+3{\cdot}4802y-4{\cdot}9118z=-12{\cdot}808$. Substituting $z=0$ the equation of the line of strike is $x+3{\cdot}4802y=-12{\cdot}808$. Solving for *y*, $y=-x\div3{\cdot}4802-12{\cdot}808\div3{\cdot}4802$. Whence $\tan\phi=1\div3{\cdot}4802$ $\phi=16°00'$. Hence the strike is N74°00′ E. The angle of dip is given by $\cos\theta=AA'+BB'+CC'\ /\ (A+B+C)^{\frac{1}{2}}(A'+B'+C')^{\frac{1}{2}}$; A, B, C, and A′, B′, C′ being the co-efficients of *x*, *y* and *z* in the equation of the plane $x+3{\cdot}4802y-4{\cdot}9118z=-12{\cdot}808$ and $z=0$ respectively. Making the proper substitutions and solving, the angle of dip is found to be $\theta=-36°24'$

28. Again taking the points 2*a*, 3*a* and 12*b* the equation of the plane is found to be $x+3{\cdot}37254y-4{\cdot}97265z=-13{\cdot}27081$. Whence $\phi=16°31'$. Hence the strike is N 73°29′ E. The mean strike is N 73°44′30″ E. The angle of dip $\theta=35°17'$. The mean dip is 35°50′, which is nearly in agreement with the angle of dip previously found.

29. The discrepancy in the two determinations of the angle of dip is due to the points sighted to being slightly above or below the true line of dip, thus causing a slight error in the vertical distances, which when introduced into the equation of the plane produces a slight difference in the co-efficients of *x*, *y* and *z*, this difference causing considerable difference in the determination of cos θ by above formula. While a difference of half a foot in elevation is not very noticeable in the plotting of the dip line, it is quite so when used in the above formula as is evidenced by the discrepancy in the two angles of dip.

30. That the points sighted to must, to ensure accuracy, be chosen precisely halfway between hanging wall and footwall, or on the line of trend of the stoping face, is evident. If the points are carefully chosen then the probability is that they are all equally in error and, therefore, the mean of two determinations should be taken as the true dip.

31. The direction of the line of strike is somewhat different from the one assumed, but not enough to cause an appreciable difference in the determination of the dip by plotting the vertical distances parallel to the assumed line of strike. This difference is explained by the fact that the East side of the stope dips less than the West side, so that the point between 7*b* and 8*b* East side would at the same dip as the West side be higher up the stope, thus the assumed line of strike would have more nearly approached the calculated line of strike.

32. Another method of determining the dip, if the footwall or hangingwall of the stope is of uniform dip, is by taking sights to three points on either hangingwall or footwall, two in one line on one side of dip line and one on the other side of the dip line, reading the horizontal angles, the vertical angles, and the slope distances.

The form of the notes and formulæ are as follows:—

Inst., Un., B. B.S., A. F.S., x_1,x_2,x_3.

x_1	H.A.	ϕ_1	V.A.	θ_1	S.D.	S_1	V.D.	a	H.D.	d
x_2	,,	ϕ_2	,,	θ_2	,,	S_2	,,	b	,,	e
x_3	,,	ϕ_3	,,	θ_3	,,	S_2	,,	c	,,	f

$\phi_3-\phi_1=90°00'$ Bearing A to B $=\theta°$ If $\phi°$ is the angle made by line of strike with the line x_1,x_2, then $\tan\phi=f\ (b-a)\ /\ (c\ (e-a)-d\ (e-b)$. Hence knowing θ and ϕ the bearing x_1,x_2, can be found and then the bearing of the line of strike. Tan (angle of dip) $=(b-a)\ /\ (e-d)\ \sin\phi$.

33. As a practical example: Bearing A to B S 30°20′ W.

x_1 H.A. 175°40′ V.A. 140°00′ S.D. 17·0
a 13·0227 d 10·9273
x_2 H.A. 175°40′ V.A. 135°00′ S.D. 31·5
b 22·2738 c 22·2738
x_3 H.A. 85°40′ V.A. 122°00′ S.D. 36·0
e 19·0771 f 30·5297

Whence bearing x to x is S 26°00′ W. ϕ = 59°01′. Hence bearing of line of strike is N 85°01′ E. And the angle of dip is 43°35′.

34. The slope area may be found without either finding the angle of dip or the line of strike, as follows: Divide the area into triangles as 1*a*, 2*a*, 16*b*; 2*a*, 16*b*, 15*b*; 2*a*, 15*b*, 14*b*; 2*a*, 3*a*, 14*b* and so on (see Fig. 15, 17 on 1 West), then use the formula

$$\text{Area} = (s\,(s-a)\,(s-b)\,(s-c))^{\frac{1}{2}}\;;\quad s = \tfrac{1}{2}\,(a+b+c).$$

$$a = \left\{(x-x_1)^2 + (y-y_1)^2 + (z-z_1)^2\right\}^{\frac{1}{2}}$$

$$b = \left\{(x-x_2)^2 + (y-y_2)^2 + (z-z_2)^2\right\}^{\frac{1}{2}}$$

$$c = \left\{(x_1-x_2)^2 + (y_1-y_2)^2 + (z_1-z_2)^2\right\}^{\frac{1}{2}}$$

x, y, z; x_1, y_1, z_1; x_2, y_2, z_2 being the co-ordinates of the vertices of the triangles. The fault of this method is that it requires numerous calculations of co-ordinates, although the labour could be shortened by taking the co-ordinates of the points, as plotted with the plotting protractor, off the plan and using the vertical distances given by the plotting protractor.

35. Stopes on a conical surface are plotted as before; the dip and slope area determined considering the stoping surface as plane, the error in so doing being inappreciable except when the radius of the base of the cone is small; in which case the surface may be developed according to the principles of descriptive geometry.

36. It will be noticed (see paragraph 25) that the dip, as plotted, changes abruptly at 10*a*, 8*b*. Actually this is not the case, as there is no slip or fold at 10*a*, 8*b*. Probably the points between 1*a* and 10*a* and 10*a* and 16*a* have been chosen slightly below their true positions and hence the dip lines are curved. The slight difference in the slope area is inappreciable.

37. Sometimes, although the pay streak, or leader, or reef, being stoped preserves its continuity, it may be carried near the hangingwall or the footwall at the time of one survey and at the next survey may be on the opposite wall. This would modify the strike, and hence the angle of dip and the slope area stoped. In a case of this kind the points sighted to instead of being chosen half way between the hangingwall and footwall should be on the pay-streak, or leader, or reef, irrespective of its position in relation to the two walls. This is again modified in case the stope is on contract, depending on whether the contractor has strayed off the leader or owing to a change in the walls, has been ordered to change the position of the stoping face. If the contractor has erred then the sights should be taken on pay-streak or leader, if not, on points half way between hanging and footwalls. In either case the horizontal plan is the same, the slope area alone varying.

38. In the stope 17 on 1 East the assumed line of strike has been used in the determination of the angle of dip. Perhaps a better plan would be to take a point 6*a* and find the point at the same elevation between 4*b* and 5*b*, drawing a line through 6*a* and the point found, giving the line of strike, and then finding the dip; find the slope area for that part of stope 1*a*, 10*a*, 8*b*, and also the point 13*a*, finding a point between 10*b* and 11*b* at same elevation as 13*a*; the line of strike and the angle of dip and slope area as before of that part of stope 10*a*, 8*b*, 15*b*; the sum of the two areas thus found giving the area stoped.

39. In paragraph 26 the method is discussed as though the line *rh* were a curved line, which is not the case. If the line were curved as *rh* (see dotted line, Fig. 15) the length of *rh* would be found with an opisometer and the average width of the horizontal area multiplied by the length found, giving the slope area.

40. In the stope 17 on 1 for another successive survey the mean of the dip line then found and the dip line of the preceding survey would be the dip line to be used in finding the angle of dip and thence the area stoped. If the line of strike were altered in direction in the successive survey, then the vertical distances of the preceding survey should be laid off parallel to the new line of strike—from the projected points of the contour of face of stope at preceding survey—on line at right angles to the line of strike. —J. J. BRISTOL, Mining Engineer, Reno, Nevada, U.S.A., in *Mining and Scientific Press*, April 2 and 9, 1910.

ECONOMIES IN THE USE OF STEEL IN MINE TIMBERING.—"The use of steel in mine timbering can be no longer considered as an experiment. It is a material of construction tested under all kinds of conditions and service, is adapted to all forms of construction, is easy to obtain, convenient to fabricate and erect and economical in all directions, and its long life, under all conditions of temperature, moisture and stress, amply compensates for the increased first cost of its installation as compared with wood. It has, therefore, commended itself, up to the present time, as the only material which, by its flexibility and convenience of use, can replace wooden construction fully and satisfactorily: and it may be well to set down in detail some of its advantages and to point out certain considerations necessary for its most successful and permanent use.

Correct Design.—The prevention of economic waste requires that no more steel be used than is sufficient for the purpose, and that strength and low weight go hand in hand. It requires, therefore, that the steel used in mine timbering be based on correct principles of engineering design. Tables of equivalent strength of wood and steel beams, columns, etc., enable the designer readily to substitute steel for wood in existing operations. At all events the variety of sizes and weights rolled by the steel makers is sufficient to permit a very close approximation to the strength desired so that there may be absolutely no waste of material. There is this difference between the behaviour of steel and wood under service conditions: steel deflects gradually under pressure and will bend greatly before breaking: indeed, unless there are defects in the steel itself, there is no reason at all why a steel beam should break under any conceivable load in the mine, whereas wood, in addition to the pulling apart of the fibre longitudinally, will also break transversely. As a consequence of this difference in the behaviour of the material wood, which has been in place and is overstrained is worthless; the steel, which has deflected so far as to decrease the head room in the roof supports below the clearance line, may be taken out, straightened and replaced. Some of the bars in the Commentry Mines, France, have been used 200 times.

The design of steel mine timbers must go hand in hand with accurate knowledge as to the character of the loads which they have to sustain and the stresses which they produce in the material itself. Where the strata are horizontal, or approximately so, the loads are applied normal to the collar with a practically uniform distribution and produce therein bending moments, with compressive stresses in the legs, these stresses in the legs being equivalent to the half load on the beam, and the minimum amount of material is employed when the legs are exactly proportioned to this half load. Where the dip of the strata is great, or where the ground is swelling, legs and collars may be called upon to take both bending and compressive stresses at the same time, and allowance must be made in the design for this joint action. In a vertical mine shaft through solid ground, the loads which need to be borne will be not much greater than the weight of the lining itself, and stiffness and strength will only be required for the proper alignment of the cage guides, etc. In heavy grounds liable to displacement, or in an inclined shaft, the wall plates may be called upon to carry both bending and compressive stresses, and certainly the buntons will be called upon to sustain a large share of the load.

An illustration of the necessity for an intelligent appreciation of the principles of correct design was seen in the mines of the Copper Queen Con. Mining Co., at Bisbee, Ariz., where certain sections were installed in 1908, in drifts where timber rotted very quickly. They used generally the 4 in. and 5 in. H-sections which took a permanent set after some length of time. Later heavier sections were tried in a part of the mine which was extremely heavy where 12 in. × 12 in. Oregon pine timbers crushed in about a month or less, and in this heavy ground the 6 in. H-section collars and posts failed in about the same time as the timber, the posts bending clear out of shape and crippling badly. The legs in this instance were designed to be of equivalent strength to the collar, whereas experience indicated that they were called upon to sustain not only the loads carried by the collar but heavy bending stresses due to the crushing down of the material on the sides and top of the gangway.

Proper Form.—The prevention of waste in steel mine timbering will, therefore, require the use of the right kind of material as well as the least weight of material. Old rails were first used for the support of roofs in mining operation in 1878 in the mines of Commentry, France. The best results were obtained when they were placed edgewise but they were very liable to breakage. Old rails have also been used in the United States in the Chapin mine of the Oliver Iron Mining Co., at Iron Mountain, Mich., fabricated into square timber sets. They have also been used for stuttles separating the horizontal frames in steel lined mine shafts. The defects of steel rails, however, were early appreciated. They are of small depth and of consequent low efficiency pound for pound of material used. They are also high in carbon and break readily. In the Commentry mines iron bars of rectangular section 3·15 in. × 1·18 in. × 12 ft. long were ultimately adopted in preference to rails, as they were easily straightened and great rigidity is not necessary. In 1889 some of the bars had been re-used 200 times. The roof of this mine was probably of first-class character, otherwise, owing to the weakness of the form of section employed, the sections in use would not have given the satisfaction they seem to have rendered.

Where transverse loads are to be borne in the collars of the square timber sets and for roof supports generally, in the horizontal wall plates of vertical shaft lining, etc., there is no form of section rolled more economical than the standard I-beam used in ordinary construction, as this form of section combines a high degree of resistance to bending with a minimum amount of material. It is not, however, suitable for use as the legs of the square timber sets or as isolated props, by reason of the fact that the radius of gyration of the section is low and it is not therefore adapted to resist compressive strains, to sustain which a shape is needed equally strong about both axis of symmetry. Hollow tubes possess these requirements in a theoretically ideal manner but suffer from the disadvantages of relative high cost and the difficulty of making proper connections, which have practically prohibited their use in building construction above ground except of the simplest character. For the legs of the square timber sets, for buntons in vertical and inclined mine shafts and for wall plates in inclined mine shafts, or vertical shafts through moving ground, the ideal rolled section, of sufficient strength for most ordinary conditions, is given us in the H-section, a form of rolled shape which possesses a large radius of gyration and the most economical distribution of material in comparison to its weight. The 5 in. beam with a 4 in. flange and ½ in. web, which reappears so frequently in English mining literature since 1885, is mine timbering, and it is altogether possible that mine timbering in the United States would have had a different history had these sections been introduced earlier into this country. The first H-sections were rolled in the United States on August 23, 1907, by the Carnegie Steel Co., whose 4, 5, 6 and 8 in. sizes have been designed as the result of careful investigation into mining conditions with an especial view to their use in mining operations. (Fig. 1.) They are respectively equivalent in strength to the best 8, 10, 12, and 15 in. dry southern yellow pine round timbers, and experience in their use has demonstrated that it will seldom be necessary to go beyond this range of sizes.

These relative proportions of the strength of steel to wood, however, are based on ordinary methods of calculation which will allow factors of safety of four on steel and of two on wood, and the use of these equivalents produces, therefore, much stronger and stiffer mine timber sets than the comparison would seem to indicate. Stiffness is as important as strength and the experience of the Copper Queen would indicate the importance of using plenty of steel, just as it is necessary to use plenty of wood. The spacing of the timber sets should be such as to compel the different sets of timber to act altogether as a unit under any sudden stress or shock, and, therefore, light sections with close spacing will be preferable to heavy sections on wide spacing, the roof itself serving as a beam to distribute the load over two or more sets; whereas on wide spacing there is more danger of a roof falling in between the sets. The close spacing also permits the use of much lighter lagging.

Simplicity in Framing.—The prevention of economic waste in steel requires the avoidance of waste in fabrication and simplicity in framing. Nothing can be simpler than the method in which the wooden timber sets are framed and the ideal steel mine timber must also possess this simplicity.

The first use of the square timber sets of structural shapes for main headings seems to have been made in 1897 at Stearn's shaft, Susquehanna Coal Co., Nanticoke, Pa. They were put in 540 ft. below the surface, and the conditions which caused their use were large spans at the foot of the shaft, and great pressure under which 24 in. yellow pine timber

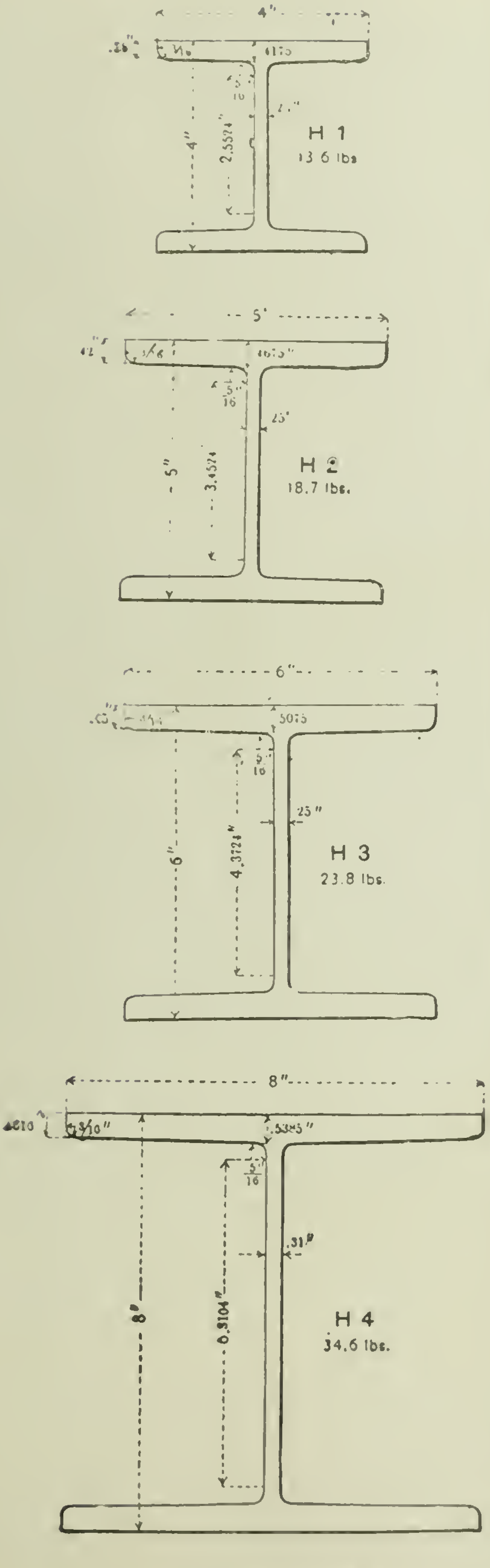


Fig. 1. —H-Sections.

collars lasted only about eight months. These timbers are still in position and in excellent condition, exposed to constant contact with mine water without signs of failure or corrosion and without other protection than good heavy coats of paint. They were constructed in the form known as Steel Gangway Support, Style A, in which the collar is composed of a standard I-beam, with frames at each end between two channels separated by gas pipe separators, and resting at their bases on a cast iron shoe of circular form, which in turn fits into a cast iron base, as shown in Fig. 2. Pin connections are used at the top of the legs, one passing through the beam and channels and the other through the channels only. Steel wedges are driven under the collar so as to distribute the weight on both pins. In this form of construction, for a standard single track gangway it is necessary to assemble 31 pieces and the cost is comparatively high. An installation of this character, however, can be adapted to almost any requirement of strength, and large stretches of steel timbered gangway in the anthracite region are timbered after this design, orignally prepared by R. V. Norris, consulting engineer, Susquehanna Coal Co. The Summit Branch Mining Co., for example, has a single stretch of 1,500 ft at its Williamstown, Pa., colliery, while other long stretches are to be found in the mines of the Lytle Coal Co., Minersville, Pa., Lykens Valley Coal Co., Williamstown, Pa., and Eureka mine No. 36, Berwind-White Coal Mining Co., Windber, Pa. In the last installation the collars are 12 in., 31·5 lb. beams 12 ft. long, the legs are composed each of two 8 in. channels and the cotter-pins are 1¼ in. diameter. The sets are lagged in part with 30 lb. rails and in part with lumber. The simplicity of the construction is shown in that the plain material was bought from the rolling mill and the fabrication was done in the coal company's own shops.

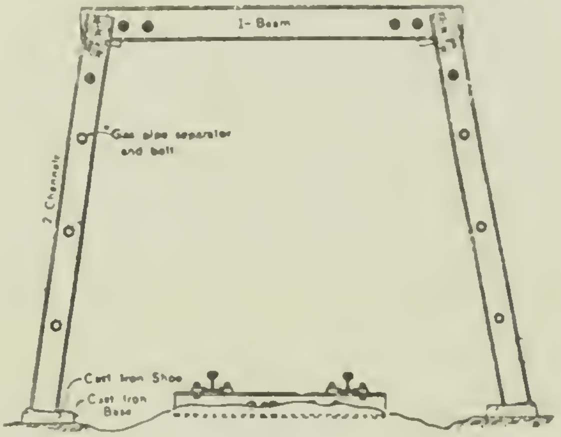


Fig. 2.—Steel Gangway Support, Style A. Pin Bearing, I-Beam Girder, Double Channel Struts, Cast Iron Base.

The three-piece wood gangway supports used in square timbering are not adjustable and, therefore, the adjustability secured by the use of Steel Gangway Support, Style A, seems to be needless and uneconomical. Steel Gangway Support, Style E (Fig. 3) shows a simpler form in which the strength of the double channel leg can be used in combination with an I-beam collar. In this form the two channels forming the legs are connected together in the shop by bolts and separators. They carry angle brackets at their tops on which the beam rests and to which it is connected by bolts, to permit easy removal, and

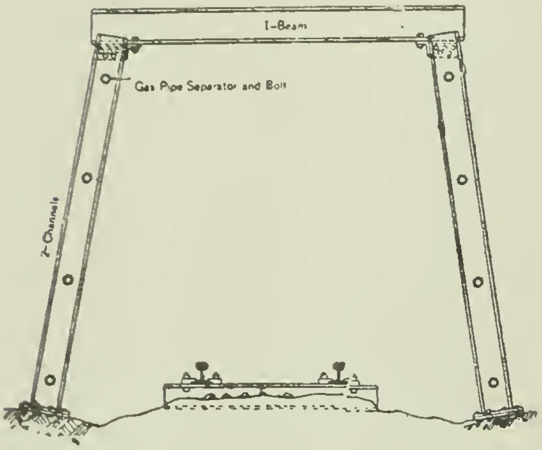


Fig. 3.—Steel Gangway Support, Style E. I-Beam Girder, Double Channel Struts, Steel Cap and Base.

rest at their bottom on a steel plate to which bars are riveted forming slots for holding the channels in position. In this form of construction there are only five pieces to be erected; the single beam collar, two legs and the base plates. If the footing is good even the base plates can be eliminated. The angles riveted to the webs of the channels transmit the load from the collar to the legs and the bent angle brackets prevent undue side motion.

In the endeavour to obtain adjustability in a simple way, Steel Gangway Support, Style I, has been devised, in which a set consists of an I-beam collar and H-section legs connected to each other by rocker castings and resting at the bottom on a casting exactly similar to that at the top, allowing free motion both at the top and bottom. Sixty sets of this type were installed in 1908 in the mines of the Copper Queen, and examples are also to be found in the mines of the Delaware, Lackawanna & Western Railroad at Scranton, Pa.

The ideal form of simplicity and economical construction is to be found in Steel Gangway Support, Style F (Fig. 4) consisting of an I-beam or H-section collar, two H-section legs with lug angles at the top thereof to prevent side motion and absolutely plain plates for bearings. If the plain plates are omitted, there are practically three pieces to handle. The set can be put together on the ground and raised into position with minimum difficulty, and in this form the steel square timber set is the exact equivalent of the three-piece wooden set in such common use. The first sets of this design used in the United States were installed in October, 1907, in the pump house of the Midvalley Coal Co. at Wilburton, Pa., and the steady demand for them indicates their perfect adaption for the usual purposes of gangway timbering. Examples of their use are now to be found in the mines of the Lytle Coal Co., Summit Branch Mining Co., Berwind-White Coal Mining Co., Alden Coal Co., Susquehanna Coal Co., Pine Hill Coal Co., and the Greenough Red Ash Coal Co.

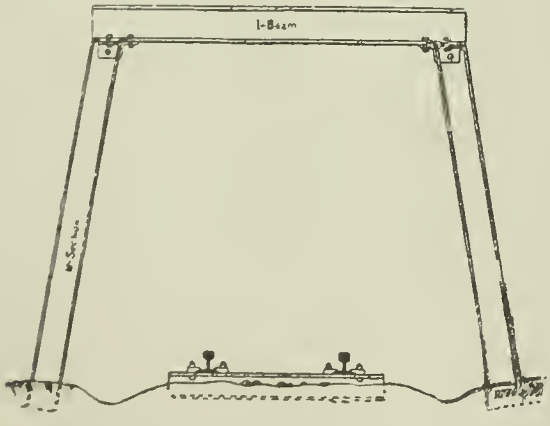


Fig. 4.—Steel Gangway Support, Style F.

The bent lug angle which prevents lateral motion may be replaced by a steel bar, Steel Gangway Support, Style G (Fig. 5) and sets of this character may be found, among other places, in the Maxwell colliery of the Lehigh & Wilkes-Barre Coal Co. In fact, the ease with which steel can be adapted to any condition of service is exemplified in the various simple modifications which have been made in the construction of mine timbers. The ideal steel mine timber of the future will be that in which unnecessary refinement will be eliminated. It need not be any more adjustable than the three-piece wooden set and, like it, will possess extreme simplicity and economy in manufacture.

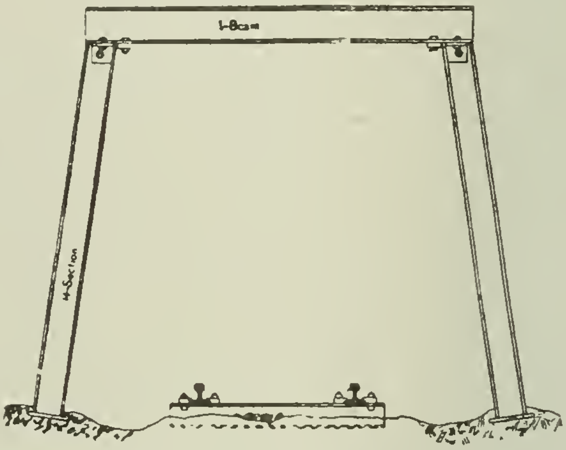


Fig. 5.—Steel Gangway Support, Style G. I-Beam Girder, H-Section Struts, Steel Cap & Base.

Reduced Cost Per Set.—A pound of steel in beam shape is, roughly speaking, equivalent in strength to a board measure foot of the best southern yellow pine or white oak. Under present market conditions the the cost of steel in Pennsylvania is about twice the cost of the wood used in square timber sets. Where the gangway has to be maintained over a number of years and the workings are in any way permanent, consideration should be had to the capitalized value of the material as compared with the first cost of installation and steel will be found economical in most cases on the basis of ultimate cost by reason of its long life and endurance. The reduction in weight to be handled obtained by the use of steel is an item of large importance, leading to cheapness in erection and, therefore, reduction in first cost. As a general proposition, the cost of erection of steel mine timbers should not exceed 50% of the cost of wood of equivalent strength.

Reduced Excavation.—By the substitution of steel for wood in this particular instance, 4 in. of headroom was gained and 32 in. in width. That is to say, had the steel been used for wood in the first place, the excavation could have been made 4 in. lower and 32 in. narrower; quite a large item where the headings are driven partly or entirely through rock. At the Adrian mines of the Rochester & Pittsburg Coal & Iron Co. 4 in. H-sections took the place of 8 in. timbers, resulting in the saving of 4 in. of headroom.

Better Air.—The presence of decaying timber in the mines is also quite a large factor in the deterioration of the atmosphere and the consequent comfort and healthfulness of the operations and, as already indicated, is a large agent in the extension of deterioration to other timbers."—R. B. WOODWORTH. —*The Mining World*, Feb. 5, 1910, pp. 327-8-9. (C. B. S.)

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PIT PONIES AND COMPRESSED AIR HAULAGE.—"Mr. Ashton remarked that in sinking their lower seams in 1903 he decided to do away with ponies underground. One reason was the difficulty of getting lads, the majority of whom, principally due to their home training, were cruel and callous. Many lads between sixteen and nineteen years of age considered themselves to be men and fully entitled to a holiday and to stop the pit when they chose; in fact, to make themselves masters of the situation. He was not in favour of electricity below ground—beyond the pit bottom and pumping, main haulage or lighting—so that the only alternative was compressed air. In compressed air plant they had the best and most economical air compressors, and the best methods of storing and keeping an ample reserve, although some managers had condemned compressed air as a power. An important advantage of compressed air would be to have a good reserve to draw from at the far end of the pipe line. The method adopted at this pit for hauling the coal out of the headings or slits was by small single drum winches which could easily be moved to any position. After six years of experience that method of haulage had proved perfectly satisfactory, and had reduced the cost underground by over 2d. a ton. At his pit they had no intention of again using ponies; they had still a few ponies in one seam, but these were gradually being displaced. The advantages of the method of compressed air haulage were reduced cost and considerably fewer boys required. Therefore trouble was reduced to a minimum. The life of a hauling engine was practically the same as that of a mine, whilst the life of the average pony was only 10 to 12 years.

The speaker proceeded to dwell on the further advantages of air haulage. There was, he observed, no cruelty to ponies; no liability of loss through disease; no veterinary surgeons; no corn, stabling, or harness required; the wear and tear of roads was considerably reduced; and the sanitary conditions of the mine were considerably improved. He drew attention to the recommendation of the Royal Commission that no lads under 18 years of age should be allowed to handle an engine of any description, and pointed out that the danger to the lads in driving these small engines was not so great as the risks attached to pony driving. He hoped that such a foolish recommendation would be strongly opposed by that Association, because if it was embodied in the new Coal Mines Regulation Act it could only result in crippling a great industry."—J. A. ASTON. —*The Mining Journal*, Dec. 11, 1909, p. 436. (C. B. S.)

MISCELLANEOUS.

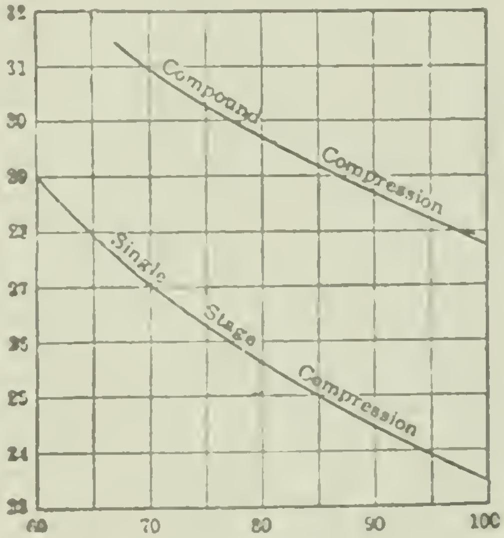
THE EFFICIENCY OF COMPRESSED AIR. —"'What is the ratio of the work represented in the compressed air to that done by the compressor?' This is a question often asked and it is a difficult one to answer clearly.

It is fairly well known that the energy transformation in the use of compressed air is not high, but the fact that it 'does things,' some of them better than by any other means and some which are not done by any other means at all, makes the matter of fuel economy of secondary importance. No apology is needed for the use of compressed air to-day.

That this matter of the fuel economy is not well understood is evidenced by the repeated appearance of propositions to develop the power of a waterfall in the form of compressed air and transmit this through pipes to some distant point, where its energy, like that of electricity, is supposed to be turned back again into work.

Such power transmission propositions as this are absurd, because the losses in the compression and expansion of the air reduce the power efficiency to too low a point. These losses are not losses of pressure by friction, nor air by leakage, but losses due to heating and cooling.

It is not a difficult matter to prove that all of the work of compression goes into heat and is lost in pipe-line radiation. All of the work performed at the far end comes from the intrinsic heat originally in the air, and the economy obtained depends entirely upon how the air is used. The actual cold compressed air travelling in the pipe does not at all represent the energy put into it by the compressor, as this has all been lost in radiation and the only answer to the question first propounded is: 'Nothing.'



Guage Pressure at Drill.

However, we all know that the air does work, and this may be compared to the compressor work (although they have no direct relation), and in this way an 'efficiency' may be stated.

In almost all cases compressed air is used in some form of reciprocating cylinder without expansion; indeed, if expansion were allowed (unless reheating is resorted to) while the air could then give up more work, the moisture always present in the air would quickly freeze, choking the exhaust ports and passages of the machine with ice.

As examples of machines using air with little or no expansion, rock drills and pneumatic tools may be cited, and some interesting figures as to the efficiency of the power transformation are given by the accompanying diagrams.

While actually indicating the drill or tool cylinder would be difficult, some rough idea of the indicated horsepower developed within it may be obtained by a little figuring, together with some judicious guessing.

Indicator diagrams of such machines would theoretically be rectangles, but wire drawing and cushioning effects of the valve mechanism would considerably modify this. It may be assumed, then, reasoning from such a thing as a steam pump cylinder, without cutoff, that the diagram factor will be about 80%. In other words, the actual mean effective pressure will be about 80% of what the theoretical rectangular diagram would give.

On this basis it is determined that a standard rock drill having a 3 in. dia. cylinder will develop about 6·2 indicated horsepower with 100 lbs. at the throttle, this decreasing with the pressure applied, down to about 3·7 indicated horsepower, with only 60 lbs. pressure.

A 3 in. rock drill will require about 138 cub. ft. of free air per minute with 100 lbs. pressure at the throttle; this decreasing to 90 cub. ft., with only 60 lbs. pressure.

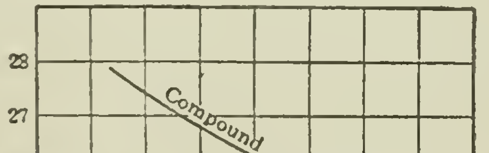
Knowing the quantity of air and the pressure, the compressor horsepower is easily calculated.

Thus, allowing 10 lbs. pressure drop in the pipe, a 3 in. rock drill will require 29·8 indicated horsepower in the steam cylinders of the compressor, with 100 lbs. pressure and single-stage compression, or 25·2 indicated horsepower with compound compression. These figures reduce as the pressure used is reduced, but this, of course, reduces the work done by the tool.

Comparing the probable indicated horsepower developed inside the drill cylinder with the actual compressor power required to furnish the air, gives the probable efficiencies shown by the chart. These efficiencies are referred to both the air and steam cylinders of the compressor, so as to give a basis for calculations for various methods of driving the compressor. They include 10 lbs. pressure drop in the pipe line.

Referred to the air end of the compressor, it is thus seen that with single-stage compression and 100 lbs. pressure, about 23·5% efficiency is obtained, increasing to about 29% with the low pressure of 60 lbs. Compound air compression brings these figures up to 27·8% with the 100 lbs. and 31% with 70 lbs.

Referring to the steam end, allowing 88% mechanical efficiency between the steam and air ends of the compressor, single-stage compression gives a little less than 21% efficiency with 100 lbs. and about 25·5% with 60 lbs. air pressure. Compounding the air cylinders of the compressor increases these figures to about 24·5% with 100 lbs. and almost 27·5% with 70 lbs. air pressure.



Guage Pressure at Drill.

While these figures for efficiency have been determined for rock drills in particular, they apply equally well to almost any machine using compressed air without expansion. It must, however, be remembered that the figures are based upon indicated horsepower only, both in the drill and the compressor. This is because of the impracticability of measuring the brake horsepower of the drill. If, however, break horsepower efficiency is required, these figures for efficiencies of indicated horsepower can be multiplied by the mechanical efficiency of the device using the air, say 90% or 80%, as the case may be. This, of course, gives a still smaller result.

It is to be noted that the higher efficiencies are obtained with the lower pressures This is because there is less loss by heating the air during compression and, therefore, it is advisable to use pressure as low as is consistent with the size and weight of the machine required to do a given amount of work."—Snowden B. Redfield, *Mining Science*, June 2, 1910, p. 515. (C. B.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

Dictionary of Metallurgical and Chemical Material. pp. 219, paper cover. Price 50 cents. Published by Metallurgical and Chemical Engineering, 239 West 39th Street, New York, 1910.

"This convenient and time-saving little volume is a model of conciseness. It gives the reader accurate descriptions of the different makes and special features of any metallurgical or chemical apparatus and material. A directory of manufacturers is included. Supplementary sections are headed 'Measuring Instruments and Laboratory Supplies,' and 'Professional Directory.'—*Canadian Mining Journal*, Oct. 15, 1910. (A. R.)

More Recent Cyanide Practice. Edited by H. Foster Bain. pp. 424. (San Francisco: *Mining and Scientific Press*, 1910. $2.

This work consists of the more interesting articles contributed, from October, 1907, to July, 1910, to the *Mining and Scientific Press*. Among the authors represented will be found W. A. Caldecott, Courtenay de Kalb, Philip Argall, E. M. Hamilton, Alfred James, F. L. Bosqui, and many other well known metallurgists.

As no attempt appears to have been made to allow contributors an opportunity of revising their work a number of unnecessary errors have been perpetuated such as gold for cold, on p. 96, an incorrectly drawn graph on p. 95, and the heading, "Size of Gold, per cent.," on p. 89.

A discussion of considerable interest on the treatment of an ideal ore with strong resemblance to Rand banket, initiated by "Viator" (W. A. Caldecott), is here rescued from the oblivion of contemporary technical literature and will be highly appreciated in this country. Among the more interesting articles will be found notes on use of zinc dust, filter presses and vacuum filters, agitation vats, tube mills, cones and diaphragms, mechanics of ore crushing and on kindred subjects of use and interest to cyaniders and metallurgists generally. A work of this kind is valuable in that it represents a concentration from much matter of ephemeral interest, and its only drawback is the fact that only one of the numerous technical papers is made use of in its preparation.—(H. A. W.)

Boving, J. O. Water Turbine Plant, a Short Practical Treatise. 4to., pp. 206. *Raithby, Lawrence.* Net 10s. 6d.

Howe, J. Allen. The Geology of Building Stones. Cr. 8vo., pp. 464. *E. Arnold.* Net 7s. 6d.

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Walker, James. Introduction to Physical Chemistry. 6th Edition. 8vo., pp. 430. *Macmillan.* Net 10s.

Abstracts of Patent Applications.

(C.) 4 10. William Pill (1), Samuel Thomas (2). Improvements in and relating to linings for tube mills. 4.1.10.

This application relates to an improved tube mill liner which is described as a double lining, constructed of pebbles, stone, etc., cemented together and kept from stripping by bars of iron or steel. The drawing shows the liner to be laid in two courses with the bars longitudinally arranged, and of the same depth as each course.

(C.) 7 10. Edmund Paul Millington Jerrard. Improvements in and relating to mine signalling instruments and the like. 6.1.10.

This invention relates to an improved "Mine Signalling Bell Pull." The inventor seeks to bring out an improved type, more especially as to its being waterproof, and also at the same time to afford greater accessibility for repairs. With this latter end in view he makes the working parts removable as a whole from the casing.

(C.) 23 10. James Edward Thomas. Improvements in or relating to tube mill feeding devices. 27.1.10.

This application relates to improvements in tube mill feeding devices, and consists of the combination with a spiral feeder for introducing grinding material into the mill; of a hopper and hinged chute to deliver the grinding material from the hopper into the spiral feeder—said chute being provided with adjustable stop and weighted hinged lid or cover and in which the lower end of lid is bent upwards and extends beyond the sides of the chute.

(C.) 35 10. John Whitford (1), Joseph Amos Mills (2). Improvements in mine ventilation. 3.2.10.

The object of this invention is to provide to underground workings a quality of air which will be free from moisture and other impurities that can be removed by refrigerating the air before it is introduced to the working places.

(C.) 115 10. William Charles Chergwin. Improvements in apparatus for use on sands and other dumps. 18.3.10.

This application refers to sectional metal chutes made for laying on the side of a sands dump, so as to cause the sand dumped from a truck to be conveyed to the bottom of the dump at a less steep angle than at present, where the sand slides down the face of a dump without any chutes.

(C.) 233 10. C. M. R. van Lennep. Improvements relating to the mortar boxes of stamp mills.

This application refers to the method and apparatus necessary for renewing the battery screens on a mortar box from time to time without stopping or hanging up the stamps.

This device consists of two rollers—one at either end of the screen opening—one of which carries the new screen and the other the worn screen as it is changed, one screen rolling up while the other screen unrolls. There is also provided a clamp for clamping the screen tightly to the screen frame when it is moved into position.

(C.) 437 10. Frank George Seineke. Improvements in means for straightening or altering the direction or course of a bore hole. 3.9.10.

This invention relates to means for straightening the course or direction of a bore hole which has deviated from the vertical, or for altering the course or direction of an inclined bore hole to the vertical.

The invention comprises a long cylindrical plumb-bob of a diameter which will permit it to just pass through the boring crown, as more particularly set forth in the specification attached to the application, in combination with the boring apparatus, a flexible means for suspending said plumb-bob from said boring apparatus, so that it can hang vertical at the point at which it is desired to alter the direction or course of the bore hole and means for fixing the plumb-bob in the latter position as set forth in the specification: the plumb-bob after it has been firmly fixed in position by cement or otherwise to act as a guide for the boring crown and drill rods.

(C.) 442 10. Robert James Worth (1), Worth, McKenzie & Co. (2). Improvements in connection with the valve controlling gear of winding engines for collieries or the like purposes. 7.9.10.

This invention relates to means for controlling the steam distribution valves of winding engines by which the valve gear is regulated from a control drum provided with one or more suitable cam paths and driven from the engine shaft by positive gearing at a rate not exceeding one revolution for each complete wind, so that a lever or its equivalent, adapted to gear in the relevant cam path, makes not exceeding one stroke as the relevant cam path runs its course, during which stroke a regulated motion is transmitted through intervening mechanism to the reversing motion where this is arranged to be used as an expansion gear also, to a separate expansion gear where this is fitted or to both, whilst at the same time the same or a similar drum also by means of its cam arrangement and intervening mechanism transmits a regulated motion to the throttle valve, or to the throttle valve and break gear, in combina tion with a hand control and sometimes further combination with a speed governor.

(C.) 446/10. Robert J. McNitt. Improvements in methods for reducing metals. 9.9.10.

This application relates to a new method of reducing metals, and consists of combining the desired metal with another metal which shall be volatile only at higher temperatures and lower pressures than the desired metal; then subjecting the combination to the action of heat under a low pressure to drive off the more volatile metal and condensing same.

As an example, sodium is electrolysed from sodium chloride and collected on a lead cathode. This alloy of lead and sodium is then conveyed to a suitable receiver, the pressure reduced and then heated to drive off the sodium which is condensed and collected, the remaining lead being available for further use.

(C.) 464/10. Kurt Wilhelm Otto Schweder. Improvements relating to centrifugal pumps. 17.9.10.

This application covers an extension of the ordinary centrifugal pump column for the purpose of regulating the outflow of the liquid pumped.

It is well known that with ordinary centrifugal pumps there is usually an intermittent flow, due to the head that the pump is capable of working against being in excess of the actual head due to the pump column. This device of Mr. Schweder's is an extension of the pump column which can rise or fall to make the head against which the pump works the minimum at which the pump will work with a constant flow at the capacity required from time to time. As the capacity changes, this device adjusts itself automatically to give a uniform flow.

(C.) 469/10. Frederick Arthur Maximilian Schiechel. Apparatus for separating minerals and like materials. 22.9.10.

This is an apparatus for separating minerals from gangue by means of a continuous flow of water rising upward through the ore to be separated, which lies upon a screen or other suitable bed. A constant motion is produced in the ore bed either by agitating or shaking the mass or by imparting pulsating shocks in quick succession to the rising stream of water.

(C.) 486/10. William Beaver (1), Charles Harrison Dixon (2). Improvements in means for separating the fine product from ores particularly applicable to conveying belts. 6.10.10.

This application relates to means for separating the fine from ore delivered from a conveyor belt. The device shown is an adjustable chute arranged to divide the ore stream at a point where centrifugal force makes the separation and takes out the fine for delivery to a washing trommel below. A stream of compressed air is added to assist the separation.

(C.) 518/10. Richard Truste Carter. An improved concentrator and amalgamator. 19.10.10.

This refers to an apparatus to be used as a concentrator and amalgamator consisting of a cylindrical vessel with pockets to be revolved at a circumferential speed not less than 1,000 ft. per minute. Water is supplied under pressure to the pockets and the heavy material driven into them by centrifugal action is thus prevented from packing too tightly and is at the same time cleaned.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 549/10. George Francis Jones. A process for obtaining hydrated carbonate of lead direct from cerusite. 10.11.10.

(C.) 551/10. Herberth Nosworth Roche. Improvements in electric water heaters. 11.11.10.

(C.) 552/10. Niels Christian Nielsen. Improvements in the method of avoiding formation of rust in tinned iron tanks or vessels. 11.11.10.

(C.) 553/10. Christian Eliza Liebenberg. Improvements applicable to headgear crushing plants for collecting the rock-dust produced therein. 12.11.10.

(C.) 554/10. Robert Irving. Improvements in percussion drills. 12.11.10.

(C.) 555/10. William M. Metzler. Improvements in and relating to conveyor belts. 12.11.10.

(P.) 557/10. William Arthur Caldecott. Improvements in the treatment of crushed ore products. 16.11.10.

(P.) 558/10. Ludwig Wahner (1), George Wilbur Goff (2). Improvements applicable to rock drilling machines for introducing water into the holes formed therewith. 16.11.10.

(C.) 559/10. George Llewellyn Davies (1). Walter Edwin Windsor Richards (2). New or improved process for the treatment of liquid hydrocarbons. 16.11.10.

(C.) 561/10. Reinhold Steinback. Improvements in grinding mills and the like. 17.11.10.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

BEATTY, J. H., *l/o* Nigel; Bantjes Consolidated Mines, P. O. Florida.
CARBIS, A. R., *l/o* Eureka; New United Reefs, P. O. Box 27, Barberton.
CARBIS, W., *l/o* Eureka; New United Reefs, P. O. Box 27, Barberton.
HARRIS, J. V., *l/o* Gwanda; Mount Morgan and Boulder Claims, West Nicholson, Rhodesia.
NICOLAUS, GEORGE R., *l/o* Westminster; Percy Tarbutt & Co., 18, St. Swithin's Lane, London, E.C.
PHILLIPS, J. M., *l/o* New Lisbon Berlyn, Pilgrims Rest; Zaaiplaats Tin Mine, P. O. Sterkwater, Potgietersrust.
THOMPSON, G. DE ROOS, *l/o* Langlaagte; New Rietfontein G. M. Co., P. O. Box 15, Rietfontein.
SHARPLY, HAROLD, *l/o* Lincolnshire, England; Blackwater Mine, Waiuta, S. Island, New Zealand.

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VOL. XI. JANUARY, 1911. No. 7.

Proceedings
AT
Ordinary General Meeting, January 17, 1911.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the South African School of Mines, on Saturday evening, January 17th, Dr. James Moir (President) in the chair. There were also present :—

64 Members : Messrs. W. R. Dowling, F. F. Alexander, R. Allen, K. L. Graham, Tom Johnson, G. O. Smart, J. E. Thomas, H. A. White, James Littlejohn, A. F. Crosse, W. A. Caldecott, E. H. Johnson, A. McA. Johnston (Members of Council), H. A. Adams, R. A. Barry, S. Beaton, G. H. Beatty, W. Beaver, A. J. Bowness, Dr. Johnston Brown, J. Chilton, F. W. Cindel, F. C. Ford, E. A. Furner, R. Gascoyne, A. D. Gilmore. F. G. Guthrie, R. E Hall, B. J. Hastings, Dr. L. G. Irwine, R. N. Kotzé, W. W. Lawrie, G. A. Lawson, H. Lea, J. Lea, W. Lea, Dr. M. Mehliss, J. E. Metcalf, P. Morrisby, S. Morison, M. T. Murray, W. Nicklin, W. J. R. North, E. A. Osterloh, C. F. Parry, J. F. Pyles, E. Roberts, G. A. Robertson, W. H. Roe, O. D Ross, A. Salkinson, K. Sartorius, A. H. Scarf, S. Shepherd, S. H. Steels, Ralph Stokes, W. A. C. Tayler, A. Thomas, O. Tonnesen, Chris. Toombs, H. Warren, F. W. Watson. C. F. Webb and E. M. Weston.

22 Associates and Students : Messrs. M. S. E. Archibald, J. P. Beardwood, F. T. Carbis, B. Cook, E. P. Cowles, J. Cronin, C. L. Dewar, J. M. Dixon, P. H. Grumitt, A. Hague, B. Hasserus A. King, G. W. Leach, L. T. Leyson, C. S. McLean, S. H. Olivier, F. J. Pooler, P. Scatterty, H. Stadler, A. W. Stockett, A. M. Thomas and J. Thorlund.

30 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as printed in the December *Journal*, were confirmed.

NEW MEMBERS.

Messrs. Tom Johnson and J. Littlejohn were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows :—

CROUCHER, HERBERT, Waikino, New Zealand. Metallurgist.

DEANE, W. M., Talisman Consolidated Ltd., Karangahake, New Zealand. Metallurgist.

DELPRAT, THEODORE DANIEL, Messrs. Reunert & Lenz, P. O. Box 92, Johannesburg.

HENRY, JOHN MITCHELL LA FAYETTE, Jupiter G. M. Co., Ltd., P. O. Box 27, Germiston. Mine Manager.

MARKS, LEWIS, Consolidated Gold Fields Laboratory, P. O. Box 108, Germiston. Metallurgist.

QUINCE, WILLIAM ALBERT, Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights. Mining Engineer.

The Secretary : Since the last meeting of the Society the following have been admitted by the Council :—

As Associates.—

BASKETT, EDMUND GRAHAM, *c/o* Maoriland G. M. Co., Ltd., Waitekauri, Auckland, New Zealand. Metallurgist

FRASER, CHARLES, St. George Mine, Mount Magnet, W. Australia. Assayer

HAGUE, ARTHUR, M.Sc. (Birmingham), Knights Deep, Ltd., P. O. Box 431, Germiston. Metallurgist.

LEFRERE, VALENTIN VINCENT DE PAUL, Ferreira G. M. Co., Ltd., P. O. Box 1021, Johannesburg. Timekeeper.

TUCHTAN, LUDWIG JOSEPH, General Mining and Finance Corporation, Ltd., P. O. Box 1212, Johannesburg Leading Draughtsman.

GENERAL BUSINESS.

The President : I am requested by the Council to call attention to the last paragraph on the agenda, under the heading of "Future Meetings." "The Council again desires to call the attention of authors to the fact that needless expense is incurred by lack of proper attention to clearness and legibility of MSS." I need not say anything more. It arises out of the fact that a lot of proof reading and editing has to be done which is really unnecessary. But to avoid mis understanding, I should say the Society, as far as I know, has never refused a paper on account of the form in which it is received. It is quite willing to pay the money for proof-corrections, but feels that it is unnecessary in most cases.

STATIONARY AMALGAM PLATES IN TUBE-MILL PLANTS.

Mr. W. R. Dowling (*Vice-President*): It has been the generally accepted practice on these fields that the amalgamating plates in a tube-mill plant should shake as it was considered that the pulp was too concentrated to allow the gold particles to come in contact with the amalgamated surfaces of the plate. Following the pioneer work of Mr. Fred Maxwell, of the Randfontein Group, some experimental work on stationary plates has for some time been in progress at the joint plant of the Knights Deep and Simmer East mines. Fifteen plates each 10 ft. 5 in. long by 4 ft. 7 in wide, of a three tube-mill plant were stopped shaking in October last, and the fall of the plates increased from 10 to 18%. Under these conditions amalgamation was most satisfactory. For the two months preceding the introduction of stationary plates the extraction by tube-mills was 21·81% of the screen assay value and 34·04% of the tailing value. For the two months succeeding the extraction was 22·94% of screen assay and 34·02% of the tailing assay.

With a view of reducing the number of plates if possible, it was decided to try a concentrating scheme. For this purpose a plant consisting of two small cones and two stationary tables was erected. It was found that the gold released from the sand by tube-milling could be concentrated into the underflow of the cones, and that the underflow with some diluting overflow could be handled by two plates. During this work Mr. W. A. Caldecott suggested that it might be possible to handle the effluent of a tube-mill with a minimum of diluting water direct over two plates. To test this scheme five and two plates took the pulp from one tube-mill on alternate days. The upshot of this trial was that there was a difference of 1·2 oz. of amalgam scraped per day between the two methods in favour of the two plates. Encouraged by these results it was decided to put the whole plant on to two stationary plates per tube-mill and this has now been going on for some time, the amalgamation still continuing to be good.

In adopting two plates instead of five or more it is important that the water ratio should be kept low and it has been found that a moisture of 55 to 57% or a ratio of solid to liquid of about 1 to 1·27 is about the irreducible minimum. At first sight the velocity of pulp over the two stationary plates seems too great for efficient amalgamation, but all the tests such as panning of tailing, final pulp, sand residue and assays of the latter, combined with the amalgam recoveries go to show that the method is satisfactory.

Reduced to its simplest form amalgamation simply means bringing the amalgamable gold into contact with the amalgamated surface, and this result is attained in the system outlined above, full details of which I hope to lay before the Society at an early date.

Whilst on the subject of amalgamating it might be mentioned that the Knights Deep tube-milling plant has been running since May last with a proportion of plates to tube-mills (22 ft by 5½ ft.) of 3·26 to 1 and for the past seven weeks with a proportion of 2·85 to 1. The plates shake and are 10½ ft. long by 4¾ ft. wide.

The President: I am quite sure we should thank Mr. Dowling for his exceedingly interesting results. They are most surprising. I should like some explanation as to why it is better to keep the plates still than to shake them?

Mr. W. R. Dowling (*Vice-President*): I think it is quite unnecessary to shake them. You get all the results you want. As long as you have sufficient contact between the gold and the amalgamated plate you get the results.

The President: You are going to bring the matter up later in fuller form?

Mr. W. R. Dowling: I should like to.

WORLD'S GLYCERINE SUPPLY.

The Secretary read the following contribution:—

Mr. W. Cullen (*Past-President*): Most people know that the basis of the explosives which are consumed on the Rand, and indeed throughout South Africa as a whole, is nitro-glycerine. One might even go further and say the same holds for mining throughout the world, but this was not always so. In spite of the invention of compositions which were alleged to be many times as strong as nitro-glycerine, this body and explosives having it as their base now occupy a position which for rough and ready work is absolutely unchallenged, and will probably remain so. Nitro-glycerine, as you know is made from glycerine, which is a by-product of the soap and candle industry. It is not so very long ago since it was run to waste, and this continued until indeed nitro-glycerine found its place in the world of explosives. *Now* it has a value of £100 to £110 a ton, and the reason for this phenomenal advance may well be asked. The question is easily answered. Quite apart altogether from the unique position of nitro-glycerine explosives as compared with others, their consumption as a class has increased enormously during the past two decades, but the most phenomenal increase has been during the *past* decade. Every Power which has a standing army has adopted powder which contains nitro-glycerine in greater or lesser proportion. This applies to the Japanese, among others, and Dr. Weiskopf is now manager of the factory where the explosives are made for the navy. Then take our own South Africa. Within the past ten years the consumption has more than doubled; in Canada during the same period it has probably quadrupled, and even in the older European States the increase during the same period has been phenomenal. Then the Panama Canal itself could take the entire production of Modderfontein or Somerset West. All this increase would not matter so much if the industries from which glycerine comes as a by-product were expanding correspondingly, but this is not so. People may be using more soap, but it is questionable whether they are using more candles. At any rate there has not been an expansion corresponding to that in the consumption of explosives, so the only thing which could possibly happen has happened. There is a shortage and prices have soared up. Some people are unkind enough to advance other reasons, but the one I have given is the true one and can be verified by independent examination. The position in a very short time, if it is not to-day, is that there is not enough glycerine to go round. The outlook is exceedingly serious, and that is why I sound this note of warning. People in the "know" have been trying to find a substitute for blasting gelatine for the past few years, and I am giving away no secret when I say that one has not been found as yet, nor do I think it will be. Gelignite and gelatine dynamite do not offer so much difficulty but I cannot give you the composition of any of the "substituted explosives" because I do not know them myself yet. I am safe in saying, however, that the existing processes at Modderfontein will be profoundly altered, as they will be in almost every factory in the world. How about mining? If a substitute for blasting gelatine is not found will our mining methods want modifying? My impression is that they will want profound modification. We shall have to find this out, and it is imperative that we start doing so now, for the changes which I have indicated will come about very soon. I appeal to mine managers and others to let us have facilities for carrying out experiments, and to assist us in every way they can, because although I am rather the reverse of an alarmist I must give it as my considered opinion that the position is a most serious one. Certainly explosives will not be *lower* in price for a very long time to come.

The President: We have to thank Mr. Cullen for this important warning. I hope the Press will take note of it, for it is just as well that the general public should know that there will probably be an increase in the price of explosives quite soon. Those mines which have made contracts for explosives a long way ahead are going to save money out of them. If I am right in supposing it costs about 1s. a ton broken for explosives now, it may cost 2s or 3s, a ton if we do not find a different explosive in a reasonable time. Of course there are other high explosives, only hitherto for unrevealed reasons they have had no chance in this country against those in use. It is to be hoped that experiments will be set in order at once, and that they will be successful.

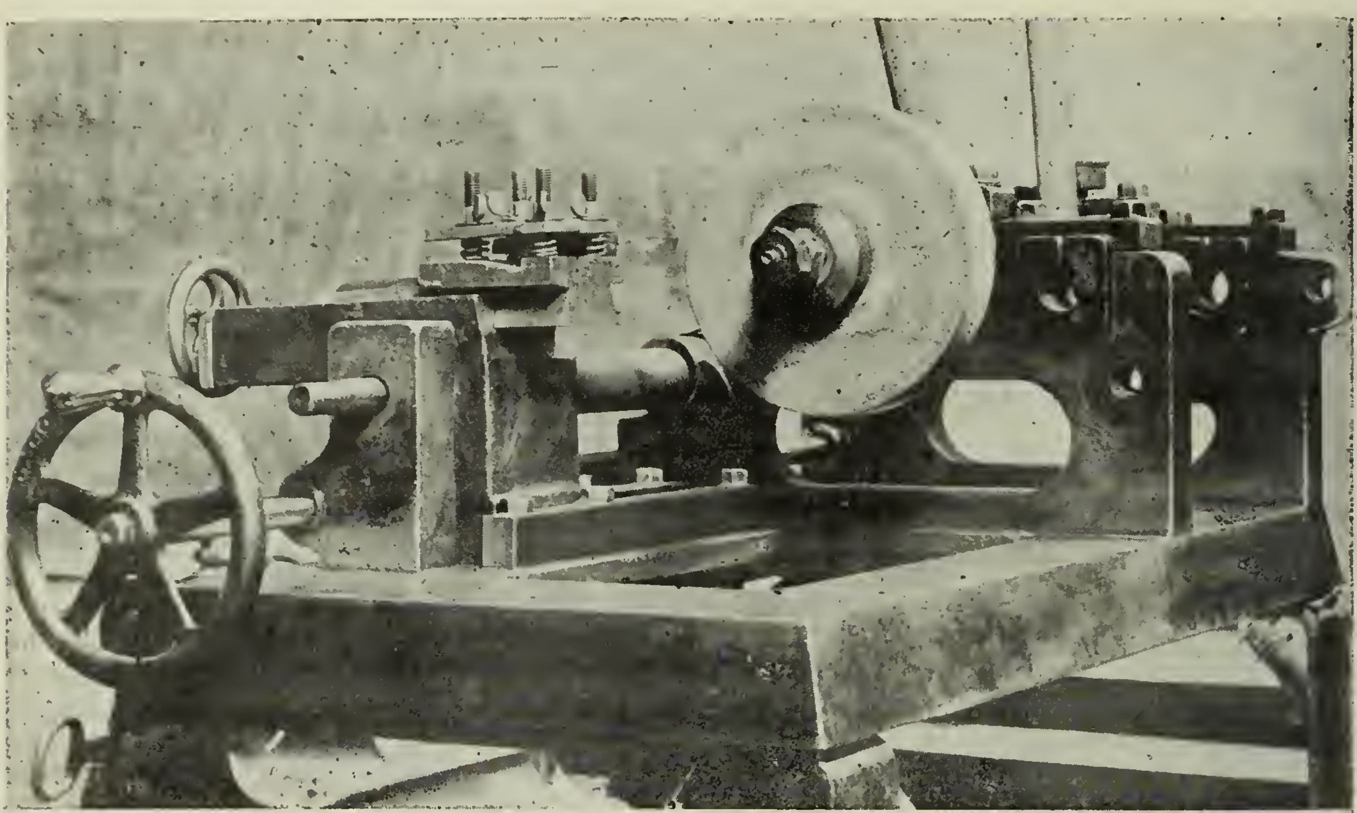
A GRINDING MACHINE FOR ZINC-CUTTING TOOLS.

Mr. F. F. Alexander (*Member of Council*) submitted the following note on behalf of the author:—

Mr. H. Brazier (*Associate*): I have recently been desired by several who have seen the above machine in operation to furnish the *Journal* with a description, for the benefit of any of its readers who may be interested.

It was introduced by me into the Crown Deep extractor house in 1908, and has given every satisfaction since its installation. The best condition of the tool, the correct cutting angle, and the necessary clearance, so that clean and economical cutting may be ensured, are difficult to obtain with hand ground tools; but with the machine here described, tools of a perfectly regular pattern are obtained, and there is no risk of deviation by subsequent regrindings.

Briefly described, the arrangement is an adaption of a discarded Liddell zinc lathe, whereby the tool to be ground is placed in the sliding tool rest, parallel with the periphery of an emery wheel, the latter mounted on the spindle that formerly carried the zinc discs. The tool holder of the slide rest has a bevelled recess, so that the tool is always held in the same position at all subsequent regrindings. By these means a "hollowing" is effected, and a repetition of the cut with a quarter turn of the tool ensures a similar grind on the upper face, the junction of the two curved faces forming the cutting edge. All the sharpening for a shift can be done most effectively by a boy in ten minutes. We use "V" or "Profile" self-hardening tool steel, cut up into 16 in. lengths with the ends cut at 45°, and these lengths are reground until only 3 in. remains between the cutting edges. Previous to this machine being installed, one Betty and two Liddells, employing six coolies and using on an average one emery wheel a month, were kept



constantly going to supply zinc for a 300 stamp plant, whereas now one Betty running twenty-two days a month on a "one tooth" feed, and employing only three coolies, supplies ample zinc for 355 stamps; whilst the emery wheel at present running was put on in June, 1908. This was an ordinary 12 in. × 2 in. "medium," and it now measures 10 in. × 2 in. Our steel bill is also appreciably less. Although but a lathe pure and simple, I believe any cyanide manager possessing an old Liddell machine, and converting to the above purpose, will find it both a time and labour saver.

Mr. Andrew F. Crosse (*Past President*): The question of zinc shaving having been mentioned, I should like to state that when in London, about two years ago, I saw a very interesting process for producing fine ribbon metal, gold, tin, aluminium, silver and zinc, etc. The metal is melted and escapes through a fine hole, impinging on a tube of metal which is kept cool by water passing through it; the tube revolves about 800 revolutions per minute, and the metal is thrown out in the shape of a ribbon; the thickness and also the breadth can be altered. The chief difficulty in regard to the product of a suitable zinc ribbon for precipitating gold is the brittleness of the zinc ribbon produced as described. I am confident that this difficulty could be got over.

A Member: I should like to mention that about eighteen months ago I had an offer from a London Company to supply zinc at 6d. a pound f.o.b. London, whereas it is now costing us about $4\frac{1}{2}$d. a pound, so we did not do business.

Mr. F. F. Alexander (*Member of Council*): As regards our own experiments on annealing zinc we find it is rather a difficult process. To turn out molten zinc in ribbon form is in my opinion almost impossible.

Dr. J. L. Aymard (*Associate*): In support of what Mr. Crosse says, I may say I have seen this ribbon It was only a question of price. It was most perfectly turned out, and is exactly as Mr. Crosse has described it.

Mr. H. A. White (*Member of Council*): I should like to ask if it is perfectly smooth on both surfaces?

Dr. J. L. Aymard: So far as I remember the ribbon was perfectly smooth. It was rather like that obtained in magnesium ribbon.

The President: We must thank Mr. Brazier for producing all this interesting discussion.

NATIVE FOOD SUPPLIES AND THEIR QUALITY.

By F. W. Watson, B.Sc. (London), F.I.C., F.C.S.

In April, 1905, the late Dr, M. M. Klein read a paper before the South African Association of Engineers entitled "Food and the Practical Aspect of same in Compound Work." Therein Dr. Klein dwelt on the "three alimentary principles (1) albuminates or proteids, (2) fats or hydrocarbons, and (3) carbohydrates," and showed the necessity of a proper combination of these three principles in the diet.

In the report of the Mining Regulations Commission published last year reference is made to the proper dieting of the native and evidence was taken on this point. In giving evidence before this Commission Dr. G. A. Turner suggested the use of "monkey-nuts" in order to make up the deficiency of fat which, as Dr. Klein had pointed out, was generally ignored in the food usually given to natives. Dr. Turner also in his evidence tabulated the articles of diet used by the natives from different parts of South Africa showing by which the carbohydrates were represented, which the proteids and which the fats In the *Journal* of this Society of September, 1910, there is an abstract of the report of the health conditions on Rhodesian mines. In this report the committee, consisting of Dr. A. M. Fleming, C.M.G., Chief Medical Officer, Rhodesia, Dr. Donald Macaulay, M.L.A. (Transvaal), and Dr. W. Gladstone Clark, M.A. (Bulawayo) commented among other things upon the diet of the native and laid special emphasis on the necessity of using fresh vegetables as anti-scorbutics. In these reports and papers the diet of the native is dealt with from the physiological aspect by members of the medical profession showing what is necessary for the good health of the native, thereby increasing his efficiency and reducing mortality. There are, however, no figures given showing the percentages of the ingredients in the various articles of diet *actually* supplied to the compounds; figures when quoted have been taken from analyses made elsewhere and nothing has apparently been done to test the food supplies purchased by the mines.

In a paper read recently by Mr. P. Snowden before the South African Association of Engineers the author deals with the quality of the food supplied and also comments on the unskilled cooking prevalent in compound kitchens. At the present time most of the materials supplied to the mines are, or ought to be, bought on specification. We have had papers in this Society dealing with the percentage of CaO and the precipitating effect of partially burnt coal in lime, and it is recognised that the seller has to supply lime up to a certain standard or a penalty will be enforced. Similarly with regard to the percentage of KCN and Na_2S in commercial cyanide a certain standard is set. In coal the calorific value is usually required, with perhaps the proximate analysis: and the same applies to cements, oils and many other materials. The seller has to supply an article up to a certain standard or the consignment may be refused. Up to the present it would appear that in the supply of foodstuffs for the compounds very little has been done to test the articles of diet supplied, though guarantees are generally asked; consequently an inferior article may be purchased with enhanced profit to the seller and a loss to the mine.

In the present paper I do not intend to deal with meat or fresh vegetables. These do not often come under the analyst's notice as they are not liable to be adulterated, and except in cases of ptomaine-poisoning or suspected addition of preservatives, these articles are not likely to be sent in for inspection to a chemical laboratory. In other articles of diet, such as cereals, beverages, etc., considerable variation in the quality of the material supplied is noticed and a great deal can be done by the chemist to assist the compound manager and storekeeper in the purchase of supplies. In samples of bread which have come under our notice considerable variation takes place from time to time in the percentage of nitrogenous matter, acidity, moisture, etc. Also in samples of nuts which are bought chiefly for fat content a great variation is found in the percentage of this constituent, usually pointing to the fact that some starchy material has been added.

The first cereal to be considered is mealie (maize) meal, as this is bought in large quantities by the mines and is an important constituent in the native diet. In this country owing to maize being plentiful there is not much temptation for suppliers to adulterate, but elsewhere other starches are sometimes added, potato starch in particular. These foreign starches can of course at once be detected by the microscope. I shall here give an analysis of an average sample of mealie meal supplied to the compounds and alongside of it an analysis of white Australian flour purchased locally. For comparison between the mealie meal and the grain before milling I give an analysis of mealies which were ground in the laboratory in a small coffee mill. These are designated whole maize.

	Mealie Meal.	Flour.	Whole Maize.
Water ...	11·00%	10·30%	11·15%
Nitrogenous matter	10·28 ,,	11·25 ,,	9·54 ,,
Fat	3·86 ,,	1·60 ,,	5·22 ,,
Carbohydrates ...	73·01 ,,	76·15 ,,	70·64 ,,
Fibre ...	0·94 ,,	0·14 ,,	2·36 ,,
Ash	0·91 ,,	0·56 ,,	1·09 ,,
Phosphoric anhydride (P_2O_5) ...	0·48 ,,	0·27 ,,	0·51 ,,
Acidity expressed as glacial acetic acid	0·28 ,,	0·17 ,,	0·23 ,,

On comparing these analyses it will be seen that the mealie meal compares favourably with the flour; the fat content is much higher, but flour has a slight advantage in nitrogenous matter. It will also be noticed that in the milling of the mealies a considerable amount of the fibre is removed and the fat content is reduced, probably owing to removal of the bulk of the embryo.

A certain percentage of mealie meal is often mixed with flour in baking bread for the compounds. This may perhaps have some bearing on the relatively high acid value of the bread baked here to that which is quoted elsewhere as an average acidity.

In the samples of mealie meal which we have examined we have not yet found a case in which adulteration could be detected or on which any suspicion could rest. No foreign starches have been observed.

Mealie meal, as a food, is chiefly valuable for heating power. It has a large percentage of carbohydrate material, and although, among the cereals, maize comes next to oats in point of view of fat contents, yet the amount of this constituent is much too small, and one would have to consume an unduly large amount of starch in order to obtain the amount of fat and nitrogenous matter required for a healthy diet. To make up for this deficiency the pea nut (monkey-nut, ground-nut) is one of the articles bought for the compounds. This, as will be seen by the analysis given, is very rich in fat and it also contains a high percentage of nitrogenous matter. In the samples of this article of diet, which we have examined, a big variation has been observed and in some cases microscopical examination has shown that a foreign starchy material has been added. These nuts are usually bought ground, which is probably a mistake, since it might be better for the mine to buy them whole and do the grinding. To illustrate this I shall give here the analyses of five different samples of pea nuts which were submitted.

	A	B	C	D	E
	per cent	per cent	per cent	per cent	per cent
Water	5·15	5·31	4·68	6·20	3·62
Nitrogenous matter ...	29·69	27·85	27·12	20·44	19·25
Fat	47·36	44·28	47·00	36·81	34·45
Carbohydrates ...	12·84	17·59	15·66	32·26	12·05
Fibre	2·60	2·59	3·06	2·13	2·67
Ash	2·36	2·38	2·48	2·16	2·08
Salt (NaCl) ...	Nil.	Nil.	Nil.	Nil.	25·88

Sample A consisted of whole nuts which were ground in the laboratory whilst samples B, C, D and E were already ground when submitted. It will be noted that the whole nuts are highest in fat and nitrogenous matter but that the ground sample C is practically the same in fat content. B is not much below A and C, but D and E are considerably lower. This is due in E to the addition of salt as shown in the analysis, and in D a microscopical examination showed the addition of foreign starches. A noteworthy point about these samples is that C and D were about the same price, this price being considerably lower than that of B, which emphasises the necessity for having these stores tested from time to time and kept free from adulterants.

In buying these nuts it would be advisable, in settling a price, to specify a certain percentage of fat as this is the constituent for which this article is usually bought. These nuts are sometimes pressed and part of the oil extracted. The cake left in the presses might be mixed with the ground nuts but the nitrogenous matter would not decrease in the same proportion. The high value in carbohydrates bears out the addition of a foreign starch as found by the microscope in sample D.

Let us now turn our attention to the bread supplied to the compounds. This article varies considerably from time to time In the baking of the flour changes take place in the composition of its components, a certain amount of fermentation goes on, with formation of alcohol and CO_2, and some of the insoluble starch is rendered soluble, while the alcohol is mostly driven off in the baking. Th. Bolas found in six samples of bread the following percentage of alcohol.*

Minimum	·221
Maximum	·399
Mean	·313

Certain changes also occur in the nitrogenous bodies, soluble albumen and globulin becoming insoluble and are no longer capable of being separated from the starch.*

In the breads supplied locally it is sometimes specified that so much per cent. shall be flour and so much mealie meal. Whether this percentage is adhered to or not would be almost impossible for the analyst to determine as, in the baking, the starch grains of the different cereals lose most of their special characteristics when viewed under the microscope, and as this, the best test for a foreign starch, fails we are unable to say whether the specified percentage is adhered to or not. If too much mealie meal is added the bread won't bake properly owing to deficiency in gluten. Potatoes are often used in bread making, the undersized potatoes being sold at a low price so that by substituting this article the baker can effect a considerable saving in the cost of his loaf. A deficiency in nitrogen is shown when this article is employed. The mineral adulterants in bread are chiefly copper sulphate and alum. Adulteration by either of these substances we have never found practised by the bakers supplying bread to the compounds. Alum can be detected by the logwood test, and its quantitative estimation in the ash is an easy matter

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* Foods, their Composition and Analysis. A. and M. Wynter Blyth. 6th Edition, p. 163.

I shall now give analyses of samples of three lots of bread supplied to the same mine at different times by the same baker.

	A.	B.	C.
	Per cent.	Per cent.	Per cent.
Crust	31·5	26·4	36·7
Crumb	68·5	73·6	63·3
Moisture	36·40	30·43	29·60
Ash	0·72	1·15	1·55
Nitrogenous matter ...	11·52	8·50	11·81
Acidity	0·18	0·60	0·24
Alum (grains per lb.)	3·61	7·77	4·40

The acidity is expressed as glacial acetic acid and the alum as crystallised ammonium alum $NH_4Al(SO_4)_2 12H_2O$. These samples were all of brown bread and show on the average a high value in nitrogenous matter. A and C are about the average of the article supplied to this mine. B is a much poorer quality and shows up badly in both nitrogen and acidity. Since this sample was analysed the quality of the bread has kept up to the standard of C most probably on account of the baker being notified that his bread was much below standard. The average amount of fat in the bread supplied to this particular compound is 1·15%. I shall now give two analyses of bread supplied by another baker to two different mines. This was white bread and resembled baked rice somewhat in structure. The crust and crumb were not determined in these cases as they were not well defined.

	E.	F.
	Per cent.	Per cent.
Water	28·17	28·65
Ash	0·79	0.46
Nitrogenous matter	9·16	7·56
Fat	1·12	11·09
Acidity	0·23	0·19
Alum	Trace	Trace

In sample F. the high percentage of fat is noteworthy. This is due to the fact that the flour had lard added to it before baking to make up for the natural deficiency of fat in this article of diet. These two samples are not so moist as the first three quoted but they have not such a high value in nitrogenous matter. The acidity is lower than the first three but it is still above the limit given by Notter and Firth, who state that 0·114% ought certainly to be the limit.*

All the breads supplied to the compounds which have come under our notice have a higher acid value than the limit above quoted, this may be due to local conditions or the mixing of other grains with the flour. It is advisable to test this acidity constantly as it varies considerably and ought to be kept within reasonable limits. Sample B, it will be noticed, has an acid value more than double that of any of the others.

Beans are sometimes bought by the compounds; this article has a relatively high value in nitrogenous matter but is much below the pea nut in this respect. A sample of the Jugo bean, a South African product which was submitted, was found to contain 18·37% of nitrogenous matter and 2·36% of fat. In composition this article of diet resembles the Haricot bean very closely and is inferior in nitrogenous matter to either dried lentils or peas. It is deficient in fat and in this respect resembles a cereal.

We shall now turn our attention to the liquid refreshment supplied to the native. The beverages which have been submitted by the compound managers have been coffee, cocoa, and the much-discussed Kafir beer. Of the first of these the samples submitted were roasted and ground coffee and chicory. I shall give partial analyses of two samples.

	A	B
	per cent.	per cent.
Water	1·40	4·60
Fat	12·29	10·26
Nitrogenous matter	12·69	12·50
Dextrose	3·23	Nil.
Ash	4·70	2·85
Aqueous extract ...	34·26	49·36

The sample A contained caramel but this was probably formed in the roasting. The aqueous extract was obtained by exhausting 25 grm. of the sample with boiling water, filtering and making up to 250 cc. when cold. This in sample A gave a specific gravity of 1·0158 at 60°F. A similar extract of pure coffee has S.G. 1·010 that of chicory 1·022. This would give coffee 52% and chicory 48% in the sample, but these can only be taken as approximations as the roasting had partially destroyed the tissues, so that the microscopical examination was rendered difficult and adulterations might thus escape detection.

Sample B had a foreign starchy material added to it. In this instance also the roasting interfered with the microscopical examination, but as far as could be seen the starch resembled bean flour. This adulterant interfered also with the estimation of the coffee and chicory by means of the specific gravity of the aqueous extract. I am strongly of opinion that it would pay the mines to buy the coffee beans whole and grind them in the compound and add the chicory as required, as if roasted and ground mixtures are bought it will be a difficult matter to ensure that the article is up to specification.

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* Theory and Practice of Hygiene. Notter and Firth. 2nd edition, p. 361.

The next beverage is cocoa. Mr. P. Snowden in his recent paper comments on the value of cocoa as a food. Cocoa has undoubtedly a high value in fat and also a comparatively high value in nitrogenous matter, but the amount taken at a time is small. Dr. Robert Hutchison says* "The place of cocoa in the diet is not really very different to that of tea and coffee. An examination of the chemical composition of cocoa might lead one to suppose that it was of considerable nutritive value. But this would be a mistake. Theoretically, cocoa is a valuable food but practically it is not, the reason being that so little of it can be taken at a time. . . . It takes about 10 grm. ($\frac{1}{3}$ oz.) of cocoa to make a breakfast cupful of the beverage, and, assuming the average composition given already, this would yield about 40 (kgm.) calories of energy. It would, therefore, require seventy-five cupfuls to yield the total amount of potential energy demanded of the body daily—obviously an impossible quantity."

I shall here give the analyses of seven samples of cocoa which were submitted.

	A per cent.	B per cent.	C per cent.	D per cent.
Water ...	4·00	5·17	3·28	6·38
Fat	26·48	27·39	24·73	23·77
Starch ...	13·52	6·05	7·10	12·10
Sugar... ..	Nil	Nil	Nil	Nil
Nitrogenous matter	17·50	18·53	18·81	18·69
Fibre	3·13	4·36	4·12	5·46
Ash	4·98	4·66	5·91	6·68
Acid equivalent of ash ...	5·31	4·90	3·20	5·62
Calories ...	5·67	5·73	5·40	5·61

	E per cent.	F per cent.	G per cent.
Water	1·36	6·52	5·60
Fat	40·61	25·97	29·15
Starch	20·55	24·28	12·88
Sugar	Nil	22·76	1·41
Nitrogenous matter ...	17·87	8·38	16·53
Fibre	3·76	1·86	5·24
Ash	5·91	1·49	4·58
Acid equivalent of ash	2·12	0·65	3·22
Calories	6·30	5·13	5·70

The acid equivalent of the ash is the number of cc. of decinormal sulphuric acid required to neutralise the ash of one gramme of cocoa. This is determined to find out whether alkali has been added in the manufacture. It will be noted that in samples A, B and D this value is high showing that alkali has been used in these. The term calories represents the number of kilogramme calories evolved by the combustion of one gramme of the cocoa. A, B, C and D were all powders of the variety known as soluble cocoa. In preparing this article a considerable percentage of the fat is removed by pressing between hot rolls and the cake is then ground to a fine powder. Soluble is an incorrect term as the treatment only renders the cocoa more easily suspended in hot water. E was in tablet form, and as will be seen is much higher in fat content than any of the others; evidently only a small portion of the cocoa fat has been extracted in this case. This product is a cruder article than the powders but makes a palatable beverage when made with sugar. It contains a certain amount of added starch. F is a sweet chocolate with added starch and sugar, the starch is added in the form of arrowroot. C was also submitted in tablet form; it shows up well on analysis, but lacks palatability. These three tablets are all about the same price and are much lower in that respect than the powders. On analysis the most economical is E, as if F is bought for use as a beverage the mine will pay a high price for sugar which can be added when the cocoa is being prepared in the compound. The beverage prepared from G was insipid, and on this account G must be considered as inferior to E. The theobromine in these samples was estimated, the highest value obtained was 0·76%; this was obtained in powder A. The percentage of the alkaloid varied from this value down to 0·31% in the sweet chocolate.

The third beverage I shall deal with is Kafir beer. The supply of beer to the native has been the subject of much discussion; this discussion has been chiefly on the effect this liquor has on the native. That question is outside the scope of this paper. The Mining Regulations Commission sent a circular letter round the mines asking several questions as to the control of Kafir beer in compounds. A question one would expect to be included in the list but which does not appear, is: "What is the average percentage of alcohol in the beer supplied?" Three per cent. of alcohol by weight is the maximum allowed, and in the samples analysed during the last few years in the Gold Fields Laboratory this limit has not been exceeded. The percentage has varied from one and a half to two and a half. As an experiment a sample was sent to the laboratory immediately after starting to brew, and the percentage of alcohol estimated periodically with the following results:—

After brewing	1 day	1·31%	alcohol	by	weight.
,, ,,	2 days	2·07	,,	,,	,,
,, ,,	3 ,,	2·55	,,	,,	,,
,, ,,	4 ,,	2·95	,,	,,	,,
,, ,,	7 ,,	3·70	,,	,,	,,
,, ,,	21 ,,	3·84	,,	,,	,,

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* Food and the Principles of Dietetics. Robert Hutchison, M.D. (Edin.), F.R.C.P., 2nd edition, p. 331.

This experiment shows the gradual increase in alcoholic content the longer the beer is kept. When the last sample was taken evolution of CO_2 had ceased and fermentation apparently was finished. According to this test the beer was below the maximum allowed up to four days after commencing to brew. Further experiments would be interesting to determine if there is a uniformity in the rate of increase of alcoholic content in the beer brewed in different compounds.

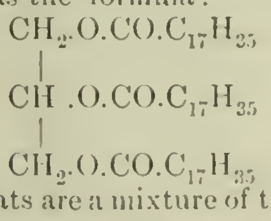
Before concluding this paper I shall give a few of the calorific values of the foods mentioned. The values are represented by the number of kilogramme calories evolved by the combustion of one gramme of the substance mentioned. These were determined by means of Mahler's bomb calorimeter.

Fat :	Beef suet	8·85
	Nut oil	9·21
Nitrogenous matter :	Albumen (white of egg dried at 100°C)	4·95
Carbohydrates :	Starch (dried at 100°C) ..	4·08
	Sugar ,, ,, ...	4 02

These represent the three alimentary principles

Mealie meal	4·02
Flour	4·08
Pea nuts	6·39
Jugo beans	4·08
Cocoa powder (average)	5·67
Cocoa tablet ,, ...	5 71

These values show the relative potential energy of these articles the analyses of which have been given in this paper. Pea nuts give a calorific value almost equal to a Middelburg coal. 6·36 kgm. calories is equivalent to an evaporative factor of 11·90 expressed as the number of pounds of water evaporated at and from 100°C by one pound of the nuts, the usual method of expressing the heating power of a coal. The suet and oil have the highest values, as might be expected from the constitution of these bodies, which are composed of compounds of glycerine with stearic oleic and palmitic acids known as tristearin, triolein and tripalmitin respectively. Tristearin has the formula :



$$\begin{array}{l} CH_2.O.CO.C_{17}H_{35} \\ \quad | \\ CH\ .O.CO.C_{17}H_{35} \\ \quad | \\ CH_2.O.CO.C_{17}H_{35} \end{array}$$

Triolein has $C_{17}H_{33}$ and tripalmitin $C_{15}H_{31}$ instead of $C_{17}H_{35}$.

Fats are a mixture of these in different proportions and, as will be seen, contain a large amount of carbon and hydrogen available for combustion. In the carbohydrates the ratio of hydrogen to oxygen in the molecule is 2 : 1, the same as is found in water.

From the analyses I have given of the various food stuffs I think the advisability of requiring some guarantee from the seller will be apparent and also that steps should be taken by the mine authorities to ensure that the conditions of the guarantee are carried out.

In calculating the nitrogenous matter the factor N × 6·25 has been used throughout for the sake of uniformity.

In conclusion the author wishes to thank the Consolidated Gold Fields, their consulting metallurgist, Dr. W. A. Caldecott, and the metallurgical chemist, Mr. A. McArthur Johnston, for allowing him to reproduce the figures quoted above.

The President : Before thanking Mr. Watson, I should like to say a word about the lamented death of Dr. Klein, who was the pioneer on this subject of native dietary, and who died this week very suddenly. He was a member of this Society for six years, and I think it is but right that we should record his death with sorrow. We have to thank Mr. Watson for many things, first of all for being one of the select little band of chemists who submit papers for our benefit, and for coming forward with this paper which breaks new ground even in this universal Society and which must have involved a very great deal of work. It will be most useful to medical men in giving them South African data instead of European data to work out dietaries from. I think this is another case in which the laboratory with which Mr. Watson is connected has acted as a pioneer in improving the efficiency of things on the mines.

Mr. A. F. Crosse (*Past-President*) : As regards mealie meal I was engaged in special work in connection with a sanitary scheme on the Ferreira Mine some years ago, and I could not help noticing that a very large quantity of mealie meal, which had passed through the bodies of the Kafirs, had not been properly digested. That would show there was a loss of efficiency in the mealie meal. Now as regards Kafir beer. Kafir beer is not supposed to have a greater percentage than 3% of alcohol. Yesterday I made a careful estimation of the alcoholic contents of some ordinary Cape wine. It amounted to 11·7% by volume, so that if the Cape people are going to supply our natives with Cape wine, it will be much over the 3% in Kafir beer.

THE MINE DUST PROBLEM.

By Dr. J. L. Aymard.

The vast importance of this subject must be my only excuse for so soon introducing it again before the Society. Mr. Penlerick read a very able paper before you recently, and dealt somewhat fully with the special ventilation arrangements made on the mines under his management. There are, however, a certain number of persons who whilst approving the general principles which he has adopted are opposed to many of the details. One would be apt to gather from Mr. Penlerick's paper that the system which has been adopted on the East Rand Proprietary Mines was meant as a model for other mines, because little reference was made to extensive and very successful schemes elsewhere.

The fact is that in very few cases can one mine be taken as a model for another. What appears to be more required is a standard series of approved principles, adaptable, with modifications, to the mines in general. It is rather with the idea of promoting further useful discussion on the subject than any personal desire to air my own opinions that has prompted me to include the subject of ventilation in this paper

Mine ventilation has received a good deal more attention than the average man has any idea of, and there are mines working to-day under able management quite devoid of expensive artificial aid which can show results far superior to those, which somewhat advertise their supposed superiority. Coal mine ventilation has reached a considerable pitch of perfection, but the fact is that systems suitable for coal mines are only in part applicable to the gold mines here. This is one of the difficulties which have to be met and overcome. Coal mining ventilation experts can hardly lay claim to superiority over our local experts until they have made provision for the prevention of the wholesale destruction of lives which we are so frequently and so forcibly reminded of. I think ventilation is too often looked upon as a mysterious science, whereas it is only one which needs the collection of facts and the application of suitable remedies. The working details may safely be left, with confidence, in the hands of our engineers, but useful practical points and hints, particularly in relation to dust and gas, may and should come from many of you.

Again, some people look upon ventilation as the sole solution to the problem of the prevention of miners phthisis, nevertheless it is a question whether if ventilation were applied as some would wish, the very persons whom it was meant to benefit most would not be the first to raise objection, more particularly if the individuals in question were subjected to the slightest inconvenience or smallest possible delay in their work. Personally I consider ventilation of such importance that in spite of the title of this paper I will deal herein first with certain questions which arise out of ventilation.

It occurs to me that there is considerable danger in applying undue force to existing air currents without fully considering all the attendant consequences. The first precaution, which, it appears to me, it would be wise to take is to see that the passage of dust-laden air throughout a mine shall not cause varying danger to the workers by reason of its varying velocity. I should like to hear this important point discussed, namely, what is the rate at which air can travel to the bottom of the downcast shaft so as to give the maximum efficiency. In its passage downwards the problem of supplying fresh air to the side workings, as well as the objectionable possibility of carrying on foul air to the various lower workings, require very full consideration. I will now discuss these two points in greater detail :

(1) *Ventilation of drives.* — Supposing, for instance, in the downcast shaft there are a number of development drives only connected here and there, then the rapidity of the passage of the downcast air past the mouth of the drives must have a point of maximum efficiency in ventilating these side drives. The rapid passage of air down one tube placed at right angles to a smaller one which is blocked, can only have the effect of tending to produce a slight vacuum at the working face, and this may account for the stifling atmosphere one so often experiences in these places before the drill is at work. The platforms erected at the various levels of the mine form, to my mind, an excellent arrangement for intercepting a portion of the *downcast* air and carrying it to the drive entrance—but think what happens where the reverse takes place in the upcast shaft. The station platform is surely, here, a most effectual barrier to any passage of air to the drive, and unless the air enters from some other source, ventilation can only take place by eddying. I have a suggestion to make which I think involves no serious mining objection, would not be at all costly, and could be easily introduced. It is, that a few feet, between the entrance to the drive and the platform, should be left open as much as possible : a few rails would make an open grid over which the track could be laid, strong iron netting in place of boards between the rails would make for safety, strips of board could be then placed either side for walking on, and at a lower level some further arrangement to catch any falling stones. This would allow the upcast air to freely enter the drive.

The manner in which air enters any drive from a shaft is rather interesting. I think I am right in saying that the air enters along the hanging wall and the return air with foul gas along the floor ; this is in an upcast shaft. This experiment on a small scale can easily be reproduced. Upon placing a horizontal division in the upper portion of the drive entrance this order is reversed, and the more this division is extended towards the shaft naturally the greater the entrance of air. The question as to whether this fact can be put to practical use is open for discussion

2. *Diffusion of Foul Air.*—The second great point, which appears to me to need consideration, is the practice of leading foul air through the whole workings of the mine before it reaches the surface. Surely this is wrong in principle? Not only is it quite against common sense laws, but I have no doubt that in many cases it can be avoided without any particular expense by some simple system of short circuiting, and I would like to hear from engineers and others whether, if not practicable in developed mines, such a system could not be introduced in opening up new ones. Without seriously interfering with the system of natural ventilation existing on many mines, I think the more extensive use of return foul-air pipes along the roofs of drives would be very beneficial, especially if these pipes were connected with a foul air shaft, placed above, beneath or alongside the ladder way, and led above into a chimney. The practice adopted on one mine of placing the ladder-way inside the foul-air shaft I think is decidedly open to objection. The ladder-way is a very important road for human traffic, and in almost continuous use, and the air should be, if anything, better and not worse than elsewhere.

I am extremely surprised to find in one or two cases where elaborate systems have been installed that so little attention is paid to the question of friction. In one mine in particular the hanging wall of the drives presented the appearance of a stalactitic cave. Not only were all the elements of danger from falling roof freely exhibited, but a very general appearance of bad mining; the very irregular nature of the undressed roof set one thinking what effect this element of friction must have on the efficiency of the huge fan working on the surface. Generally speaking, my experience goes to show that the higher and cleaner a drive the better the ventilation throughout the whole mine. Again, in some mines one finds almost at all hours the stations resemble the quay alongside an off loading cargo boat—rails, timber, dynamite and pumps forming a most efficient barrier to the free passage of air. In other mines with different systems the stations one finds to be clean, well lit, and as deserted and empty as a tube station after the departure of the last train. These mines strike one as always the best ventilated and yet without any extensive artificial aid. It is impossible to leave this subject without referring to the remarkable ventilation system installed on the Village Deep. The huge fan alone, and the great attention which is paid to cleanliness and efficiency, are in themselves worth a visit to this mine.

On examining the evidence given before the Mining Regulations Commission one is struck by the varied opinions on the subject of ventilation. Mine captains and shift bosses appear to champion the efficiency of their particular mines, whereas miners and others appear to take a different view. Professor Yates states: "That natural ventilation has many things in its favour, but the low increment of temperature in the Rand mines, while otherwise an advantage was a drawback to it." Dr. Irvine lays special stress on local ventilation of winzes and raises, and I think all of us are agreed that, with regard to the former, the most stringent rules must be enforced. The accumulation of gases in winzes with the very frequent fatal results must be firmly dealt with. Lastly, turning to the recommendations of the Commission:—

"(*a*) The ventilating current from the downcast shall be suitably split at the entrance to every working drive. Such entrance to be provided with regulating doors, so constructed that the openings for the passage of air can be varied as required. After passing through the workings the air shall be led as directly as possible to the main return air-way." Were these suggestions to be forced upon mines in general I believe the result would be little less than disastrous. Who is to be told off to regulate the doors at all times of the day and night? I think the idea in general is unpractical and unscientific. The latter portion of this paragraph is undoubtedly right in principle, but details would have been advantageous.

"(*b*) The courses of the supply to all working faces and of the foul return air from such places should be kept separate wherever necessary." This is exactly what I strongly advocate: in that foul air shall not be carried round the mine but short circuited and carried up a foul-air compartment.

"(*c*) Disused stope drives and other workings shall where necessary be completely closed in." I do not see the force of this recommendation at all. The arguments given as to these places being huge reservoirs where fumes and gases accumulate I think are entirely wrong. My own experience would rather lead me to look upon these places as extensive and very useful reservoirs of cool air, which during certain periods of the day when surface currents are slack, may serve a very useful purpose. To recommend them to be completely closed in is absurd. Stopes require periodic examination, and purely for sanitary reasons I would recommend a barbed wire barrier with padlocked gate.

After this somewhat wandering preamble, I pass to the dust question proper.

The *medical aspect* of the dust problem remains to-day very much as it did ten years ago. Still recently a few important matters have arisen. Dr. Cumpstone, an Australian Government doctor, in a recent report holds the view that the very finest dust merely circulates in the blood and does no material harm, and that it is a coarser dust which by its constant passage through the air channels causes the fibrous growth. I think those who have interested themselves in the subject would be inclined to agree with this. Recently, in investigating the the dust microscopically and also lung specimens kindly lent by Dr. Mitchell, a doubt has arisen as to the quantity and position of the quartz. Dr. Moir discovered that only certain portions of the supposed quartz showed with polarised light, and that the pigmented larger masses generally taken for quartz dust presented none of the characteristics of quartz, but he has considerable doubt as to whether the particles which shine in polarised light are really quartz. This very interesting point you will see here in a pair of beautiful micro-photographs taken by Dr. Moir. The question as to whether the pigmented mass is not merely the ordinary surface oxidised red sandstone will have to be carefully gone into, because it is around this mass that the growth of fibrosis appears to be most active. There is no reason whatever why we should consider red sandstone harmless, and provided sufficient enters the lung it is clear that it is quite as injurious as coal dust or any other of the dusts which produce a precisely similar disease to miners' silicosis. It certainly is a fact that the addition of the quartz element to the other accounts for its intensity.

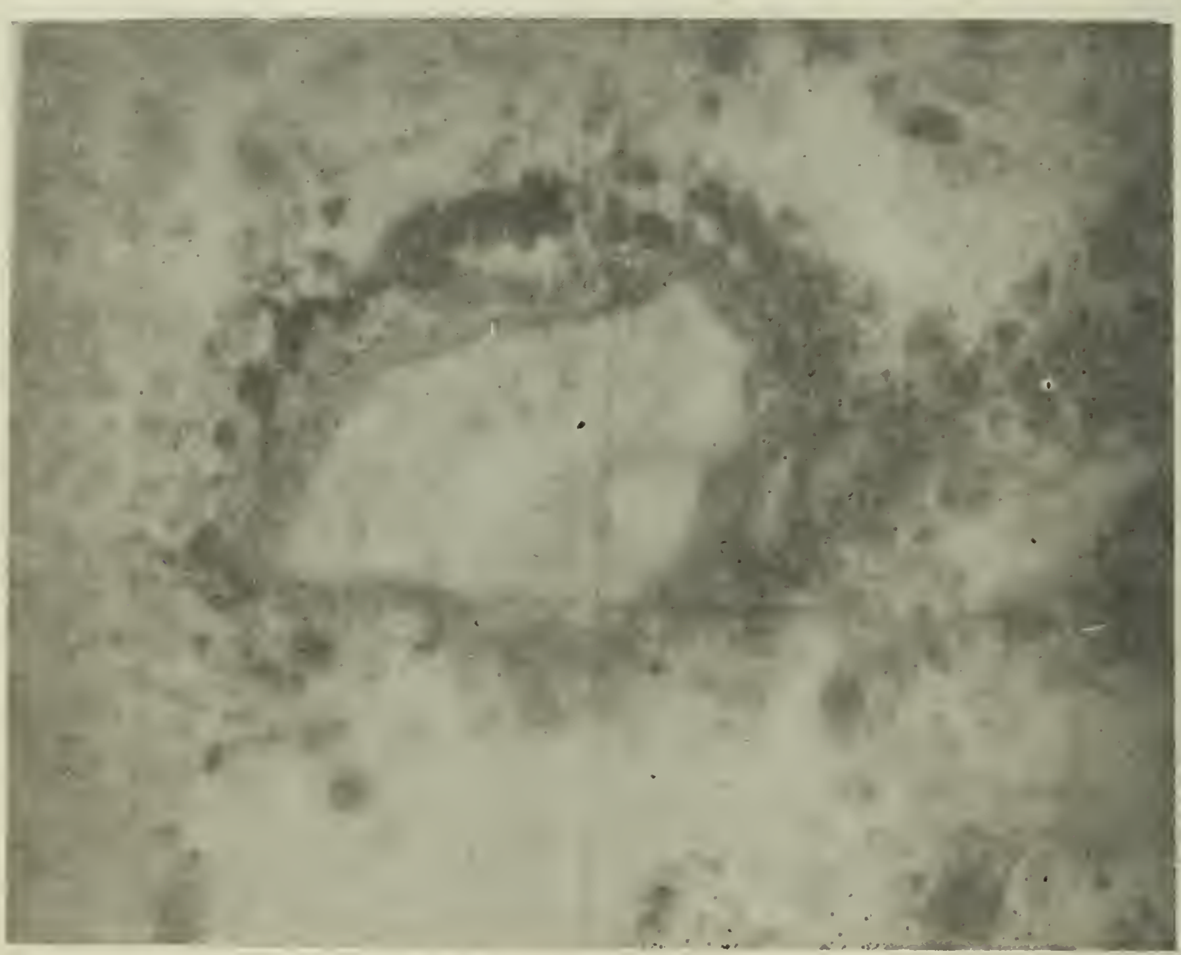


FIG. 1.—Ordinary Illumination. Nodule Silicosis in Lung, ×380 diameters.

I fear we have all taken the pathology of this disease as definitely settled, whereas it is quite clear it is not so. Considerable work will have to be done in this direction before we can definitely say that the acuteness of the disease is in any way proportionate to the amount of quartz to be found in the lungs after death. It may be that it is only the passage of the comparatively coarser particles which by constant friction causes the disease, or it may be that the pigmented portions are really accumulations of the very finest particles, or that both combined really contribute equally to the disease. It is also quite clear that before there can be any possible hope of really scientific treatment, the absolute pathological cause or causes must be established.

At the same time I do think a vast deal more could be done for those in the earlier stages of affection. Breathing exercises either induced or voluntary together with suitable inhalations and careful regulation as to exercise, carried out in some central institution, would, I feel, do a great deal towards checking, if not curing, many cases of early disease. Such an institution, if successful, might then be extended in connection with the various hospitals along the reef. This would meet the class of cases of men continuing work and those who would not for many reasons care to enter the final sanatorium. That silicosis does not, in all cases, proceed to a fatal ending, I think, is clearly proved by the considerable number of men who have survived. I, for one, do not accept the view that silicosis once started must invariably take a fatal course. The question raised as to the grade of dust which does the most harm is, I think, only of importance medically, when one takes into consideration the fact that all the grades in question easily float in the air.

Test for dust in mine air.—As the sense of smell is one of the best tests for certain gases so we may say that the sense of sight is the best test for dust. Complicated systems of trapping air lead to many objections, and are quite unnecessary. For all practical purposes the acetylene gas lamp against a dark surface will demonstrate the presence or otherwise of very minute quantities of dust, and when the mine-air in general is so purified that little is shown by this method, we may take it that silicosis is no longer being produced. Dust has to be reduced to a certain minimum; after this the ordinary arrangements of the air passages of the human lung will suffice to deal with it.

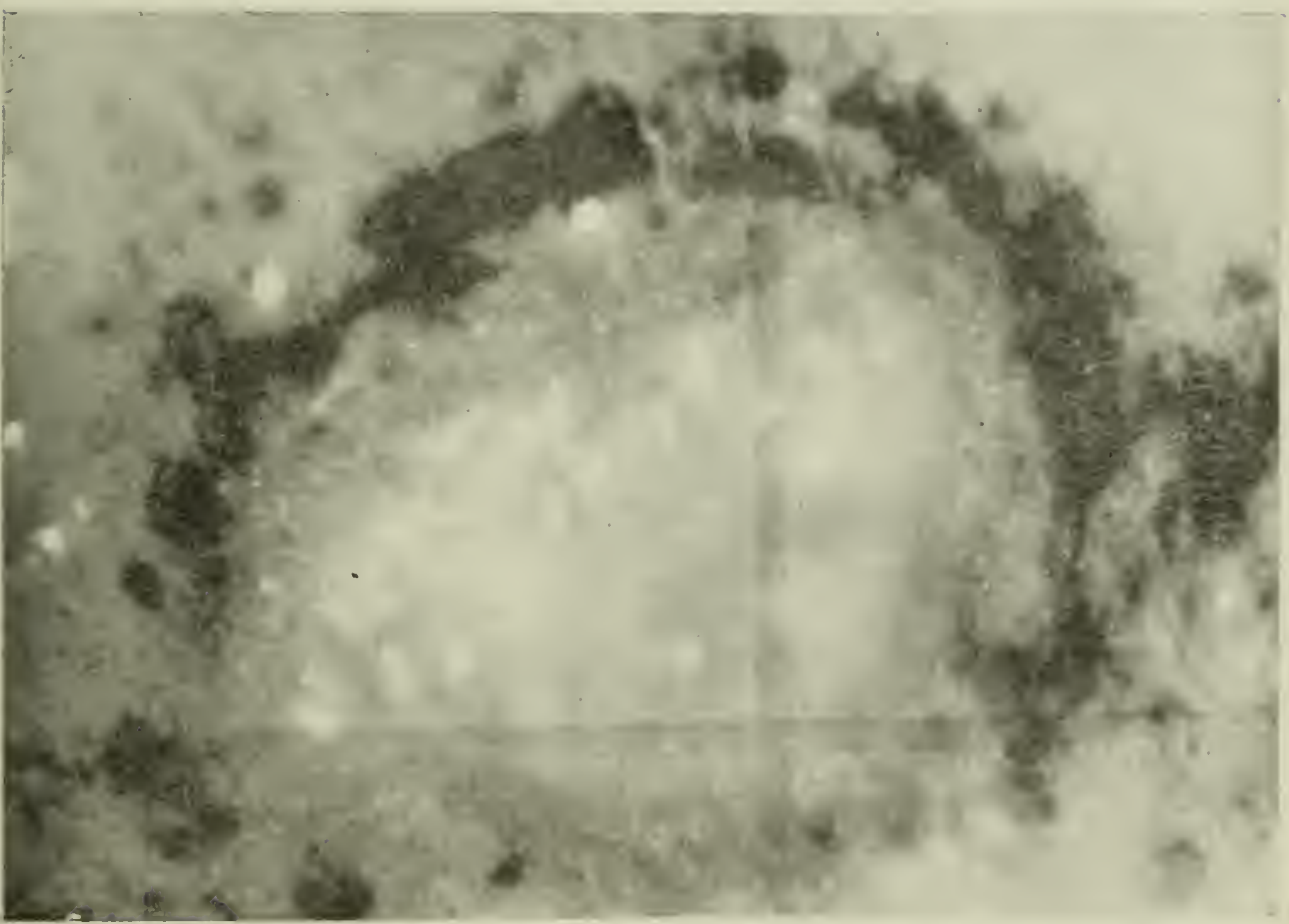


Fig. 11.—Nodule of Silicosis in Lung, Polarised Light. 680 diameters.

Dust produced by blasting and shovelling.—The practice of wetting drives and stopes must not be by any means diminished. There are different systems on almost every mine. The object to be aimed at should certainly be the more even distribution of the water, and this can only be attained by using proper means. I should like to draw your attention to two simple forms of sprays. The small one is in use with excellent results on the City and Suburban, and the larger one on the Village Deep. The small spray is made from an ordinary gas plug, and the larger one from a water plug, but both on the same principle. There is nothing to clog or get out of order, and the maximum surface can be evenly wetted with a minimum supply of water. The small type is suitable for the drives and raises, the large type for stopes. I was surprised to see how easily the hanging and foot wall could be watered at the same time, and the large area damped down with so little water. The regulation as to the extent to which a drive should be wet down *before blasting* might easily be extended to fifty feet, considering that in most cases it is quite easy, standing still, with a hose, to wet a drive properly 25 feet each way. I see no reason why, say once a week, the whole of the drives in a mine should not be systematically washed down with some suitable disinfectant. This could be easily done, and the benefits which would result would be well worth a trial. In perusing a recommendation of the Commission for the use of water in allaying dust, I find that its use is advocated in rising, and that the discomfort so produced may be obviated by the miner wearing a mackintosh. Had the Commission recommended that the individual should strip, the same might have been seriously considered, but the idea of a mackintosh in a stifling place like a rise strikes me as absurd, as also another recommendation to the effect that a jet of water should be played over the collar of a hole, quite regardless of the fact that the dust comes out principally below the drill, and that water playing above the drill, would quickly block the hole.

Prevention of Dust produced by Drills.—Six months ago when I made the statement that the dust so produced could not be properly destroyed by sprays or atomisers, I do not think the same was taken seriously. Our worthy President, I think, is himself now quite satisfied that a spray projected against falling dust leaves a good deal of the finest harmful particles quite free. The simple expedient of keeping the working faces wet does infinitely more good than any attempt at directly spraying the falling dust. The average miner *will not* take this simple precaution of wetting the face, although no one knows better than himself the great advantages which can be gained.

My system of catching and destroying the dust at the drill is now so generally known that I need not refer to it here.* The difficulty of "collaring" the hole I despair of overcoming by mechanical means, not so much because of the difficulty itself, but on account of the indifference of the average miner. I propose, however, getting over the same in another way. The miner may or may not wear a respirator—if he wishes to kill himself no one can entirely stop him—but we can prevent him killing others in the mine. During the short process of collaring the hole the Kafirs should be induced to wear respirators of a suitable type. The dust produced can be prevented from reaching the other workers by intercepting a wet sacking baffle at some distance back from the drill. I sincerely hope that those interested in any of the other numerous attempts to destroy the dust from drills will bring any practical scheme forward and promote discussion. Provided the main object is not delayed I would willingly give way to, and accept any system which was generally considered to be better than my own. The fact that a successful inventor ought to reap a reward of several thousands a year in royalties would be welcome to me, but I may here add, that, the only reason I did not myself take a commercial attitude was that I knew perfectly well the difficulties would be so greatly multiplied, that the object I had in view, namely, the relief of suffering humanity at the earliest possible moment, would be very seriously delayed.

I regret to say that though there are many intelligent miners on the reef who readily accept means of relief, the majority remain as indifferent and careless as ever, thinking of little or nothing but their monthly cheque. This class of man is himself the main obstacle to all reform. Fines are useless ; nothing short of risk of imprisonment will bring him to a sense of his duty to himself and those depending on him. The dust produced by blasting may be greatly lessened by proper watering before blasting, and not after. I propose, also, that a systematic introduction of wet sacking baffles in various parts of the mines be inforced, which will to a large extent catch the finer dust. The objection that it may to some degree interfere with ventilation in general is I think without any serious foundation. These baffles may even be placed horizontally in the upper portion of a drive and thus serve a double purpose. The object, after all, is not to circulate enormous quantities of air, but *sufficient*, in a pure state, free from dust.

The single shift.—Undoubtedly the universal introduction of the single shift principle would be of great benefit. The mine, like any building inhabited by a large number of individuals, should have a period free from occupation in which the air can be replaced. The important point is, that if, with such a system, blasting at all hours is still to be allowed, then the whole idea will be nullified.

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* This *Journal*, August, 1910

The firing of the fuse.—This very important question has been dealt with previously, but at the last meeting was treated in a somewhat frivolous manner. Fifty pounds (more or less) of burning dynamite daily in a mine is no laughing matter, and yet this is not very far from correct. Lighting fuse with burning dynamite is primitive and wrong in the extreme, and yet when other methods are suggested some members make rude reference to their grandmothers. Mr. Laschinger suggested a blow-lamp, and another urged electric firing. Of course, we all knew that what is wanted is an efficient and harmless fuse lighter. There are several in the market, but one hears very mixed opinions of the same. When one recollects that the going out of the lighter, perhaps, only once in a thousand times, may mean the loss of the miner's life, one realises the importance of the subject. Personally, I see no objection to electric firing in drives, rises and winzes. The question to the contractor is merely one of price. I understand that on an average it costs about eight-pence for dynamite. If all the fuses cut as usual were brought together and inserted in some form of Bickford igniter there would be no difficulty on the score of price. Cheap leads could be easily carried along a drive, and the firing take place from the station; as to whether this would be practicable in extensive stopes I must leave others to decide.

Failure of Water Supply.—The total absence of water despite the Regulations at the present time in some mines in certain places calls for more than passing comment. I do not blame the manager: I blame the directors, and I strongly recommend the experiment of a short term of imprisonment without the option, for the managing director of any company which wilfully disregards the law. Let a law be passed that any company not supplying the necessary water shall be punished in the above form, then you will find the law will be obeyed. No director will ever go to prison, because he won't run the risk. Anyhow the time has arrived when a stringent law must be enforced. The commonest causes of water failure are burst and injured pipes and hoses. Iron pipes rapidly corrode. If the pipes were wrapped in tarred strips of canvas and covered, they would last probably for years. The tank system, with ball-valve, and properly covered, as existing on the Village Deep, is one of the best on the Rand. The hose pipe is very commonly at fault; the half-inch pipe, used extensively in some mines, and particularly on the Nourse, is quite sufficient for all purposes, and has obvious advantages over those of larger diameter. The failure of the hose pipe is due to several causes, principally the unsatisfactory armouring, and also the connections. In many mines the iron cocks are being replaced by brass, with very satisfactory results. The mine water is seldom sufficiently strongly impregnated with sulphuric acid to affect brass, though in most, the action on iron is very marked indeed. Hence connections should either be of brass or iron, tinned. The armouring of most hoses is their weak point, and this refers to both air and water ones. The wire is galvanised and the zinc is rapidly eaten away, the wire rusts, loses its rigidity, and is stamped into the outer coat. The outer coat of most water hose pipes has a very large percentage of sulphur which under these conditions very soon cracks, outside water travels along the canvas lining which quickly rots, with the result that the pressure from within soon completes the destruction. Hose pipes seldom if ever degenerate from within. The remedy is simple. All hose-pipes should be armoured with wire coated only with pure tin, and not galvanised. This suggestion was made to Mr. Laschinger for a recent specification and I believe he embodied it in the same. The extra cost would be trifling. In specifying for water hose-pipes the percentage of rubber to sulphur should be clearly stated, and the point remembered, that cheap pipes are the dearest in the long run.

Respirators and their Uses.—Time will not allow me to deal fully with this question. Publicity has already been given to the new type introduced by me. The baffle principle is new, and I believe it will serve a useful purpose. The fitting of all drills with some system of dust destruction will take time, and in the interval deaths will be increasing. *The respirator is only meant as a second line of defence, but its use should be encouraged both in whites and blacks. It will always be useful during the process of hole collaring on piston drills.* Whether boys shovelling will be able to endure the slight extra exertion remains to be seen. That it is possible for those occupying such posts as shift bosses to wear the same I am now firmly convinced. One of my instructors continually wears his respirator and his work is more than equal to the above, and he tells me he finds no inconvenience even when compelled to breathe deeply.

Why so many young men are to-day falling victims to the disease is a question I will try to answer. The shift boss of to-day is generally a man of gentle birth, often an athlete, and a man of study. He is called upon to perform the most arduous duty in a mine, that of climbing up innumerable stopes, rises and ladder ways. In the process he breathes very deeply, and thus, more rapidly inhales the deadly dust into his lungs. More shift bosses should be appointed and their duties lessened, otherwise, unless they are able to adopt precautions, their death rate will shortly be appalling. I may add my personal conviction, that the shift boss of to-day is called upon to perform very nearly twice the amount of work of his predecessor of six or seven years ago; whether he does so as thoroughly is not for me to judge.

My work in this direction is entirely due to the suggestions recently made before this Society by Mr. Hildick-Smith.

The Problem Politic.—Probably not in the mining history of the country has a more serious position had to be faced. Public opinion demands that miners' phthisis, being a preventible disease shall cease—and yet the vast majority of the very beings most interested offer a stubborn resistance. Are the miners absolutely insane or rendered devoid of thinking power by reason of their calling, or what is it? I will try and portray the very near future. The position to day, is that exactly what most of us feared would happen, is taking place. The miner (principally the contractor with three drills on a bar) refuses almost point blank to accept and use the means placed in his hands for preserving his own life and those around him, because it is a little extra trouble and might, at the very outside, in the course of a whole shift, delay his work five minutes. The above, to a lesser degree, applies to a large extent to those using percussion drills, but the worst offenders of all are the foreigners. The miner thinks, rightly or wrongly, that he knows his strength with regard to his personal value and indispensableness, and acts up to it. With regard to the percussion drill he is somewhat conscious of his weakness; true, my system is slightly less trouble in this instance, and there is the additional fact, that he can be far more easily replaced.

The Government threatens to step in, not directly, by forcing the use of these life preservers by the men themselves, but indirectly, by introducing a Compensation Act, and making others do the work.

Firstly let us take the Government attitude. Would effective compulsion direct be possible by means of introducing a large staff of efficient inspectors or by any system of espionage? The former would be impracticable, and the latter unsatisfactory. Only those intimately connected with mining difficulties can possibly realise the objection to the former, and inspectors *versus* men backed by boys must end in failure The hundred and one ways of communicating the fact of an inspector's approach, and the vast difficulty in rapid movement in a mine, would, also make for failure. This, is clearly realised, hence the Government's only hope is to throw the whole responsibility upon the mining houses. Now what is the position to day! The majority of the mining houses have made, and are making, a determined effort to cope with the difficulty, but are met with the stubborn opposition of the men above referred to. Realising this deadlock, I endeavoured to get a meeting of all the mining houses, with the object of continuing, and making a general move, so that the men would be treated universally and compelled to adopt the means placed in their hands for the preservation of their own lives, and those working in the rest of the mines. As long as each mine loyally forced its men to adopt some particular system then it would be useless to threaten removal on the part of the men, because they would be subjected to similar treatment everywhere. The houses replied, that, though willing to meet, they could not see that any good would result. This is quite inexplicable considering the fact that their own engineers have stated to me that a general move in the above direction was the only remedy.

It is only possible to read between the lines, and draw the conclusion that some few of the houses, though agreeing to the principle, could not be trusted to carry it out, and by laxity, would obtain the best men. Loyalty between the mining houses can no more be expected than the same in any other branch of commercial life. Still it is the remedy and one which will have to be recognised if force is to be used on their side. The question of compensation, which will almost immediately be brought forward, will be fraught with innumerable difficulties and endless trouble. How will it affect the miner? And here I cannot but admire the pluck of the Government; that is, if it has really seriously considered the consequences.

I have no authority to speak for the mining houses, nor have I any but my own ideas of what they will do, but as a reasonable man who can see as far through a wall as the rest, I think they will say something to this effect: "We have tried all we can to induce you men to save your own lives, both now and in the past; by your folly you have largely brought about the present position, we are compelled to provide for you in the future, therefore, you must pay your share." Now, what will be the cost? Unless some very elaborate system of exclusion is introduced and all the present men replaced by healthy ones, and these healthy ones only permitted to mine for a comparatively short period, the cost must be almost prohibitive for mining. If, on the other hand, the present system is to proceed, and the men are to be simply compensated, what does it mean? It means that every miner must, for the purpose, be looked upon as sooner or later liable to contract the disease. Hence his pay must be reduced by one-half. This is the fact that miners and the Government have to face. Under all these circumstances the miner should consider his position before it is too late.

He must remember the history of some, of the industrial disputes. He will recollect the cotton troubles, which resulted in the introduction of machinery to the disaster of the workman. Industrial upheavals almost invariably lead to industrial reforms, and the latter, in many cases, means mechanical replacement of the labourer. The after-dinner speech of a certain prominent engineer the other night should make the miner reflect; there are other possible ways even of drilling holes in rock which would produce no dust. If the miner means to place himself in direct opposition to his own interests, then science backed by wealth will step in and beat him, and he will get no pity or sympathy, and will have no one to blame but himself.

The President: I have pleasure in moving a hearty vote of thanks to Dr. Aymard for the paper, especially for the great trouble he has been to in making this useful model of a mine which will be of service for future experiments. There is plenty of controversial matter in this paper and I anticipate that there will be much interesting discussion on the subject. I have to show you also a couple of lantern slides. The first displays particles of street dust moderately magnified. In addition to red veld dust and white angular quartz, there are several large organic particles. The second is the finest dust from Dr. Aymard's device for catching dust at the drill. It was obtained by shaking it through the pores

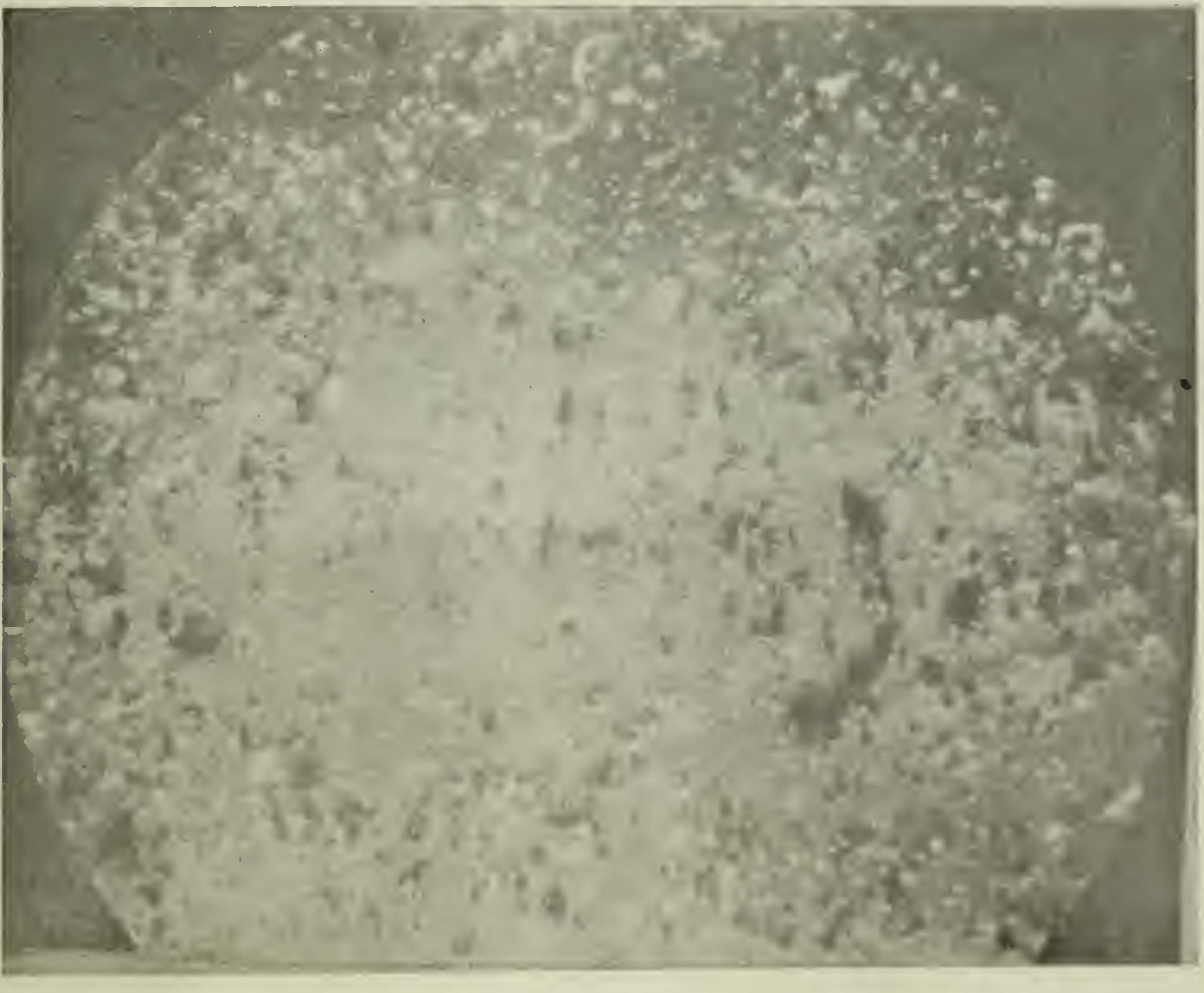

FIG. III.—Street Dust deposited in a fortnight's exposure on slide in Laboratory, 25 feet from ground. × about 150 diameters.

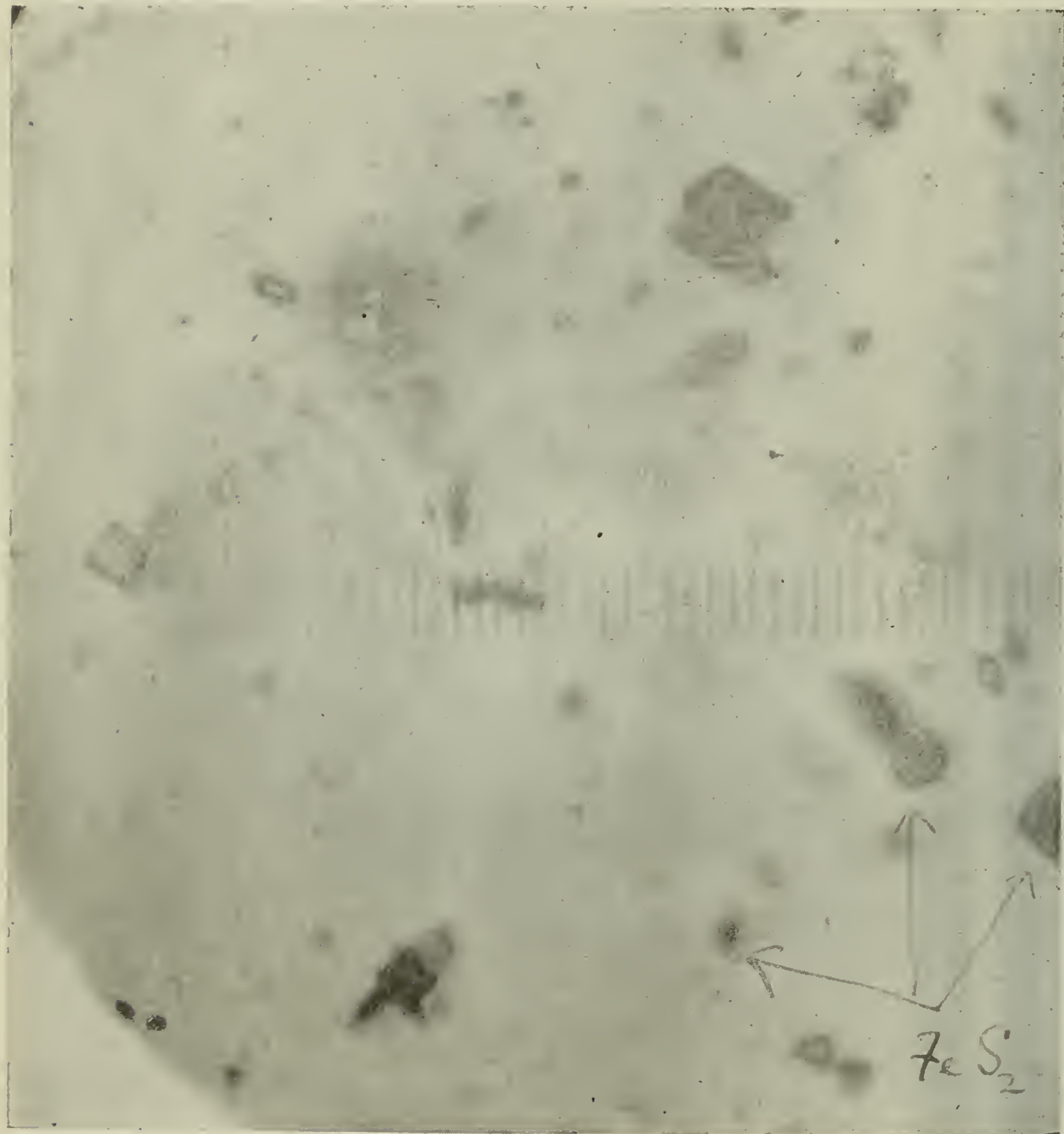

Fig. IV.—Finest Dust from Drills, showing pyrite. × 330 diameters. Scale-division = 0·0075 mm.

of a linen bag, nevertheless it shows a remarkably large number of pyrites particles, which may have a special bearing on its action on the lungs.

Dr. Aymard (*Associate*): I did not know that Dr. Moir was going to bring up street dust again. Here (referring to the strike of Tramway employes) we have an example of what has to be done to bring the Town Council to a sense of its responsibilities. We have spoken frequently about the Municipal Crusher House at the Ferreira, but I suppose it will be necessary to throw a cart-load of dust into the Town Council meeting before they will take the slightest notice, and at all events we shall certainly be put to considerable trouble before they will take notice of it. The men at the Municipal Crusher House are living in this dust, but nothing is done to keep the dust down. One may refer to the matter in the public Press, but still they take no notice. It is only when violent measures are taken that they suddenly arouse themselves to their responsibilities.

THE DISTRIBUTION OF PULP FOR TUBE MILLING.

(Read at September Meeting, 1910.)

By G. A. Robertson (Member).

REPLY TO DISCUSSION.

Mr. G. A. Robertson (*Member*): I might have said a good deal in an attempt to prove the merits of a device which did not in the knowledge of the author exist. Instead of so doing, and stating with reference to present methods adopted for distributing the pulp between tube mills that "none of those methods could be considered satisfactory under all operative conditions," I thought some one might have challenged my statement, or perhaps shown a more efficient design than the one put forward in my paper.

I have to thank Prof. Stanley for pointing out the omission, showing how the pulp got into the distributor, and have also to thank Mr. Fraser Alexander for his frank admission of the difficulty in pulp division. This was my excuse for writing the paper.

For a small tube mill plant the wedge principle gives wonderfully good results. The accompanying sketch shows the wedges in the bottom of the launder which, being secured with bolts, slide in the slot shown, and with adjustment will divert a portion of the coarse sand in the direction required without necessarily touching the division gates. Coarse sand always flows in the middle of the stream where the velocity is greatest, and this is the chief reason why the equal division of pulp cannot be affected by the mere use of gates placed equi-distant in the stream. Bearing in mind also that the coarsest sand is nearest the bottom of the launder, and can be divided up by the method shown in Fig. I., (which was designed by Mr. T. W. Travers), the division of the moisture is also made easy by the adoption of an adjustable suspended gate dipping diagonally into the stream. The wedges in Fig. I. are 2½ in. deep, and the depth of the stream is about 10 in.

Having alluded to another distributor which could be adapted to large tube mill plants,* a modification in that design can be seen in the Kidney pulp distributor.† The author there points out that when a concentrating table is stopped the shut off segment in the distributor overflows to the next adjacent segments and consequently overloads two concentrating tables. This is perhaps not of much importance for a few minutes stoppage of a concentrating table, but as we have at times to shut down a tube mill for 24 hours a further modification in these designs is necessary. Either of these designs, however, could I believe be made to give more accurate results than the one shown in my paper, at least in so far as giving any desired and different pulp loads to the various tube mills. To these methods which give me the "keynote" I now show the following improvements relative to the discharge from the distributor. The advantages here are: Equal spacing of the gates to represent any number of equal segments without interference with the discharge, greater simplicity in operation, an

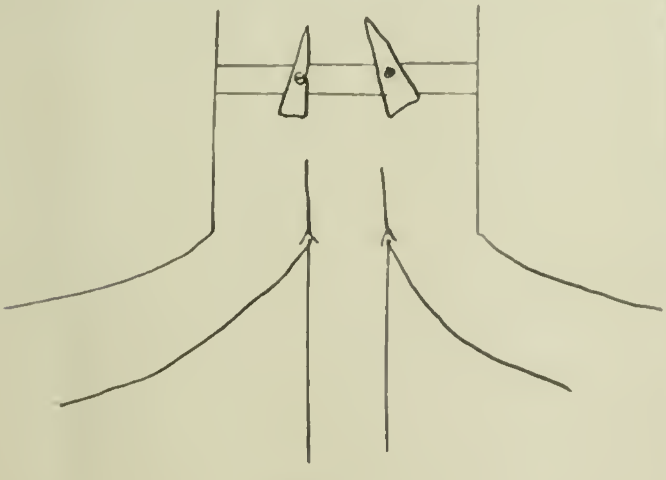

Fig. I.

FIG 2

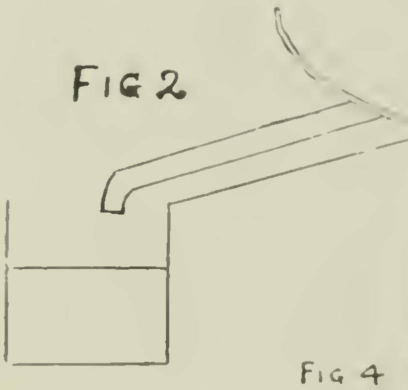

FIG 4

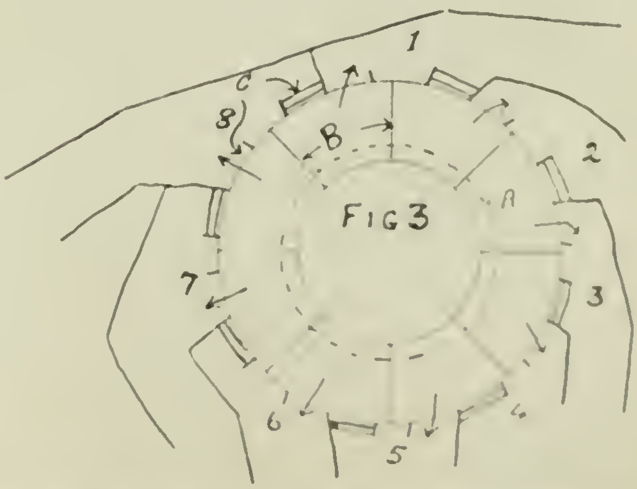

FIG 3

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* See this Journal, Vol. XI., July, 1910, p. [illegible]

† See Engineering and Mining Journal, Nov. [illegible], 1910, p. [illegible]

adjustable fall for the gravitation of pulp from the distributor, and a full view of the pulp flow by the shiftsman. Fig. II. shows the half sectional elevation of the revolving barrel. Fig. III. the sketch plan of the circular launder and division gates. The pulp from the revolving barrel, or Butters' distributor, should be delivered on the inside of the launder about the line A.

It will be seen that each launder leading to the tube mills has two discharges C in the side of the distributor, either one or the other being available for the segment share of pulp flow under all conditions. To divide the pulp in accordance with the grinding capacity of each tube mill, gates (Fig. IV.) should be placed on the top of the equally spaced gates, and being adjustable to any angle will decrease or increase the length of the pulp flow arc. The actual difference between a discharge in the bottom of the distributor and a discharge through the side of same is that the former is a fixture for all grades of pulp to flow through, whereas the latter by heightening the division gates also allows the heightening of the discharge lip, thereby giving a greater fall for an increasingly coarser pulp.

Mr. H. W. Jane points out[*] that a tube mill when newly lined with 6 in. liners absorbs 85 h.p. and after 66 days 125 h.p. It would seem ridiculous to give these tube mills the same pulp load. The use of the top gate, Fig. 3, is obvious. This gate is easily fixed by a wood wedge against the periphery of the launder. The method shown, if properly applied, will give accurate results under all conditions. Other unpatented devices for pulp distribution can be seen in the *Mining Magazine*[†].

The fullest efficiency of any method of hydraulic classification, whereby single classifiers or a combination of classifiers act independently for each tube mill, and fed from one and the same head can only be seen to advantage when the stream delivered to such classifiers contains the like ratio of solids to moisture in all cases, irrespective of different tonnages of solids delivered to such classifiers. The proper adjustment of the underflows of the classifiers will then make all the overflows of the same grading, and also of the same assay value. Without amalgamation of the mortar box flow an increase in the moisture delivered to any classifier over that of the others, will carry in that overflow a greater proportion of the free gold, and this independent of the gradings being equal.

THE CLASSIFICATION OF TAILING PULP PRIOR TO CYANIDING.

(Read at October Meeting, 1910.)

By Edward H. Johnson, M.I.M.M. (Past-President).

DISCUSSION.

Mr. A. McArthur Johnston (*Past-President*): The very interesting as well as instructive paper submitted by Mr. E. H. Johnson brings forward one or two features which can well occupy our attention, and which have not only a technical bearing but also an educative aspect.

As it may interest many of our members I should like to place before them a few figures, which were obtained at the Simmer Deep[*] some eighteen months ago, and which present some features differing from those quoted by Mr. Johnson.

One of the main points elucidated by him is that during separation in the cones of the sand and slime, the richer portion of the finest sand, *i.e.*, pyrite, is retained in the sand charge where longer treatment is available than in the slime circuit. This advantage was long ago recognised at the Simmer Deep, and periodically tests and assays were made to determine the stability of such separation. I may here quote a typical example:—

	Per cent. by Weight	Pyrite (FeS_2) per cent.	Assay Value dwt.
Sand Original.—			
– 150 + 200	19·56	2·95	3·76
– 200	12·11	8·96	9·11
Slime Original.—			
– 150 + 200	5·98	0·27	0·82
– 200 (less washed slime)	35·41	1·39	2·46

The advantage of putting this low grade fine sand into the slime plant, where working costs are lower than in sand treatment vats, is evidenced by experimental further treatment of the residue slime, when it was found that no lowering of the undissolved gold value therein took place.

Dealing next with the rate of solubility of the gold in a sand charge when using a weak cyanide solution, you will have noted that with the collector vat as treatment vat, Mr. Johnson's figures and chart show that the maximum value of the percolating solution entering the precipitation boxes is attained between the 60th and 70th hour after starting to fill the vat, whilst this solution does not get to 0·10 dwt. till between the 185th and 200th hour, or approximately 8 days after filling began. These figures are extremely interesting as showing what may be expected when dealing with this quality of sand and with this treatment. As indicative of what happens with a lower grade proposition and with a transfer from a collector to a leacher as part of the treatment, I would place before you the following chart (see Fig. I.).

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[*] See discussion on "Grading Analysis and their Application," this *Journal*, July, 1910, p. 19.

[†] See Mining of Lead Silver Ores, June, p. 446; also Flow Sheet of Miami Concentrator, Sept., p. 221.

[*] For description of S.D. plant, see this *Journal*, May, 1909.

Grading of Charge.—

	I.	II.
	per cent.	per cent.
+60 (·010 in.)	14·7	15·9
−60+90 (·006 in.)	36·5	40·7
−90	48·8	43·4
	dwt.	dwt.
Assay value of original ...	2·88	2·84
Assay value of undissolved gold at time of transfer ...	1·00	1·16
Assay Value of Residue.—		
Total gold	0·52	0·51
Undissolved gold	0·50	0·50

From this chart it will be noted that the maximum value of the leaching solution is reached shortly after transfer, viz., 40 hours after starting to fill, whilst in 88 hours this solution is coming off steadily at 0·10 dwt.

It will be noted also that at the time of transfer, 30 to 40 hours after milling, close upon 60% of the total gold in the charge, or 70% of the soluble gold has been sent to the precipitation boxes.

The strength of sodium cyanide solutions used was from 0·026% to 0·030% KCN for filling and leaching before transfer, whilst about 100 tons of ·100% KCN solution was pumped on after transfer, the remaining solutions being weak and,

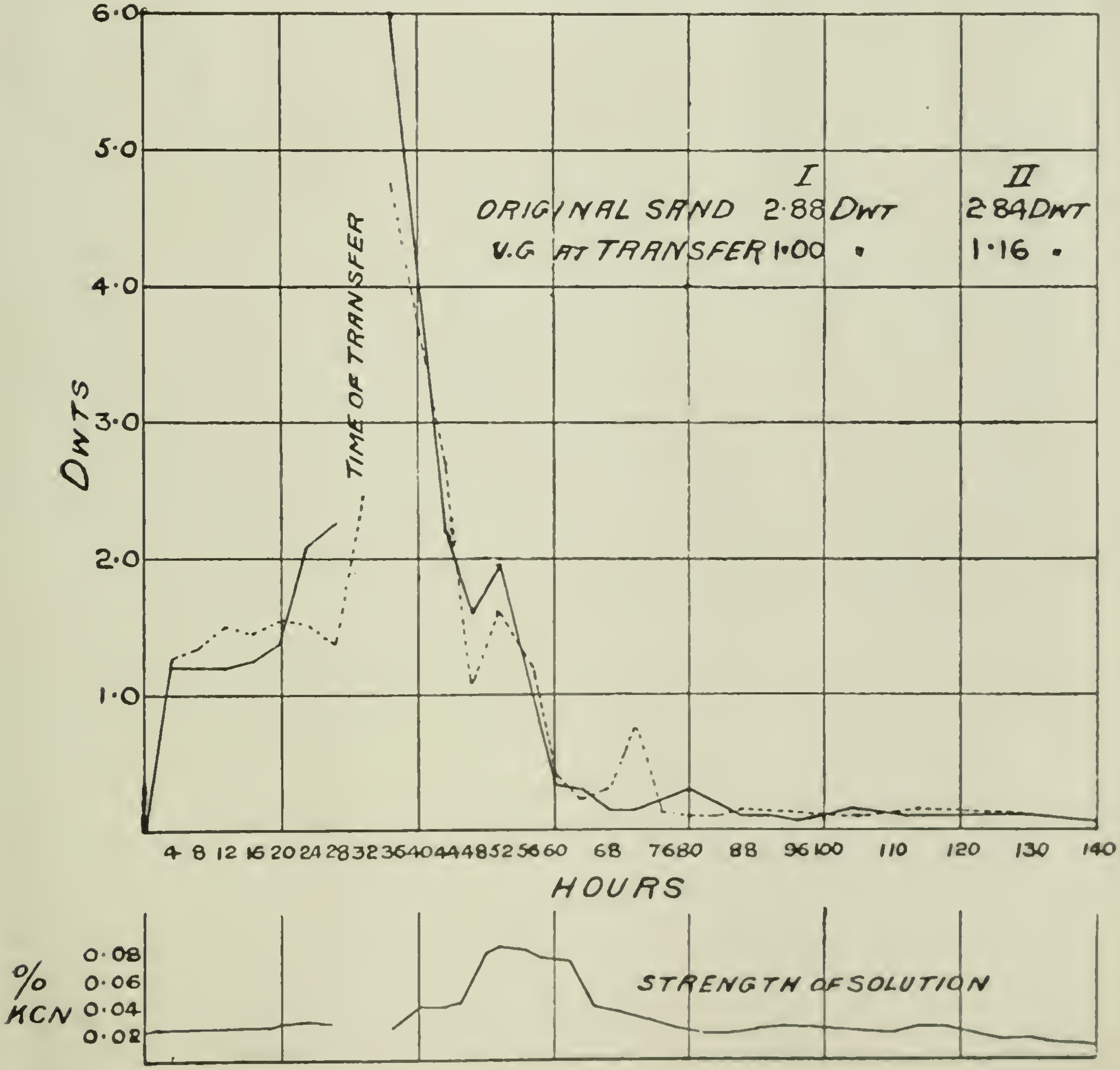

Fig. I.

finally, slime washes. The lower chart (Fig. I.) depicts the strength of cyanide solution after leaching.

In order to ascertain whether length of treatment or increased extraction would be affected by dropping the transferred sand into solution, I made a series of tests as nearly as possible similar except that in A the strong solution was pumped on the sand after this had been levelled, whilst in B, the solution was placed in the vat before starting to fill with sand. This, by means of the Blaisdell distributor, was scattered fairly evenly and owing to its freedom from slime, no difficulty was experienced in leaching.

Herewith are seen the comparative results :—

TABLE I.

A.—*Ordinary Filling.*

Number of Charge.	Assay Value Original.	Assay Value Residue Total Gold.	Assay Value Residue Undiss. Gold.	Last Drains.	Moisture.
	dwt.	dwt.	dwt.	dwt.	
(1) 598	2·88	0·66	0·65	0·08	—
(2) 600	2·92	0·56	0·54	0·10	—
(3) 602	2·74	0·56	0·55	0·05	12·5
(4) 604	2·60	0·56	0·54	0·06	—
(5) 605	2·80	0·53	0·52	0·07	—
(6) 607	2·76	0·57	0·55	0·08	—
(7) 610	2·76	0·56	0·54	0·05	—
(8) 612	2·78	0·64	0·63	0·06	8·0
Averages	2·78	0·580	0·565	0·07	10·25

			per cent.
Average grading	+ 60	...	13·35
	− 60 + 90	...	45·67
Extraction (based on U.G. value)...			79·67
Extraction (based on T.G. value) ..			79·13

B.—*Solution Filled Vats.*

Number of Charge.	Average Value Original.	Tons Solution Added.	Assay Value Residue Total Gold.	Assay Value Residue Undiss. Gold.	Last Drains.	Moisture.
	dwt.		dwt.	dwt.	dwt.	
(9) 597	2·80	50	0·68	0·66	0·09	—
(10) 599	3·08	75	0·56	0·55	0·10	—
(11) 601	2·60	75	0·52	0·50	0·06	—
(12) 603	3·00	100	0·56	0·55	0·08	—
(13) 606	2·64	100	0·56	0·55	0·09	—
(14) 608	2·92	100	0·55	0·54	0·08	—
(15) 609	2·88	100	0·60	0·58	0·14	8·0
(16) 611	2·72	100	0·58	0·56	0·06	9·5
Averages	2·83		0·576	0·561	0·087	8·75

			per cent.
Average grading	+ 60	..	13·27
	− 60 + 90	...	45·30
Extraction (based on U.G. value)...			80·17
Extraction (based on T.G. value) ...			79·64

TABLE II.

Comparison of solutions draining from leaching vats filled in ordinary way and in strong cyanide solution.

A.—*Filled in Ordinary Way.*

Charge numbers ...	625	633
Tons	745	745
Original average value	2·56 dwt.	2·88 dwt.
Solution in collectors	560 tons	775 tons
1st 4 hours' leaching	3·06 dwt.	3·70 dwt.
2nd ,,	2·67 ,,	2·12 ,,
3rd ,,	1·80 ,,	2·35 ,,
4th ,,	1·20 ,,	1·54 ,,
5th ,,	1·06 ,,	1·26 ,,
6th ,,	0·56 ,,	0·77 ,,
7th ,,	0·40 ,,	0·60 ,,
8th ,,	0·32 ,,	0·34 ,,
9th ,,	0·26 ,,	0·26 ,,
10th ,,	0·12 ,,	0·26 ,,
11th ,,	0·11 ,,	0·20 ,,
12th ,,	0·10 ,,	0·18 ,,
13th ,,	0·08 ,,	0·12 ,,
14th ,,	0·15 ,,	0·15 ,,
15th ,,	0·10 ,,	0·12 ,,
16th ,,	0·05 ,,	0·08 ,,
17th ,,	0·05 ,,	0·06 ,,
18th ,,	0·03 ,,	0·05 ,,
19th ,,	–	0·07 ,,
20th ,,	—	0·06 ,,
21st ,,	—	0·05 ,,
Last drains ...	0·06 ,,	0·06 ,,
Total solution added	1,530 tons	1,597 tons
Residue total gold ...	0·55 dwt.	0·64 dwt.
Residue undissolv. gold	0·54 ,,	0·62 ,,
Moisture in residue ...	9·3%	9·5%
Grading (residue) + 60	13·2 ,,	13·4 ,,
+ 90	49·6 ,,	41·6 ,,
Extraction (base on undissolved gold value	78·9 ,,	78·4 ,,

Though these results are not definite enough to draw incontrovertible conclusions from, it is significant that the average extraction in charges 9 to 16 compares more than favourably with that in charges 1 to 8. In addition, in Table II., a similar state is seen, and there is a tendency to the formation of the opinion that a slightly increased extraction is probable, though little gain in time can be expected.

These results would in general emphasise the advantage of dropping sand into solution in preference to having the solution percolate through the sand. Much greater improvement might be anticipated in cases where no previous solution of the gold had taken place as it must be remembered that such a treatment had been already meted out to this sand.

B.—*Filled in Cyanide Solution.*
(100 tons added before filling begins)

Charge numbers ...	624	634
Tons	745	738
Original average value	2·64 dwt.	2·96 dwt.
Solution in collectors	580 tons	565 tons
1st 4 hours' leaching	0·63 dwt.	0·71 dwt.
2nd ,,	1·00 ,,	1·06 ,,
3rd ,,	1·75 ,,	1·98 ,,
4th ,,	1·48 ,,	2·30 ,,
5th ,,	0·90 ,,	2·79 ,,
6th ,,	0·44 ,,	2·83 ,,
7th ,,	0·21 ,,	1·64 ,,
8th ,,	0·20 ,,	1·22 ,,
9th ,,	0·11 ,,	0·58 ,,
10th ,,	0·05 ,,	0·47 ,,
11th ,,	0·03 ,,	0·30 ,,
12th ,,	0.08 ,,	0·22 ,,
13th ,,	0·07 ,,	0·16 ,,
14th ,,	0·05 ,,	0·12 ,,
15th ,,	0·05 ,,	0·14 ,,
16th ,,	0·06 ,,	0·06 ,,
17th ,,	0·06 ,,	0·10 ,,
18th ,,	0·07 ,,	0·08 ,,
19th ,,	—	0·07 ,,
20th ,,	—	0·06 ,,
21st ,,	—	0·06 ,,
Last drains ...	0·05 ,,	0·04 ,,
Total solution added	1,600 tons	1,665 tons
Residue total gold ...	0·56 dwt.	0·60 dwt.
Residue undissolv. gold	0·55 ,,	0·58 ,,
Moisture in residue ...	10·5%	9·7%
Grading (residue) + 60	13·0 ,,	12·6 ,,
+ 90	43·8 ,,	40·4 ,,
Extraction (based on undissolved gold value	79·2 ,,	80·4 ,,

In conclusion, I would like acknowledge the courtesy of Mr. H. S. Macgregor, manager of Simmer Deep, Ltd., in allowing me to place these figures before you.

Mr. G. O. Smart (*Member of Council*): The paper under discussion deals with an operation which has passed through the various stages mentioned, as its importance became more generally realised.

Elsewhere, as in Mr. C. W. Merrill's earlier methods at the Homestake,* the operation is facilitated by using an ample supply of water which may be permissible under local conditions. The remarkable classification of sand and slime at the East Rand Proprietary Mines with little water shows a distinct advance, since 88% of the −200 mesh product in the crushed ore is separated in the 50% of slime collected, whilst almost as good results, as shown later, can be obtained in practice without washing water. The advantages of a clean separation of sand and slime are obvious in yielding a uniform freely leachable sand,* and increasing the slime tonnage with its low cost of treatment, and in addition the sand residue is left in an excellent condition for ready drainage in mine stope filling.†

On this last ground alone the more extended use of good classification is likely to receive a strong impetus.

The Simmer and Jack installation‡ differs from that at the East Rand in having no washing cones, thus saving fall; the slime overflows eight filter table cones to the return sand cones *en route* to the slime plant. With the double sand treatment so perfectly washed a sand is not required as at the East Rand where single treatment in the collecting vats without transfer is practised. The Simmer and Jack table cones, as also the cones for sand filling are fitted with two diaphragms, as shown in Fig. I., which yield the maximum tonnage of sandy underflow, and help to prevent any tendency for the fine sand to pack hard in the upper portion of the cones. The following are typical gradings of the Simmer and Jack sand and slime:—

Sand (58%). *Slime* (42%).

GRADING.		Per cent. by Weight.	GRADING.		Per cent. by Weight.
Mesh per Linear Inch.	Aperture Decimal of 1 in.		Mesh per Linear Inch.	Aperture Decimal of 1 in.	
+ 60	·01	11·8			
+ 90	·006	31·4			
+ 200	·003	43·1	+ 200	·003	2·0
− 200	·003	13·7	− 200	·003	98·0

The effect of the above clean separation resulting from cone classification for the continuous collection of sand has been marked in reducing the last draining to ·05 dwt. per ton, from a much higher figure. It has also permitted a satisfactory cyanide treatment of sand containing fine metallic gold due to the elimination of amalgamated plates in the stamp mill; eighteen amalgamated plates only, operating on the tube mill pulp, are

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* See Proceedings *American Institute of Mining Engineers*, C. W. Merrill, p. 590, Oct., 1903, Vol. xxxiv. "In fact, there is little doubt but that the importance of the most perfect classification possible will be recognised shortly as a vital consideration in the cyaniding of wet crushed ore; and metallurgists will not follow the old German practice of spitzkasten and spitzlutten, which are very imperfect machines as compared with a cone-classifier or sizer for separating granular from flocculent material. My judgment is that a scientific classification-system by which all the granular or angular material may go to the leaching vats, and all the amorphous portion to the slime plant, will in the future be a feature in designing a plant on which the greatest care and experimentation will be put, and the highest grade of technical skill utilised."

The average grading of Homestake sand in 1904 was + 100 mesh 34%, +200 mesh 33%, −200 mesh 33%.

* See this *Journal*, Vol. x., J. R. Williams, August, 1909, p. 54.

† See this *Journal*, Vol. xi., A. R. Hughes, October, 1910, p. 174.

‡ See this *Journal*, Vol. xi., September, 1910, p. 132.

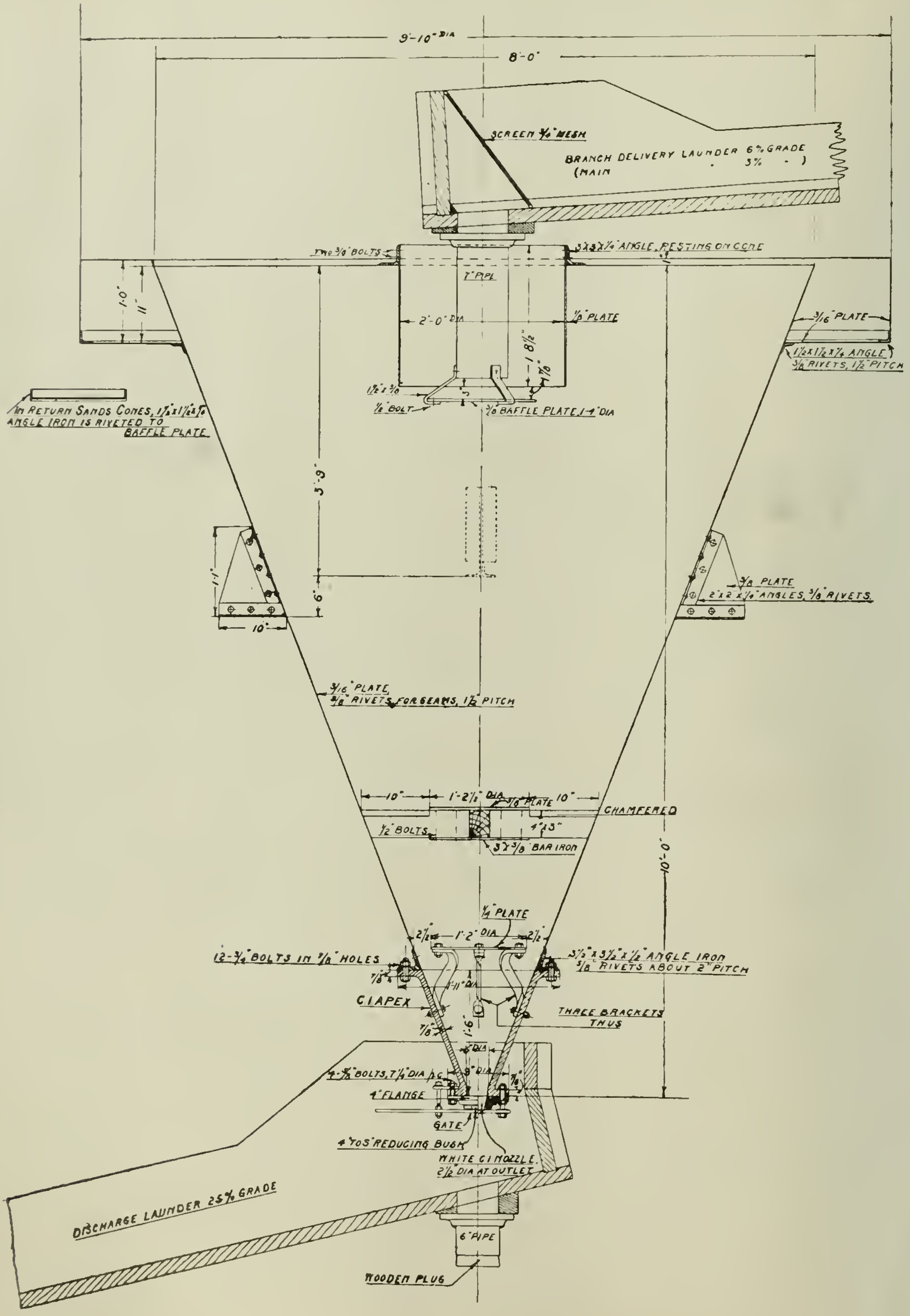

FIG. I.—Standard Diaphragm Cone as used for Continuous Sand Collecting Filter Tables and Underground Sand Filling (Simmer and Jack Proprietary Mines, Ltd.

now used, instead of the ninety-four formerly employed for the amalgation of mill tailing and tube mill pulp. The percentage of slime has also been increased from about 33·5% to 42% with the same fineness of crushing.

As the amount of comminution due to explosives and handling underground is not generally realised, the following grading which is much coarser than that given by the author may prove interesting.* The ore entering the mortar boxes of sixty stamps was sampled for two days. Before taking a sample the feeder was worked by hand until the mortar box was supplied with a surplus of ore, so that the feeder was not operated by the stamps during the taking of the sample. A special sampler was then placed between the feeder and the mortar box opening, the feeder being operated by hand until the required sample was delivered into the sampler. The operation of the feeder was then stopped until the sampler containing the whole of the ore feed while it was in position was withdrawn. The ore sampled was drawn from the deeper levels of the mine, and the sample when dried weighed 400 lb.

The grading of the + ·006 part of the sample was performed at the Simmer and Jack by wet methods the portion remaining on the screens being well washed, whilst the – ·006 gradings were done dry at the Consolidated Gold Fields laboratory.

Gradings.

Aperture.	Mesh per Linear Inch.	Percentage.	Cumulative.
in.			
+ 1		51·87	
+ 0·5		15·80	67·67
+ 0·280	3	9·36	77·03
+ 0·146	4	6·49	83·52
+ 0·097	8	3·41	86·93
+ 0·0633	12	1·98	88·91
+ 0·0328	20	2·28	91·19
+ 0·0225	30	1·28	92·47
+ 0:01	60	2·15	94·62
+ 0·006	90	1·10	95·72
+ 0·004	150	0·49	96·21
+ 0·003	200	0·67	96·88
– 0·003	200	1·13	98·01
Washed slime	—	1·99	100·00
		100·00	

This grading shows that 23% of the ore delivered to the stamps is – 0·28 grade (9 mesh per square inch), and can be advantageously fed direct to the tube-mill diaphragm cone classifiers, whilst 5·4% is already – 60 grade.

The results of a series of sand tonnage determinations by weighing 2,400 tons of sand in the trucks during discharge from the vats, with frequent moisture determinations, showed an average of 23·4 cub. ft. of sand per ton, which approximates to the author's figures of 23·5 cub. ft.

The results of the slime tonnage determinations, using an apparatus similar to that described by the author and illustrated in Fig. VII. of his paper, on an average show 34·33 cub. ft. of our settled slime to weigh a ton. This is higher than the 30 cub. ft found by the author, and is probably due to the slime being still partly derived from oxidised ore. In addition to the above, the cubic footage of slime per ton is liable to be influenced by the temperature, the depth of charge, the time available for settling charges, and the fineness to which the ore is crushed.

It will be seen from Mr. McLean's diagrams that quite an interesting mathematical problem is presented in calculating the number of cubic feet of settled slime pulp represented by samples from concentric rings in a conical-bottomed vat, since if of equal breadth these have very different areas and average depths. The following method of calculation is due to Mr. C. S. McLean, who carefully carried out all the determinations.

Mr. C. S. McLean: The apparatus used for the following tests was similar to tnat shown in Fig. VII. of the author's paper, with the following differences :—

(*a*) The central pipe was 9 ft. long instead of 20 ft.

(*b*) The cylinder, fitted with similar guides, was made of heavy iron piping, and weighed 200 lb. This was lowered into the vat with a block and tackle.

(*c*) As it is obvious that the apparatus could not stand upright if placed on the sloping bottom of the vat, a wedge shaped piece of iron plating was used to make a horizontal base. This maintained the centre pipe in a truly vertical position, thus enabling the cylinder to be lowered over it without disturbing the settled slime. The cylinder was accurately calibrated with water and carefully re-checked. From this data a table was made showing the cubic capacity at points ½ in. apart, from the bottom to the top of the cylinder.

It is quite evident that the average cubic feet of pulp per ton of dry slime cannot be derived by taking the arithmetical mean of a number of results obtained by sampling at equal distances between the centre and the edge of the vat. The reason for this is that a sample near the edge represents a larger volume of pulp than one near the centre. To reduce this error as far as practicable the vat was divided into a number of sections as shown in Fig. II. These sections were selected because, in an average charge the volume within a 4 ft. radius from the centre, is approximately equal to that at the periphery contained in a section between the radii of 24 ft. and 25 ft. Owing to the tower in the vat no samples could be taken nearer the centre than 4 ft.

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* See this *Journal*, Vol. xi., July, 1910, p. 21.

Two separate tests were carried out, number 1 being between the centre and the side of the vat near to where the charge entered, whilst number 2 was on the opposite side. The arithmetical mean of a number of samples taken within each section is assumed to give the average number of cubic feet per ton of dry slime in that section (see Fig. II.). The average depth of the charges in each test was determined from the mean depth of the pulp samples in each section (Fig. II.). From the above the volume in a charge, 1·95 ft. deep at the periphery, is calculated to be 6,446 cub. ft.

The following table shows the results obtained:

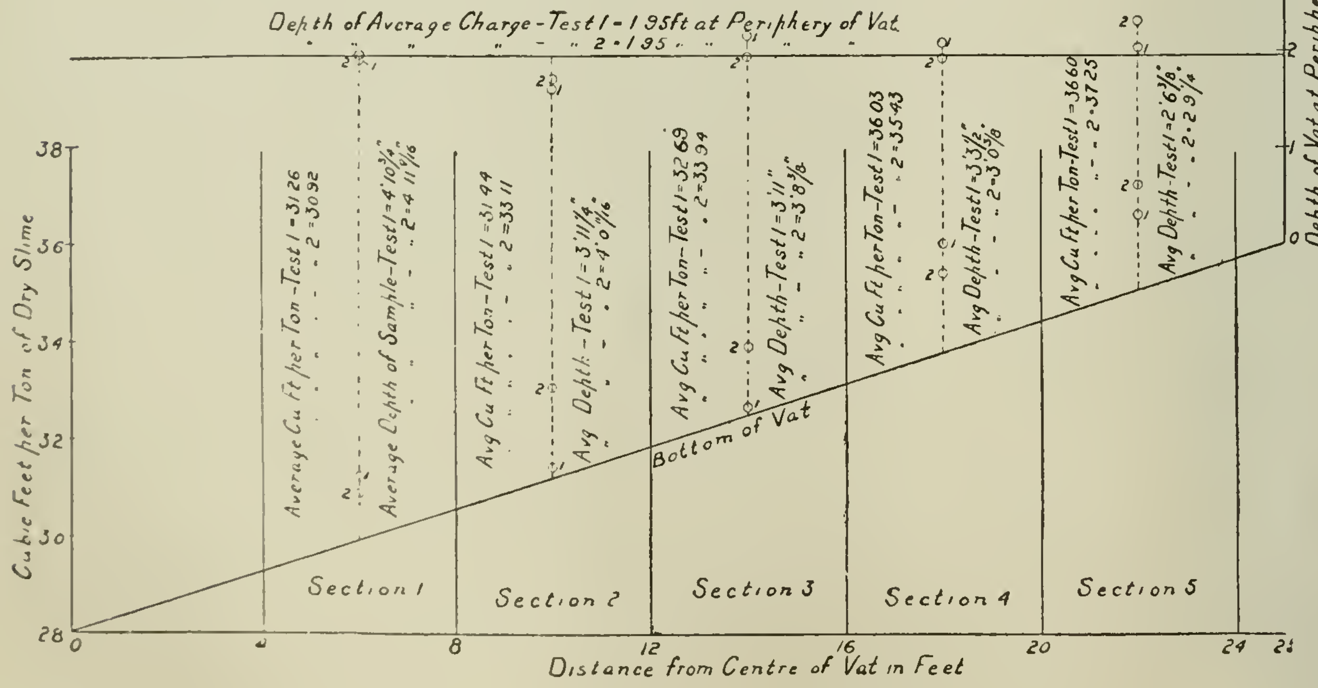
The mean cubic footage per ton was found by multiplying the average cubic feet per ton of dry slime for each section by the cubic feet contained therein, adding the products, and dividing by the total number of cubic feet in the five sections.

Test No. I.

Section number.	Position of section in feet from centre.	Position of average sample in feet from centre.	Average depth of sample.	Average time of settlement.	Per cent. moisture by volumetric method.	Per cent. moisture by gravimetric method.	Cubic feet per ton of dry slime by volumetric method.	Cubic feet per ton of dry slime by gravimetric method.
	ft. ft.	ft.	ft. in.	hours.				
1	4 to 8	6	4 $10\frac{3}{4}$	38	38·11	37·61	32·81	31·26
2	8 to 12	10	3 $11\frac{1}{4}$	$40\frac{1}{2}$	38·04	37·79	31·96	31·44
3	12 to 16	14	3 11	36	39·36	40·12	32·25	32·69
4	16 to 20	18	3 $3\frac{1}{2}$	36	42·53	42·69	35·99	36·03
5	20 to 24	22	2 $6\frac{3}{8}$	$39\frac{1}{2}$	42·71	43·33	36·04	36·60
Average				38				34·05

Diagram Showing
(a) Method of Dividing Charge into Sections
(b) Depth of Average Charge
(c) Average Cubic Feet per Ton of Dry Slime for each Section

Fig 2



Test No. II.

Section number.	Position of section in feet from centre.	Position of average sample in feet from centre.	Average depth of sample.		Average time of settlement.	Per cent. moisture by volumetric method.	Per cent. moisture by gravimetric method.	Cubic feet per ton of dry slime by volumetric method.	Cubic feet per ton of dry slime by gravimetric method.
		ft.	ft.	in.	hours.				
1	4 to 8	6	4	$11\frac{9}{16}$	36	38·15	36·98	31·72	30·92
2	8 to 12	10	4	$0\frac{11}{16}$	$35\frac{1}{2}$	40·17	40·09	33·15	33·11
3	12 to 16	14	3	$8\frac{3}{8}$	38	40·79	41·04	33·80	33 94
4	16 to 20	18	3	$0\frac{3}{8}$	38	42·29	42·00	35·58	35·43
5	20 to 24	22	2	$9\frac{1}{4}$	$37\frac{1}{2}$	43·14	44·02	36·63	37·25
Average					37				34·61

This is shown in the following table :—

Test No. I.

Section number.	Average cubic feet per ton of dry slime.	Area of section in sq. ft.	Average depth of samples in this section.		Volume of section in cubic feet approx.	Volume X cubic feet per ton.
			ft.	in.		
1	31·26	150·8	4	$10\frac{3}{4}$	738·3	23,079
2	31·44	251·3	3	$11\frac{1}{4}$	989·3	31,109
3	32·69	351·8	3	11	1377·9	45,043
4	36·03	452·4	3	$3\frac{1}{2}$	1489·1	53,652
5	36·60	553·0	2	$6\frac{3}{8}$	1399 8	51,234
Average	34·05	Totals	—		5994·4	204,117

Average cubic feet per ton $\frac{204,117}{5994\ 6}$ = 34·05 cubic feet.

Test No. II.

Section number.	Average cubic feet per ton of dry slime.	Area of section in sq. ft.	Average depth of samples in this section.		Volume of section in cubic feet approx.	Volume X cubic feet per ton.
1	30·92	150·8	4	$11\frac{9}{16}$	748·5	23,143
2	33·11	251·3	4	$0\frac{11}{16}$	1019·6	33,759
3	33·94	351·8	3	$8\frac{3}{8}$	1300·0	44,122
4	35·43	452·4	3	$0\frac{3}{8}$	1371 3	48,585
5	37·25	553·0	2	$9\frac{1}{4}$	1531·8	57,060
Average	34·61	Totals	—		5971·2	206,669

Average cubic feet per ton $\frac{206,669}{5971\cdot2}$ = 34·61 cubic feet.

The two tests give an average of 34·33 cub. ft. of pulp per ton of dry slime. This is plotted at a point 15·4 ft. from the centre which divides the volume of the average charge in two equal parts.

Fig. III. shows the number of cubic feet per ton at any position in the vat. This was obtained by plotting the average cubic feet per ton for each section on the cubic feet it represented, starting from the centre of the vat. The mean cubic feet per ton for each test was plotted on half the volume, and the line showing the cubic feet at different positions in the vat must pass through this point with a slope that shows equal areas above and below. Fig. IV. is taken directly from Fig. III. using distance from the centre as the base instead of volume.

The same vat was used for all determinations, and every charge passing through it for a period of eight weeks was sampled. The results of several specific gravity determinations showed an average of 2·7. This figure must be approximately correct, since the volumetric determinations based on a specific gravity of 2·7, are practically the same as those obtained gravimetrically. The curves and calculations are worked out from the gravimetric figures, the others being used solely as a check. The calculations were simplified by the coincidence of the depth of the average charge being the same for both tests.

The number of cubic feet per ton in test No. 2 is 0·56 cub. ft. higher than in test No. 1. This may be due to the following reasons :—

1. The time of settlement per charge was one hour less.

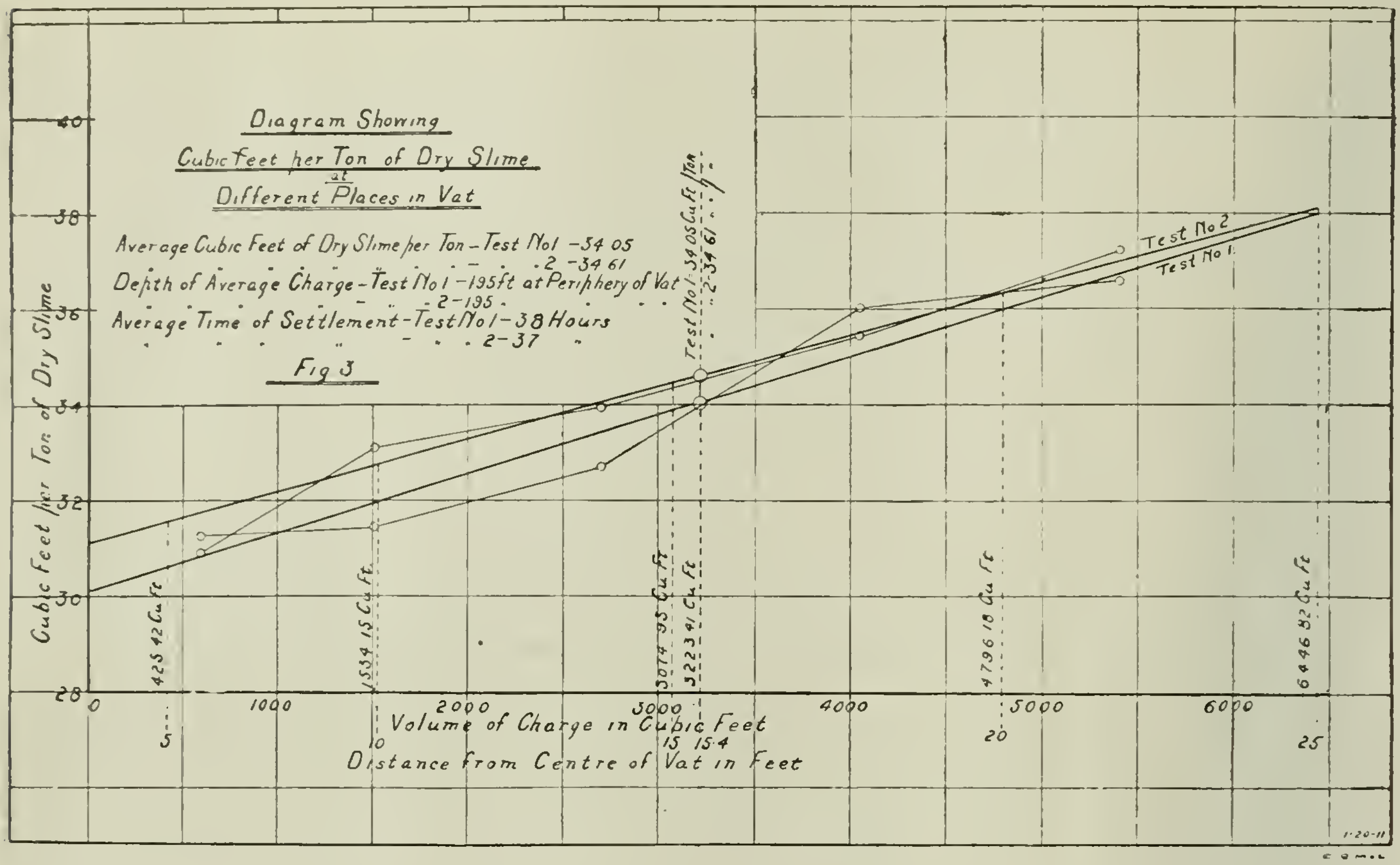
2. The gradings of the slime in the tests were as follows :

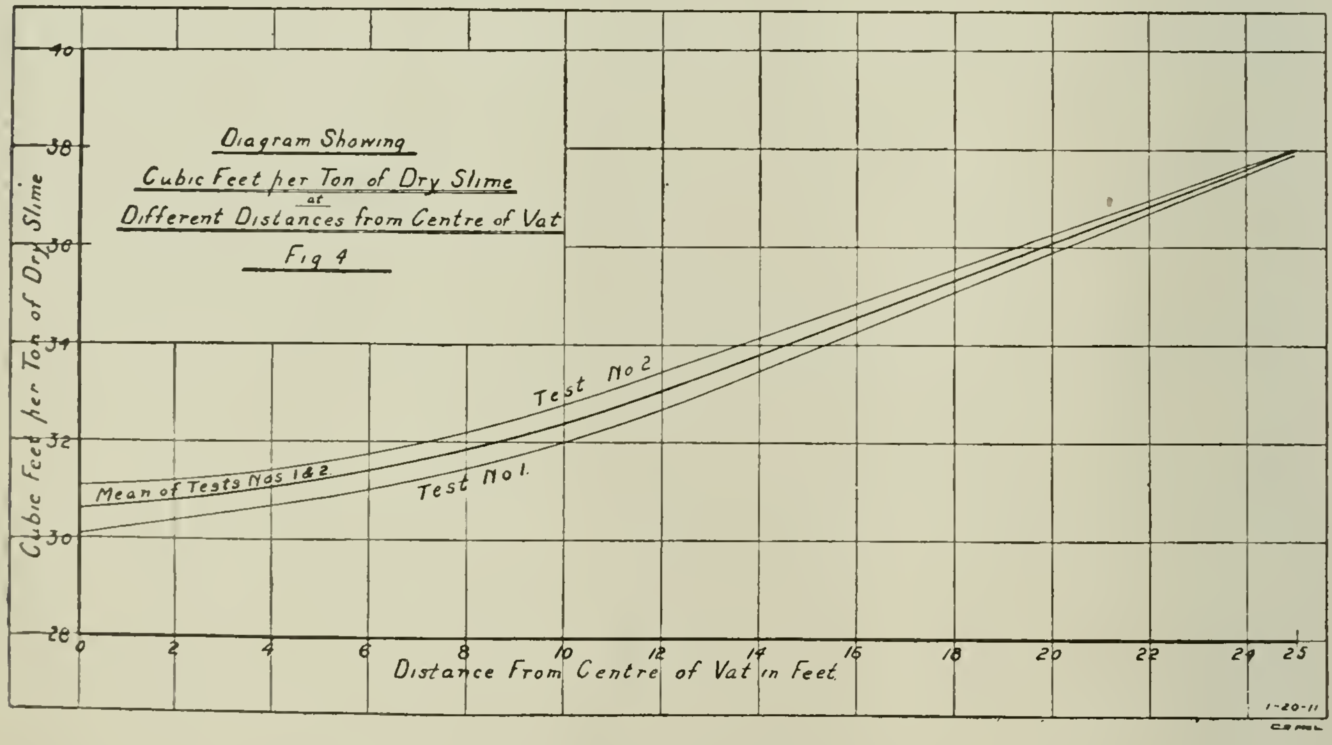
Test No. 1 + 200	2·94%
„ „ 2 + 200	0·87%

Since the + 200 settles towards the centre of the vat, test No. 2, having the smaller amount of + 200 grade would tend to give a higher figure near the centre and gradually approach test No. 1 at the periphery. Owing to a change in the methods of operating the classifiers, the + 200 grade was reduced from 2·94% in test No. 1 to 0·87% in test No. 2.

3. There might also be a larger proportion of + 200 in test No. 1, owing to the pulp being deilvered into the vat close to where the samples were taken.

It will be seen from Fig. III. that the average number of cubic feet per ton of dry slime is





found at a point dividing the volume in half, viz., 15·4 ft. from the centre in a 50 ft. vat with a depth of charge as shown. It is therefore apparent that an average of a number of samples taken at this point should give a result sufficiently accurate for a rough determination.

The following table, showing the distances from the centre of the vat at which samples should be taken to give the average cubic feet per ton of dry slime for different depths of charges, will be found very useful; as it enables an approximate determination to be made by taking three samples at one point only, instead of twenty-five at different points, which were necessary to obtain the data from which the following table is constructed.

Vat 50 ft. diameter.

Depth of charge including cone.		Depth of charge at periphery of vat.		Distance from centre where sample should be taken to obtain mean cub. ft. per ton.	
ft.	in.	ft.	in.	ft.	in.
2	9	1	5	15	0
3	0	1	8	15	2½
3	3	1	11	15	5
3	6	2	2	15	7
3	9	2	5	15	9
4	0	2	8	15	10½
4	3	2	11	16	0
4	6	3	2	16	1

In the author's reply an explanation of the following points, which are not clear, would be appreciated :—

1. In what position was the long 1 in. pipe when it was securely fastened to the platform to the centre of the vat (with the disc to which it is shown attached) resting flat on the sloping bottom of the vat as stated? If this pipe was not standing upright how could the cylinder drop freely over it?

2. What is meant by the columns, first section and third section under the heading "Form of Observations"?

3. It is also stated that any difference between the inside and outside measurements is due to loss of slime carrying 60% moisture, which is the result of many experiments. Was the loss of slime at 60% moisture due to leakage, and how was the 60% moisture determined? It may be pointed out that there was absolutely no leakage from the bottom of the cylinder in the tests carried on at the Simmer and Jack.

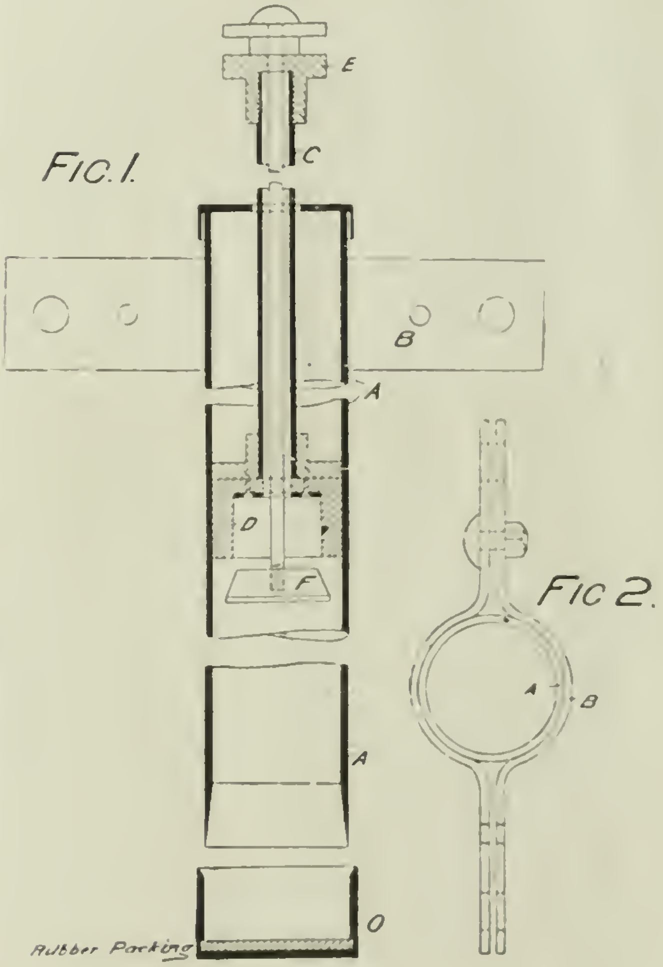
4. When sampling near the centre of the vat I found the inside measurements to be less than the outside, but this only occurred at this point, and never exceeded 1 in. The difference was probably caused by the brackets preventing some of the slime and heavy sand from entering the cylinder. Samples showing a difference of 1 in. between the inside and outside measurements were discarded. The bottom 6 in. of slime pulp near the centre contained less than 30% moisture and showed a grading of 37·8% +200 mesh.

Thanks are due to the general manager, Mr. A. W. Stockett, for permission to publish the above information obtained from the Simmer and Jack tests.

Mr. O. Tonnesen (*Member*): Since Mr. E. H. Johnson read his excellent paper on "The Classification of Tailing Pulp prior to Cyaniding," in which the apparatus then in use on the East Rand Proprietary Mines for sampling slime is described, I have invented and patented a new slime sampler, which has proved a success.

As from time to time this subject has been up for discussion in this Society, a brief description of my sampler and its use may perhaps be of interest to some of its members.

Fig. I. is a longitudinal section, Fig. II. a horizontal projection of the apparatus. It consists of a tube A of 2 in. outside diameter, and bevelled off at the bottom to a cutting edge. The tube is suspended to a rope fastened to clamp B. The sampler is lowered into the slime. When the tube has reached the bottom of the slime tank, a ladder is lowered into the vat. The depth of the slime outside and inside the tube may then be accurately measured. By taking sufficient time in lowering the pipe to allow the slime to find its own level, the difference of outside and inside slime level is easily brought down to nil. The rod C is now put into the tube until the circular disc F reaches the solution, and then inserted about ½ in. further in order to make sure that the bottom edge of the rubber cup D is immersed in water. Screw E is then unscrewed until F wedges tightly against the rubber cup D forcing it airtight against the inside of the tube. On top of the slime contained in the tube the air pressure is now nil. The tube can there



fore be pulled out of the slime tank without disturbing the sample, the core remaining intact, supported by the outside air pressure.

If specific gravity of slime is, say, 1·8, atmospheric pressure = 12·3 lb. per square inch, then the greatest height of core supportable is approximately $\frac{12{\cdot}3 \times 2{\cdot}3}{1{\cdot}8} = 15{\cdot}7$ ft., and as the height of slime core is generally from 3 ft. to 5 ft. only, this leaves a large margin of pressure. When the bottom edge of the tube approaches the surface of the slime the opening of the pipe is closed by the lid G to prevent air from getting into the pipe. The sampler is now pulled up and the contents emptied into the drying dish by removing the lid G and lowering the disc F by screw E, whereby the atmospheric pressure is admitted above the core. The inside of the tube should be thoroughly cleaned by inserting a rod with a rubber washer, thus no moisture is lost.

A practical proof that no slime is lost is easily obtained by inverting the tube before emptying it and removing the lid. It will be found to be full of slime up to the brim, thus proving that no slime can have escaped. Neither can it be imagined that some of the heavier bottom core should be torn off at the first stage of pulling up and refilled later, because the outside pressure at the first stage is as much greater than the pressure at the last stage as the weight of the outside slime column.

This slime sampler has many advantages over the old method referred to in the October *Journal.* It was often argued, rightly or wrongly, against the old method that the centre rod might cause concentration during the circulation. Although the new sampler gives a smaller sample, yet accuracy in determination of moisture is gained, because these small samples can be weighed on a gold scale, while the old sample was distributed into two baths and weighed on a rough pound scale.

On p. 163, October *Journal*, Mr. R. E. Sawyer remarks : "An 11 in. galvanized iron pipe fitted with centre guides is then slipped over the 1 in. pipe and dropped into the tank, falling freely about 12 ft. The depth of the slime inside and outside the pipe is then accurately measured." The difference of outside and inside measurements was during the Cason experiments found to be considerable, going as high as 6 in. averaging 3 in. or 6% of the volume. This difference I have been able to bring down to nil by lowering the tube very slowly instead of "dropping it into the tank." The Cason experiments also showed a difference in inside depth before and after the washing out of the vat varying from an additional loss of 5 in. to a gain of 1·5 in. The old method gave further a loss of nearly 1 in. of the heavy slime in the bottom of the vat owing to the thickness of the circular disc and rubber plate. The method of emptying the pipe into baths caused a further additional loss through splash and transport and through slime sticking to the inside of the pipe. The whole arrangement was complex, and required many hands, and was therefore rather expensive in use.

The new slime sampler has done away with all these causes of inaccuracy. It has also proved very expeditious and convenient in use. There is no need to wait for the vat to be washed out or delay the cyanide work, and a large number of samples can be taken with little trouble and inconvenience. The drying of the samples takes only one to two hours as compared to one to two days previously, and the results can therefore be out in time for the day's tonnage.

The cost of the old apparatus for an efficiency of one sample in each of eight vats came to about £100, while the total cost of my new apparatus does not exceed £5 with an efficiency covering the total number of vats and any reasonable number of samples in each vat.

This sampler has now been in use for some time on the East Rand Proprietary Mines, and proved a great success. Its many advantages combined with moderate cost will undoubtedly recommend its use to many up-to-date slime plants of the Rand

I am greatly indebted to Mr. P. R. Nethersole for valuable hints and assistance in constructing the sampler and carrying out the experiments, and to the consulting metallurgist of the East Rand Proprietary Mines, Mr. E. H. Johnson.

The President : I am sure we have to thank Mr. Tonnesen very much for showing us this very useful device.

The meeting then closed.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Volhard's Method for Copper Determination.—"In searching over the chemical literature for methods on the rapid determination of copper, it is surprising that Volhard's titration method has not found the recognition which, on account of its exactness and rapidity, it should deserve. An outline of the method may perhaps be of value to technical chemists who are still in search for a good and rapid method. The results obtained with Volhard's titration agree very closely with those obtained by the electrolytic method, and in comparison with the iodine method it has the advantage that the ammonia sulphocyanate solution which is employed for the precipitation of the copper is much more stable and does not require constant checking. Besides the copper can be titrated direct without making a separation from the impurities necessary.

The following solutions are required:

Ammonium sulphocyanate. This solution is prepared by dissolving 14 g. c. p. ammonium sulphocyanate in 1 litre water. One cc. of this solution will precipitate 0·01 g. Cu.

Silver nitrate. 10·8 g. fine silver are dissolved in 50 cc. nitric acid and made up to 1 litre.

Ferric ammonio sulphate. A saturated solution of the pure salt.

The ammonium sulphocyanate solution is standardized as follows: 0·4 g. of c. p. copper are dissolved in nitric acid, made slightly alkaline with ammonia, acidified again with sulphuric acid and 50 cc. of the ammonium sulphocyanate are then added from a burette. Now add a few drops of a saturated solution of sodium sulphite and a few more drops of sulphuric acid till the solution smells distinctly of SO_2, and warm to 60° C till the precipitate settles to the bottom. Cool and transfer to a 250 cc. graduated flask, and mix well. Filter the precipitate through a dry filter, and take two 50 cc. suctions from the filtrate, add a few drops dilute nitric acid and iron ammonio sulphate solution and titrate back the excess of sulphocyanate with the silver nitrate solution. The endpoint is reached when the brown color disappears. The silver solution must be standardized against the ammonium sulphocyanate solution in order to determine the number of cc. of silver solution corresponding to the sulphocyanate solution.

For the determination of copper in the sample take an aliquot portion of the nitric or sulphuric acid solution, containing about 0·4 g Cu, neutralize with ammonia, make again slightly acid with sulphuric acid, add 50 cc. ammonium sulphocyanate solution and then proceed in the same manner as described for the standardization of the sulphocyanate solution. To calculate the Cu in the sample, deduct the number cc. ammonium sulphocyanate solution, corresponding to the number cc. of silver solution required, from the number cc. sulphocyanate solution added and multiply the difference by the titre of the sulphocyanate solution.

The method is very advantageous for all kinds of copper residues, slags, alloys, etc., and works as well on high-grade material as on low-grade. No metals except silver and mercury interfere with the direct titration of the copper. In the presence of silver the metal can be precipitated with hydrochloric acid; the silver chloride is filtered off, the filtrate is evaporated with sulphuric acid to fumes, and the method is then carried out as above."—ROBERT KANN.—*Mining Science*, Oct. 25. (K. L. G.)

ESTIMATION OF LIME IN PRESENCE OF MUCH MAGNESIA.—The estimation of lime in presence of an approximately equal amount of magnesia, as in dolomite, presents no difficulty. Double precipitation of the lime is sufficient to ensure results sufficiently accurate for any ordinary purpose. When, however, small amounts of lime have to be estimated in presence of 40 to 50 times their weight of magnesia, the usual methods of analysis fail so badly that on one occasion at least, a sample of magnesite containing 1·5 % of lime was returned by two experienced chemists as containing 0·1 %, and 4 % respectively. It is comparatively easy to secure complete precipitation of the lime as oxalate; but with much magnesia present it is impossible to give any directions which will prevent the precipitation of some of this, and though by repeated re-precipitations a number somewhere near the truth may sometimes be obtained, an examination of the precipitate and filtrate will nearly always show that this is due to an accidental compensation of errors.

A method depending on the solubility of magnesium sulphate in 35 % alcohol has been recommended for use in such circumstances; but it is now shown by E. Murmann in the *Zeitschrift fur Analytische Chemie* (1910, p. 696) that calcium sulphate is by no means insoluble in dilute alcohol, and that, in consequence, the method gives low results.

A slight modification of the method, however, works well. Enough, but not much more than enough, sulphuric acid is added to the solution of chlorides of calcium, and magnesium to combine with the whole of the calcium, and the solution is evaporated to dryness. The residue is extracted with 90 % alcohol, and in the soluble portion—which consists of calcium sulphate and a little magnesium sulphate—is then dissolved in water. To this solution oxalic acid is first added, and then, by degrees, ammonia to faint alkalinity. The precipitate is pure calcium oxalate, and accounts for the whole of the lime if sufficient sulphuric acid was added at the commencement of the experiment. Since the exact amount of lime present is of course not known in advance, it is advisable to use different amounts of sulphuric acid in duplicate experiments. For example, in the analysis of 0·5 gm. of magnesite supposed to contain about 1·5 % of lime, quantities of 4 cc. and 6 cc. of decinormal sulphuric acid might be taken in duplicate experiments, enough that is, to fix lime up to 2·24 % and 3·36 %. If the results agreed, they would be considered satisfactory; but if the one in which the larger amount of sulphuric acid came out a little more than 2·24 %, it would be taken as correct; whilst if a number approaching 3 % were found, a fresh experiment with 8 cc. of acid would be advisable. *Chemical Trade Journal*, Sept, 17, 1910, p. 262, and *Zeit. fur angewandte Chemie*. (T. D.)

METALLURGY.

GRINDING PAN PRACTICE.—"In the milling practice at the Great Fingall Mine the ore is coarsely crushed in the battery and subsequently ground in Grinding Pans, each of 5 ft. diameter. Instead of the ordinary form of spitzkasten discharge, however, a discharge pipe connected to a 3-in. bend is fitted into the pan at the side opposite to that from which the 'feed' is supplied, about 18 in. above the bottom and 12 in. from the side.

This apparatus has been in use for over twelve months, and is the results of the experiments based upon the principle of segregation by centrifugal force. The experiments were made by Mr. C. C. Freeman, who is responsible for the idea.

The particular advantages of the Freeman classifier over the ordinary type are that it requires no attention whatever, and its work is most consistent. The chief feature about it is that the particles, as soon as they are crushed to the desired size, are immediately liberated from the grinding pan: thus the grinding surfaces of the shoes and dies of the pan are always in contact with the coarser particles. The speed required for working this system is about 60 revolutions per minute.

Prior to the introduction of the Freeman classifier the gradings of the sand supplied to the leaching vats of the cyanide plant showed there was from 10 % to 15 % of 'on 30' product. This was the result of crushing through 10 × 10 and 8 × 8 screens, with subsequent grinding in pans fitted with ordinary classifiers, whereas the sand to vats since the installation of the Freeman system shows only 1 % to 6 % of 'on 30' product, crushing through 7 × 7 screens in the battery, and this despite the fact that the stamp duty has been increased from 7·5 to 9·2 tons per stamp per 24 hours.

A pan fitted with this form of discharge comfortably handles the whole of the product from five head of stamps using 7 × 7 screens, and giving a duty of 9·2 tons per stamp per diem."—G. C. KLUG. —*Monthly Journal of Chamber of Mines of W. Australia*, Sept., 1910, p. 309. (H. A. W.)

A MODIFICATION OF PACHUCA-TANK PRACTICE.—"Having seen various articles in your journal on the subjects of pulp agitation and zinc-dust feed in connection with the cyanidation of silver ores, and believing that there are some interesting developments on these lines, I give you the following data: In this plant the usual custom in grinding is followed, namely, preliminary crushing in rock-breaker and stamps (20), then tube-mills, one 4 ft. 6 in. × 16 ft. Allis-Chalmers mill and one 8 ft. Hardinge conical mill. Classification is accomplished with Dorr classifiers and thickeners, followed by pneumatic agitation in Pachuca tanks, 13 ft. × 55 ft., filtering on two Oliver continuous slime filters, 11 ft. 6 in. × 12 ft., and precipitation with zinc dust in Merrill press. The crushing, classification, and settling of the ores here present no unusual features, but in the matter of agitation and precipitation I think practice here has developed some new points which will be of interest to the profession.

Agitation.—While Pachuca tanks are recognized as a simple, cheap, and, to a certain extent, efficient means for the agitation and aeration of pulp, they fall considerably short of what might be desired in the matter of aeration, most of the air escaping in large bubbles at the top of the column pipe with no benefit to the pulp. Observing this, efforts were directed to overcome the defect by cutting off the column pipes as follows: Out of three Pachuca tanks the column pipe in one was cut off at one-quarter distance from the top and raised, all the air thereby being brought into more general contact with the pulp from a point one-quarter distant from the top of the tank, and a reduction in the value of tailing became immediately apparent. The column pipe in No. 2 Pachuca was then cut in two in the middle and the upper half raised, resulting in still better aeration and a still further reduction in value of tailing. Pachuca No. 3 was left with column pipe full length. With the tanks in this condition several charges were run in each, the average tailing from No. 1 being 0·05 oz. per ton lower than from No 3 and those from No. 2 being 0·16 oz. lower than No. 1, or 0·21 oz. lower than No. 3, the unaltered tank. The theory being thus apparently confirmed, a second series of five charges per tank was run with the column pipes as above described (No. 1 one-quarter off, No. 2 one-half off, No. 3 full length), resulting in average tailing as follows: from No. 1, 1·90 oz. per ton, from No. 2, 1·84 oz. per ton, and from No. 3, 2·09 oz. per ton. In all cases after 18 hours agitation. Subsequently all column pipes were cut in the middle and the upper portions raised, resulting in a corresponding reduction in the value of tailing. The cutting of the pipes has the further advantage that it is possible to commence the agitation of a tank when it is only half charged; also agitation is more easily started when for any reason it has been necessary to temporarily cut off the air; and agitation from the middle of the tank tends to prevent the settling of sand. The maximum power required for agitating a 100-ton charge of 1½ solution to 1 of pulp is 4 h.p. Leaving the Pachucas the pulp goes direct to the Oliver filters, which easily handle 200 tons per day between them in a satisfactory manner, considerably exceeding the rated capacity. The solution is then precipitated by the use of zinc dust in a Merrill press, to which it is fed by gravity flow.

Precipitation.—The zinc was originally fed with the Merrill zinc-dust feeder, consisting of a 12 in. belt conveyor, 10 ft. long, operated by floats, and discharging into a cone, where emulsion was supposed to be made by the use of air. Owing to the irregularity of the operation of the belt and the uneven distribution of zinc-dust on it, and the fact that the cone failed to make a perfect emulsion, a large percentage of the zinc settling and caking on the sides of the cone, the whole apparatus was abandoned, and the following equipment made, which is now in use. This equipment consists of a receiver having a diamond-shaped cross section (as described by J. S. Colbath in *The Engineering and Mining Journal* of February 26, 1910), this taking the solution direct from the press, measuring the flow, as indicated by a register attached to a tripper, and also regulating the quantity of zinc required. The receiver operates the tripper above mentioned, which makes an electric contact each time the receiver dumps. The contact current passing through a coil-magnet operates the zinc-feeder, which has a ratchet attachment for the purpose, and which is situated at the clarifying tanks, 400 ft. distant from the press. The feeder is operated on the Challenge principle, the feed of zinc being further regulated by a gate on the hopper. The zinc dust is discharged into a launder and flushed direct into a miniature tube-mill, 4 × 14 in., charged with pebbles and run at a speed of 60 r.p.m. This makes a perfect emulsion, which we find indispensable; has materially reduced our consumption of zinc, and resulted in producing a high-grade precipitate, running 80% silver, whereas formerly our precipitate ran only from 35% to 50%. This scheme has been in operation several months with excellent results. It is automatic, and the cost of operation is practically nil, the tube-mill being operated with a ¼-h.p. motor."—AMOS J. YAGER.—*Mining and Scientific Press*, Oct. 22, 1910, p. 539. (W. R. D.)

EFFECTS OF LEAD SMELTING.—"The following has been submitted to the Central Board of Health (S. Aus.) by the Chairman (Dr. Ramsay Smith):—'I held an enquiry under the Health Act, 1898, into the occurrence and cause of lead-poisoning at Port Pirie. In addition to examining witnesses who were summoned and witnesses who volunteered to give evidence, I made a detailed inspection of the smelting works. The medical witnesses, the manager of the works, and the employes showed remarkable unanimity in their testimony in regard to matters of fact, and there is little, if any, difference in their evidence in respect to matters of opinion. Briefly, the evidence may be summed up as follows:—1. The general population of Port Pirie and the neighbourhood has not suffered in any way from lead-poisoning due to the smelting works. 2. All the cases of lead-poisoning that have occurred have been among people working in the smelting works, and those handling lead on the wharves, and painters. 3. Since the disease is not a notifiable one, there are no complete statistics of the extent of its occurrence; but from the evidence I estimate that during the past three years from 150 to 200 cases have occurred. 4. The sufferers were nearly all young men from 20 to 35 years of age. Some had recurrent attacks, but such people are generally advised to undertake such work as would prevent them from being again exposed to poisoning, and they wisely take the advice. 5. The disease showed no local peculiarities. Colic and anaemia were the most common symptoms, but paralysis also occurred, and in a few cases fits and mania. 6. Some of the cases were very severe, and a fairly large proportion proved fatal. One medical witness said he knew of fully a dozen deaths within the past 10 years. 7. The disease occurred in strong and weak alike. People addicted to alcohol acquired it more frequently, and suffered more severely; but one of the most severely affected was a total abstainer, and he had a recurrent attack. 8. The witnesses ascribed the disease chiefly to exposure to fumes on the top floor, at the lead wells, and the slag pots at the discharge holes. 9. The company (The Broken Hill Proprietary Company, Limited) has for years provided extensive and adequate facilities on the premises for bathing, and such appliances as lead workers require in order to take proper precautions against poisoning; but very few of the men have taken advantage of them. Some do not know of their existence. 10. Some workers are careful to observe all the precautions that are recognised as safeguards against lead-poisoning in the matter of foods, drink, tobacco-cutting, bathing, and changing clothing; yet some of these may suffer. On the other hand, most of the cases occur in those who, from ignorance or carelessness, do not observe proper precautions. 11. The placing of hoods working automatically over the slag pots at the discharge holes has been followed by a very great diminution in the number of cases of poisoning. The hoods prevent the fumes from being inhaled by men at the spot, and from being blown to the upper floor. 12. There is a difficulty in respect to the lead wells, but there is a prospect of its being overcome, if not by mechanical contrivances, then by the use of respirators and by other precautions on the part of the workers. My recommendations are:—1. Recent action by the company had enormously reduced the number of cases, but I think it would be well, as suggested by the manager to draw up a set of instructions to the workers, advising them how to take advantage of the facilities provided, and how to attend to matters of personal health and habits, so as to avoid those dangers that are inseparably connected with their health and livelihood. 2. Although it has been found necessary in other places to introduce legislation making certain precautions compulsory on the part of the workers, as well as the employers, I do not advise such procedure, unless it should appear that cases of lead-poisoning still continue to occur on account of the negligence of the men."—*Australian Mining Standard*, Oct. 3, 1910. (K.L.G.)

THE ELEMENT OF CHANCE IN THE SAMPLING OF ORE.—"All that can be done in the matter of sampling is to mix and to take a portion. Mixing produces unevenness; an even mixture of different things can only be obtained by counting, sorting, and ranging them.

Impartial mixing is the basis of sampling

The question of what is a 'safe' weight of sample has to be studied empirically or by trial and the best way of doing so is to study extreme cases. In the following brief summary of investigations each individual assay, unless otherwise stated, was made on a portion of the ore-sample ground to 100 mesh or finer, commonly called the 'pulp' and weighing one-half of an 'assay-ton.' The assay-ton or A.T, for short, weighs 29,166·66 mgm and each milligramme of gold obtained per assay-ton represents 1 oz. of gold per ton of ore. The results are stated in ounces per ton, generally to the second, and more rarely to the third, place of decimals. The balance used in weighing the buttons is sensitive to 0·005 mgm., which corresponds to 0·005 oz. gold per ton, and the buttons are weighed to the nearest 0·005 mgm. It is usual to make the fusions and assays on portions of ½ A.T., consequently the results reported in ounces per ton are correct, so far as weighing is concerned, to the nearest 0·01 oz. gold per ton, and deviations of more than 0·01 are due to other causes than avoidable errors in weighing. In commercial practice it is often agreed between the buyer and seller that differences of 0·03 oz. per ton or less shall not be disputed or submitted to umpirage.

In order to throw light upon the extent of differences arising in good assaying practice a mixture of sand and limestone of 100-mesh fineness was soaked in a dilute solution of gold chloride to compose an artificial gold ore of about 2 oz. per ton and of homogeneous content. The mixture was dried and calcined with the intention of coating each particle with a film of gold so that the particles should be as nearly equal in gold content as art could arrange. If the intention was successful then the vagaries of chance would be reduced to a minimum, and each ½ A.T. would contain well within the sensitiveness of the assay-balance the same quantity of gold, and the visible deviations between one assay and another would be due to error in assaying.

In the first set of six assays there were five of 1·92 oz. and one of 1·94 oz. Here there was an extreme difference of 0·02 oz. per ton. In a second set of four assays two were 1·92, one of 1·94, and one of 1·91. Here there was an extreme difference of 0·03 In another series 10 results of 1·915 were obtained with one of 1·895, the extreme difference being 0·02 oz per ton. Combining the 24 observations, the mean is found to be 1·918, with a mean error of ± 0·0017 and an extreme difference between observations of 0·015. It would thus seem that with a homogeneous ore a difference as high as 0·015 oz. per ton, or a difference higher than the limits of good weighing, can arise, due presumably to deviations in the assay operations other than those of weighing, between the highest and lowest results of a number of ½ A.T. assays. The extreme differences between individual assays of ½ A.T. found in the ordinary practice of sampling gold ores are very much greater than 0·005 oz. per ton.

If we want to study the magnitude of the element of chance as it affects the sampling of ores we cannot do better than to choose an extreme case, such as may be found among ores containing free gold, where the value of the mineral is high and the difference of specific gravity between it and the gangue is wide. If all the particles of gold in such an ore were free and distinct from the particles of the gangue and all the particles both of gold and gangue were of one size and that size were equal to 100 mesh, and the ore were of a grade of 1 oz. gold per ton, then in one assay ton there would be approximately 4,000,000 particles of quartz or gangue and 21 particles of gold. Of course, no one would expect to be able to take out from a bulk of such ore of 100 mesh fineness various small portions that would always contain 21 particles of gold and no more or no less for every 4,000,000 particles of gangue. We are not wizards. We would naturally and instinctively expect considerable variations from the average number of 21 particles of gold per 4,000,000 particles of quartz. A difference of one particle would make an error of 5%. We could reasonably expect any number of particles, say, from

10 to 30 of gold, rather than exactly 21, neither more or less.

In most ores that have been crushed the valuable ingredient is chiefly in the fine and it is therefore likely that the average particle of mineral is smaller than the average particle of gangue. In other words, it is probable that in our samples there are more particles of mineral than there would be if the average size of the particle of mineral were equal to the average size of the particle of gangue. For this reason in our hypothetical ore, with all its gold free, there would probably be many more than 21 particles of gold for every 4,000,000 particles of quartz, because their average size would be smaller than the average size of the whole ore, but, relatively to those of quartz, the number would be extremely small and the chances of deviation from the average number in any one small sample would still be very great.

In this article I want to insist upon the element of chance, which makes it impossible for us to get an absolutely true average sample from a mixture of different things. We can only get a sample the deviation of which from the true average is negligible by taking a sample containing a sufficiently large number of the particles or things to be estimated. If, for instance, we had an average, as in the case imagined, of 21 particles of gold in a sample we would expect to get deviations of relatively great magnitude in different samples. By taking our sample so that on an average there were 210 particles, the percentage of deviation would be less, and so forth. In a general way it is supposed that the percentage maximum deviation of a sample from the true average would be reduced 3·17 times by taking a sample 10 times greater.

To show the importance of these differences the first two assays of many 'pulps' of different gold ores containing free gold were noted. The average difference between the two assays of each of 24 *consecutive* samples of an ore marked (*B*) was 0·043 and included differences ranging from nothing up to 0·23 which was the maximum difference. Thirty-six sets of 2 assays of each of 36 *consecutive* samples of an ore marked (*A*) gave an extreme difference of 0·33 and an average difference of 0·05. Sixty-four sets of 2 assays each of 64 consecutive samples of (*C*) gave an extreme difference of 0·6 and an average difference of 0·068.

A gold ore known among assayers as being 'spotty' (that is, giving wide differences when assayed) in 12 assays gave an extreme difference of 0·345 between two ½ A.T. assays or a difference of 0·69 oz. on a total assay-value of 2·93 oz. per ton, a difference of 23%.

The evil effect of such large differences between individual assays can be neutralized by making a larger number of assays and taking the average of them. In the parlance of ore sampling, ½ A.T. is not a 'safe' sample of such ores.

Before discussing the question of weight as affecting the 'safety' of a sample of such 100 mesh pulp or one that with another of the same weight might only differ within the limits of good assaying, for example, 0·045, the mean of which would be 0·0225, it is well here to refer to a series of experiments made with the 'pulp' as delivered to the assayer. This pulp is commonly ground so that all of it will pass through a 100 mesh screen, and a great many experiments were made with such pulp along the following lines: First, could a method of mixing the pulp be discovered that would render the distribution of the gold more even than is indicated by ordinary practice? Could these extreme differences between ½ A.T. samples be reduced?

Second, and conversely, could the pulp be so handled by the assayer, on the rolling cloth or paper, within the limits of fair mixing or selection of the sample as to increase these differences?

Both of these lines of investigation failed to show that the extreme differences could be favourably or unfavourably influenced. The pulp was divided in every conceivable way. The ½ A.T. sample was taken in small portions from over 100 different parts of the bulk or as one portion from one part of the bulk. A stream of the pulp was cut a number of times by a miniature time-sample and the ½ A.T. was composed of these cuts. The pulp was rolled in various ways, thought likely to be conducive to the segregation of its mineral or to its perfect distribution, though it is difficult to conceive how segregation could be induced by the slow rolling of a material the particles of which tend to cling to and lock into each other.

I shall not enter into the details of experiments made upon the finer grinding of the samples. The conclusion reached by those engaged in the work was that finer grinding was of no assistance in reducing the differences, and that as a size for the assayer 100 mesh or even less was practically as safe as 200 mesh. The results were always the same. It was as impossible to combat the element of chance as it would be in the case of different coloured balls taken blindfold from a bag or of cards dealt from a fairly shuffled pack.

There is a well-known formula used by physicists and chemists for the purpose of calculating the mean error, plus or minus, of the mean of a number of observations. It is:

$$F = \frac{\Sigma d^2}{n(n-1)}$$

in which

F = mean error of the mean of all the observations.

d = deviation of each measurement from the mean.

Σd^2 = sum of the squares of the deviation.

The satisfaction, if any, we get from this formula, is to learn that we can only reduce the mean error of the mean in the proportion of the square root of the number of observations. If we want to reduce the mean error of the mean to one-tenth of its magnitude we must take 100 times as many observations.

Given an ore that with samples of ½ A.T. exhibits a difference between the highest and lowest of 0·6, if we wanted to take samples sufficiently large to exhibit a difference no higher than 0·06, we should (neglecting the possible but smaller error due to assaying details) have to make the weight assayed as high as 50 A.T. If by convention it were agreed that a 'safe' weight of sample should be one that did not differ from another sample of the same weight more than 0.06, we should have to make that weight for such an ore 50 A.T. This is 10 times as heavy as the sample usually distributed by the sampling works to the seller, buyer, and umpire. 50 A.T. would be the smallest safe division we could make of such a pulp.

Every time a sample is divided, and in the sampling operation it is divided many times, there may be an error, which will be important according to the number of particles of the sample and the character of the ore.

Having determined for an ore or class of ores the safe weight for a sample reduced to 100 mesh, can we then from this knowledge alone determine the safe sample-weight of any coarser size?

H. A. Vezin, who is identified with this question of the safe weight, having found that of the pyritic ores of Gilpin county, Colorado, one ounce or 1 A.T. was a safe weight for a sample that had passed a 20-mesh screen, for all other sizes larger multiplied this weight by a number that represented the number of times the volume of the largest sized particle was greater than the volume of a 20-mesh particle, that is, by a number proportional to the cubes of the sizes of the largest particles. Vezin's statement that the safe weights of another larger size than 20-mesh could be obtained by multiplying the ascertained safe weight of the 20-mesh size by the number of times the volume of the larger size is greater than the volume of the smaller size, could only be founded upon the assumption that the ratio of the number of particles of the richer mineral and the poorer gangue was the same in all the stages of the crushing of the ore from coarse to pulp and that during the grinding the mineral and the gangue were crushed at the same rate.

We know that the fine ore in any stage of crushing contains more of the valuable mineral than the coarse. The valuable component is chiefly in the fine ore. When the pieces of ore are 2 or 3 inches in diameter we do not find pieces of gold that are 2 or 3 inches in diameter, nor do we notice similar conditions in other smaller sizes. The particles of the richest mineral, namely, gold, in ores containing free gold, are in size obviously smaller than the other particles of the ore.

In the following experiment a gold ore was sized and the various sizes were assayed with results as stated:

Vezin's principle that a definite ratio between the weight of the sample and the size of the largest particles shall be preserved through the whole of the crushing operation, is intelligible if the relation between the number of particles of the mineral and of the gangue remains constant, otherwise it would be purely arbitrary. If it has any merit it errs on the safe side because in the coarser sizes the proportion of the valuable mineral particles is, owing to their fineness, much greater that in the finer sizes.

His table, which applies particularly to the pyritic ores of Gilpin county, Colorado, is thus given in Hofman's 'The Metallurgy of Lead.' This table starts with a sample of 20 mesh or $\frac{1}{30}$ of an inch, for which an assay-weight of sample of $\frac{1}{16}$ lb. or about 1 A.T. was taken.

It is difficult to think of any method other than that of actual trial for approximately deriving the safe weight of all sizes from the safe weight of one size as determined by trial. To construct a curve to serve for any size we want three points or the known safe weight of three sufficiently different sizes, at least, and the curve then would be true only for that ore or class of ore. For the ores discussed in this paper I had the opportunity of making a similar comparison between the contents of two samples of the same ore taken at ½ in. and at 100-mesh, the latter being the size to which it is reduced before it passes into the hands of the assayer. In the sampling of these ores two cuts of from 50 to 100 lb. each were taken when the ore was of a maximum size of ½ in. According to the standard practice obtaining in the United States this weight of sample is considered safe for that size. The volume of ½ in is 11.737 times that of a particle of 100-mesh size. According to the practice recommended by Vezin, if 50 A.T., or 3·2 lb., were the safe sample at 100-mesh, then at ½ in. we would have to take 373,584 lb. as a safe sample. In the following table is given the maximum and average differences noted between pairs of samples of different ores at 100-mesh and ½ in. maximum size.

It will be noted that generally the 50 to 100 lb. sample at ½ in. is safer than ½ A.T at 100-Mesh, both in respect of maximum and average difference. In this case the difference may be the result of the accumulation of deviations arising in different divisions of the sample, because in reducing the bulk of a sample at ½ in. size to a few pounds of 100 mesh pulp it has to be crushed and divided successively into halves. This division is made six times. At each division a difference large or small, is undoubtedly originated, and these differences may not always be neutralized by the next division, and thus the differences may be cumulative and not truly representative of the difference between the two 100-mesh portions taken at the ½ in. stage. This is the difficulty that pursues the ore sampler at every step

	Ore E.		Ore F.		Ore C.		
Size in mesh	Percentage by weight	Assay Oz. per ton	Percentage by weight	Assay Oz. per ton	Size in mesh	Percentage by weight	Assay Oz. per ton
On 8	69·88	1·72	62·41	2·02			
8 to 16	11·36	3·41	13·90	2·60			
16 to 30	6·70	4·29	9·33	3·74			
30 to 60	3·67	4·88	5·54	4·45			
60 to 120	2·75	6·30	2·25	6·34	60 to 80	15·8	0·44
Through 120	5·69	8·81	6·57	8·63	80 to 100	16·	0·49
					100 to 120	10·2	0·69
					120 to 150	4·6	0·90
					150 to 200	9·6	1·25
					Through 200	42·4	2·73
					Loss (dust)	1·4	
Calculated average		2·75		2·93			1·56

Diameter of largest particle in inches..	$\frac{1}{30}$	$\frac{1}{15}$	$\frac{1}{8}$	$\frac{4}{15}$	½	1⅛	2¼
Minimum weight of sample in pounds...	$\frac{1}{16}$	½	4	32	256	2048	16,348

Ore.	Difference between pairs of ½ A T. assays.		Difference between pairs of 3 A.T. assays including the equation of the two assayers.		Assayers.	Difference between pairs of 50-100 lb. sample weights at ⅛ in.	
	Maximum.	Average.	Maximum.	Average.		Maximum.	Average.
B	0·23	0·043	0·124 0·164	0·045 0·040	V and W V and X	0·195	0·034
C	0·60	0·068	0·19 0·19	0·049 0·050	Y and Z Y and W	0·395	0·060

and is particularly noticeable with special ores. If one half of the difference be attributable to an original difference at the time of dividing the ⅛ in. sample, then for the ore (C) with a difference of 0·395, the weight of a sample to show no greater difference than 0·06 would be about 815 lbs. If the whole of the difference is attributable to the same cause then the weight would be 3,260 lb., a quantity immensely smaller than required by the old-fashioned rule of the cubes of the largest particles.

It is the practice in Western American sampling plants to distribute the samples of the 100-mesh pulp in 5 oz. packages to the shipper, buyer, and, when required, the umpire. If a number of results obtained by different assayers are arranged in horizontal lines, each line containing the results found for the same ore, and if each assayer be given a separate column, we can compare both the work of the assayers among each other and also the natural unevenness of the different portions of the same 100-mesh pulp as divided and submitted to the different assay-offices. On adding the columns in corresponding groups of 10 or 20 horizontal lines we obtain the total results of the different assayers on corresponding samples. These totals generally harmonize except for certain differences that are usually one-sided.

Thus, for example, on a number of results assayer W will be higher or lower than some other assayer by an amount that for each sample seems trifling though in the aggregate it may be quite important. This one-sided difference in the totals is distinct and characteristic. It may be called the equation of that particular assay-office and represents the sum total of the methods and persons involved in the work, and it may be large enough to be serious. If we examine the items in the horizontal lines we note differences sometimes larger than is at all usual between the work of the best assayers, but these differences are not all in one direction; they are capricious, erratic, and impartial. These differences are mainly due to the natural unevenness of the ore.

To illustrate this unevenness of pulp, the assay-values of different portions of 5 oz. taken from the same bulk of 100-mesh pulp, of which 3 A.T. were used in each of the results reported by the different assay-offices, were compared and set forth in the following tables. They represent the average and maximum differences found between different assays, and for the purpose of intelligent comparison I give also the differences noted between pairs of ½ A.T. assays made in the same assay-office and also those differences noted between duplicate 50 to 100 lb. samples taken at the size of ⅛ in., all for the same lots of ores B and C, and from a large number of consecutive samplings.

If we turn back to the assays of an ore all of which was reduced to less than 60 mesh, given according to screen sizes, we note that for such an ore when assaying 1·56 oz. the worst that art could do by screening, when dividing such an ore at that stage into two halves by weight, would be to get one half of an assay-value of 0·3 oz. and the other of 1·26 oz., or a difference between two samples of 0.96 oz. The perfectly mixed pulp at 100-mesh of similar ore yields differences between two ½ A.T. samples of 0·60 oz. That was the work of chance.

AVERAGE ASSAY.—OZ. GOLD PER TON.

Size.	East.	West	Difference.
½ to ¾ inch	1·18	1·64	0·46
½ inch	1·13	1·06	0·07
	1·14	1·13	0·01
	0·985	1·12	0·135
¼ inch	1·10	0·99	0·11
	1·12	0·99	0·13
	0·875	0·93	0·055
⅛ inch	1·32	1·58	0·26
20 mesh	1·48	1·435	0·045
	0·97	1·05	0·08
	1·05	1·03	0·02
	0·855	0·825	0·03
40 mesh	1·45	1·495	0·045
	0·99	1·00	0·01
	1·07	0·98	0·09
	0·81	0·795	0·015
60 mesh	1·36	1·88	0·52
	0·93	0·97	0·04
	0·98	1·00	0·02
	0·79	0·83	0·04
100 mesh	1·37	1·36	0·01
	1·03	0·98	0·05
	0·96	1·01	0·05
	0·82	0·795	0·025

In order to see what would happen when 1½ lb. lots of ore of different sizes were divided over a single edge into two portions without any special care to avoid size separation the following results were obtained. One side of the dividing edge is called 'East' and the other 'West.' The samples were poured over the dividing edge from a shovel.

The number of experiments of this kind is far too small to determine maxima yet it is instructive.

Ore	Average Assay Oz. per ton	Number of pairs	Difference between pairs of ½ A.T.			Difference between pairs of 50 to 100 lbs. lots of ⅛ inch.		
			Maximum	Average	Max, Avg.	Maximum	Average	Max. Avg.
A	1 48	36	0·33	0·050	6·6	0 340	0·04	8·5
B	0 85	24	0·23	0·043	5·3	0·195	0·034	5·7
C	2·10	64	0·60	0·068	9·	0·395	0·060	6·6

There need be no surprise at the difference of 0·52 oz. at 60 mesh. A very little experience with what are called 'spotty' ores will convince anyone that with the best assaying larger differences than 0·52 can arise between ½ A.T. samples, crushed to 100 mesh.

The more we ponder on the assumption implied in the Vezin rule the more we are at a loss to understand its inception. It seems opposed to the instincts of a practical mind to imagine that this proportionality of the numbers of the particles of different things is going to be preserved through all the crushing of the ore from coarse to fine in all its stages. The general observation that the pieces of valuable mineral are generally smaller than the gangue should have discredited the idea that a fixed ratio between the weight of a sample and its largest particle should remain constant through all the stages of crushing.

To raise the matter of ore-sampling to a perfect art, the science of the unevenness of mixed things must be applied to its operations and, this being recognized, means should be taken so that the sampling operation can be checked within itself by the duplication of the sample at any of its many divisions." — LEWIS T. WRIGHT. — *The Mining Magazine.*—Nov., 1910,—p. 353, (A. R.)

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THE FIELD ASSAY OF WATER —" During the last six field seasons more than 3,000 field assays of natural waters, by the methods suggested by Marshall O. Leighton in 1905, have been made by United States Geological Survey parties. The purpose of Leighton's methods have been widely misunderstood. They are not intended to displace laboratory analyses, nor is it claimed for them that they yield results comparable in accuracy or availability with laboratory methods. Leighton's purpose was to place at the disposal of engineers, chemists, geologists, and scientific explorers in general a means of making in the field partial approximate assays of natural waters by the use of solid reagents and apparatus that can be carried in a knapsack. The experience of the Geological Survey has been that in careful and trained hands Leighton's tests, granted a proper appreciation of their limitations, on the part of the user, are of great practical utility in industrial and geological reconnaissance.

The limitations of the field assay are defined by R. B Dole of the Survey in *Engineering News* for August 11. Fifty-two waters, ranging in total solids from 25 to 5,2 0 parts per million and representing the common kinds of surface and underground waters in the United States, have been analysed in the laboratory and tested by field methods. The results of the two methods Mr. Dole presents in parallel columns. Without reproducing the exact numerical data we may summarise briefly Mr. Dole's discussion of the results and his conclusions as to the practical utility of the field assay.

Dealing first with the determination of bicarbonates, 'the estimates of bicarbonates in the laboratory were made by titrating measured amounts of water with *N*/50 sulphuric acid in the presence of methyl orange indicator. The field tests were done in reverse manner; a proper number of tablets of sodium acid sulphate were dissolved in a small amount of the water to be assayed, methyl orange was added and then water until an alkaline endpoint was reached, after which the volume of liquid was measured. One of the tablets used in the tests is equivalent to 2·49 mgm. HCO_3. The parts per million of bicarbonates (HCO_3) in a water can readily be computed from the amount of water, the number of tablets, and the value of one tablet in milligrams of HCO_3.'

The average error of the field tests in the 44 cases which Mr. Dole considers significant was 6·2 parts per million, or 3.5%, with water containing 100 to 350 parts per million of bicarbonates; the errors ranged from 0% to 9% and were equally divided between positive and negative. It is clearly established that the error is not a constant one, due to error in the value of the tablets or to the change in shade of the indicator, but that it is variable and is caused probably by error in measuring the water. The average error of 3·5 % corresponds to 2·1 cub. cm., not an unreasonable variation considering the facilities for measurement. 'Most waters do not contain normal carbonates (CO_3) and a qualitative test suffices to show their absence: but if it is necessary to make the quantitative determination it may be performed on a separate portion of water in the presence of phenolphthalein and the bicarbonate figure may be corrected in accordance with the result.'

Chlorides were determined in the laboratory, when less than 100 parts per million, by titrating 100 to 250 cub. cm. of the water concentrated to 25 cub. cm. with a solution of silver nitrate, 1 cub. cm. of which was equivalent to 0·5 mgm. of chlorine: and when more than 100 parts per million, by precipitating and weighing the chlorides as silver chloride. In the field tests, 50 cub. cm. of water were titrated with tablets and quarter tablets of silver nitrate. 'For low chlorines tablets equivalent to about 1 mgm. of chlorines were employed, and as one-quarter tablet in 50 cub. cm. of water is equivalent to 5 parts per million of chlorine, the field results may vary from the true values 5 parts, a difference which may be equivalent to an error of 100 %. Such discrepancies offer little practical disadvantage, however, for it is as useful in geological or industrial reconnaissance to know that a water contains less than 9 parts of chlorine as to know that it contains, for instance, exactly 5·4 parts.' The accuracy of low-chlorides estimates is increased by using 100 instead of 50 cub. cm. of water wherever practicable, but no particular advantage has been gained in using weaker tablets instead of quartering the tablets of ordinary strength.

The average error of 22 tests of waters containing more than 50 parts of chlorides was 6 parts, or 3·2 %, and the errors are almost equally divided between positive and negative. 'If the chromate indicator is added at the beginning of the titration of waters exceeding 400 or 500 parts of chlorides, an excess of silver nitrate chloride formed during solution of the tablets becomes stained with silver chromate that is decomposed with difficulty. It is better, therefore, to make first a rough estimate of the chlorides and then to repeat the determination, the indicator in the latter test being added near the end of the titration. All the tablets except one or two may be ground with a small quantity of water, then the remainder of the measured portion of the water may be put in, and the titration completed in the usual manner. This modification involves little extra work and increases the accuracy of the results.'

Sulphates were determined in the laboratory by the usual gravimetric method in 250 or 500 cub. cm. of water. In the field tests about 1 gm. of coarsely pulverized barium chloride was added to about 100 cub. cm. of water made very slightly acid to methyl orange by the addition of hydrochloric acid (1—1). The mixture was then shaken for ten minutes or until the precipitant was entirely dissolved and the turbidity of the liquid was then measured by means of the regular form of candle turbidimeter in a shaded corner of the laboratory in daylight. The solution was kept constantly and thoroughly agitated while three readings were made; a standardization table gives the amount of sulphates corresponding to the average of the three readings. The success of the field method depends upon the addition of a minimum of acid and the constant agitation of the liquid in order to prevent concentration of barium chloride at the bottom of the tube during precipitation and to prevent agglomeration after precipitation. The use of the regular turbidimeter is limited to the estimation of sulphates from 30 to 600 parts per million. Less than 30 parts of sulphates were determined by comparison of the turbidity with that produced in waters containing known amounts of sodium sulphate treated with acid and barium chloride. These standards of course cannot be carried in the field, but it is possible for a skilled observer to estimate from memory below 30 parts certainly within 50%. Above 600 parts the water was diluted to a convenient strength with water free from sulphates before precipitation. The best results are obtained in readings on waters containing between 56 and 265 parts of SO_4, and it is best to dilute to bring the readings between these limits.

The average error of the tests of the 27 waters containing more than 30 parts per million was 10%. For turbidimeter readings between 2 cm. and 19 cm. the average error is 8·5%. For readings less than 2 cm. and for sulphates greater than 600 parts, the errors of single estimates are much greater, averaging 15·4% in the nine cases reported by Mr. Dole. Indeed, individual determinations of high sulphates are liable to great error.

In the laboratory tests for iron, 'the iron in the residue from 500 cubic cm. of water was removed as ferric hydrate and redissolved; then the amount of iron in this solution, or in an aliquot part of it, was estimated by the usual colorimetric method, the reddish hue developed by potassium sulphocyanide being compared directly or indirectly with that developed in standard solutions of ferric chloride. In the field tests, 45 cubic cm. of water and 2 cubic cm of concentrated nitric acid are thoroughly mixed and allowed to stand 5 minutes; 3 cubic cm. of a 2% solution of potassium sulphocyanide are then added.' The red colour of the solution was then compared with that of standard celluloid discs. This is the regular field method recommended by Leighton, with the exception that the colouring comparison was made immediately after the addition of the potassium sulphocyanide instead of 10 minutes later, as it was found that the compounds producing the red colour are progressively decomposed by free nitric acid. In a second series of tests the mixture of water and nitric acid was allowed to stand one hour before addition of potassium sulphocyanide. It is apparent from the results that in most cases the iron is more completely oxidized if the second method is followed. The errors in the tests by the first method vary from 0·2 to 2·4 parts per million (8% to 100%) and in those by the second method from 0·1 to 1·0 (5% to 100%). The degree of change into the ferric form depends on the composition of the water and the chemical condition of the iron. Apparently much organic matter retards the change more than anything else in natural waters.

The estimation of the total content of scale-forming ingredients in the field is made difficult by the lack of satisfactory field methods for the determination of calcium and magnesium. The turbidimetric method proposed by Leighton for calcium is subject to great and uncertain errors, and the procedure devised by Catlett for magnesium has not yet been applied to water assaying. 'The figure for total hardness obtained by titrating a measured portion of water with sodium oleate tablets can be considered an approximate value for scale-forming ingredients up to 100 parts per million. Figures for total hardness greater than 100 parts are usually too low because of the interference of calcium and magnesium soaps with the reactions. For very hard waters an estimate of the probable scale can be made from the figures for total hardness, sulphates and bicarbonates. If the sulphates are computed to calcium sulphate and the bicarbonates to calcium carbonate and the two amounts added together, the sum represents the greatest amounts of calcium and magnesium compounds that could be made in the scale, An average between this sum and the total hardness by the sodium oleate method represents more or less approximately the amount of scale-forming material in normal hard waters. The relation may be expressed thus:

$S = 0{\cdot}41\ HCO_3 + 0{\cdot}71\ SO_4 + 0{\cdot}50\ H$ in which S, HCO_3, SO_4, and H represent respectively the amounts in parts per million of scale, bicarbonates, sulphates, and total hardness. If this approximate figure is considered in connection with the amount of chlorides and the possible presence of hydrogen sulphide, as revealed by the odour, or of free sulphuric or hydrochloric acids, comparative classification in respect to steaming qualities may be made accurately enough for reconnaissance work. This method of computation was successfully used in classifying 300 assays of waters made in Iowa and Indiana during a study of boiler-water conditions in those states.

'It may be concluded,' Mr. Dole says, 'that the assay tests for chlorides, sulphates, carbonates, and bicarbonates are as accurate as they need to be for preliminary work. The iron test is likely to give low results in many waters, and more time should be allowed for the oxidation of the iron. The test for total hardness, though laborious in hard waters, yields information valuable enough to make its performance worth while. The calcium test cannot be relied upon more than to indicate whether waters are high or low in calcium, and the results of this test should not be expressed in figures. In reporting results no decimals should be given except in iron, where one decimal place may be reported. The assays should always be made by persons having some knowledge of chemistry and a thorough training in the technique of the tests as well as an appreciation of their limitations'".—R. B. DOLE.—*Engineering Magazine*, Oct., 1910..—[Reprinted from *Engineering News*.]—October, 1910., p. 99. (R. A.)

HYDRAULIC AGITATION IN CYANIDING.—"The agitator consists of a conical bottom tank similar to those used for air agitation, and fitted with a central circulating tube, which takes the place of the air-lift in the air agitators. The tank is also equipped with an inner annular baffle, which provides a calm zone between the baffle and the tank where no agitation takes place. Comparatively clear solution rises to the top from this annular space. A small amount of this clear solution is drawn off continuously through a flexible decanting pipe and supplies the suction of a small centrifugal pump. This delivers the solution under pressure to a nozzle at the base of the central lift tube. By this means all of the sand and slime is kept in constant agitation, without passing any sand through the pump. Immediately over the discharge of the central lift, a damper and distributor are provided which cause the ascending pulp to be distributed in a radial sheet, which assures a moderate and even aeration. In case additional aeration is required, a small stand pipe is attached to the suction of the pump, and equipped with a valve. By opening this, any requisite amount of additional air can be drawn in by the pump and brought in contact with the pulp during its passage through the central tube. The tanks are fitted with central discharge valves and are equipped for either intermittent or continuous process systems of agitating. The great advantages of the system are: (1) Low first cost, small centrifugal pumps being cheaper than air-compressors, and simpler to operate. (2) Economy in power, it being the lowest in power consumption of all systems of agitation. (3) Lower consumption of cyanide, due to the avoidance of excessive aeration. (4) Absence of 'foaming' which is such a serious trouble in air-lift agitators. (5) The apparatus will permit agitation of almost any range of pulp density or size of sand. (6) The current will start after settling for a number of days. (7) Perfect cross-agitation is provided which will distribute the sand evenly over the whole cross section of the settling area.

Tests recently made on a working scale have shown the agitation to be nearly perfect, the extraction being as high as 94% in from 12 to 16 hours on gold ores, and from 83 to 90% on silver ores in 20 hours and under."—*Mining and Scientific Press*, April 20, 1910, p. 260. (K. L. G.)

MINING.

THE MECHANICAL PURIFICATION OF AIR.—"The problem of air purification on a commercial scale has been brought to the front by the wider introduction of the plenum system of heating and ventilation. The removal from the air of the floating particles of dust, soot and other impurities constitutes, especially in densely populated or manufacturing towns, the most serious difficulty in the science of ventilation. The importance of this problem will be fully appreciated when it is borne in mind that with plenum ventilation the fans installed completely change, several times within each hour, the whole of the air in the building which they are ventilating. The volumes of air handled are in many cases enormous. In a public building recently completed in Scotland the heating and ventilating plant has a total capacity of over 212,000 cub. ft. of air per minute.

It is thus evident that where large volumes of fresh air are hourly driven into buildings an infinitesimal degree of impurity in the air will in a very short time cause a dirty and insanitary deposit.

The necessity, however, for the efficient ventilation of public halls, buildings and factories was early recognised, and the fan system of introducing fresh, warmed air into both new and existing buildings was adopted in America and later into England, even though a certain degree of uncleanliness was involved.

The cruder and less efficient methods of air filtration have been superseded by more carefully thought-out and better designed apparatus.

The existing forms of air filter may be roughly divided into five classes: (1) the dry stationary; (2) the wet stationary and the (3) wet revolving filters; (4) the spray nozzle and the (5) revolving spray wheel types.

Perhaps the earliest type of filter used was the dry stationary, which consisted essentially of cheese cloth screens, placed at right angles to the path of the entering air.

Filters of this class, whether employing cotton, jute or cocoanut fibre as filtering mediums, were found to have only a crude cleansing effect upon the air, and after a short period of working, became dirty and insanitary, and thus becoming in themselves a source of impurity, soon defeated their own object.

In addition to the above disadvantage, all filters of this class offer a high resistance to the passage of air. The extent to which a small resistance cuts down the capacity of a centrifugal fan is well known, and the increased power required to drive the fans under these conditions militates seriously against the commercial success of the plant.

These filters have the further disadvantage that they do not affect in any way the humidity of the air handled, and it is now recognised that absolute control of the amount of moisture in the air is an essential qualification in a successful heating and ventalating apparatus.

Dry air produces a more rapid evaporation of moisture from the surface of the human body, and this causes a lower sensible temperature. As a matter of fact the sensible temperature of the body corresponds more nearly to that of the wet bulb of a hygrometer, than to the temperature of the dry bulb. Air at 68° F., containing 50% humidity, feels more comfortable than air at 75° F., having a humidity of only 20%.

With an outside winter temperature of 30° F., and 80% saturation, the air will contain 1·5 gr. of moisture per cubic foot. If, without the addition of moisture, its temperature be raised to, say, 70° F., its saturation will be only 19·5%, since air at this temperature is capable of absorbing 8 gr. of moisture per cubic foot. Under the latter conditions the atmosphere will greedily absorb moisture from the human body.

Apart from its effect on health, the control of the humidity of the air has its industrial value and perhaps its most important applications lie in the textile industry.

Although the westerly winds that blow in from across the wide expanse of the Atlantic are, as a rule, laden with moisture and thus produce atmospheric conditions such as are favourable to the great Lancashire cotton industry, the east winds that often prevail, coming to England across the continent of Europe are comparatively dry, since their short passage over the North Sea does not permit of much moisture being taken up. Thus it is frequently necessary to resort to artificial means for producing the required degree of humidification. The plenum system which gives the right volume of air at the required temperature and humidity, is universally admitted to bring about increased production, stronger, more elastic and more evenly twisted yarn and a better and brighter cloth.

Absolute control of humidity is essential also in cooling and low temperature drying, as will be later explained.

Thus we find the application of water to the stationary filter with the double object of increasing the humidity of the air, and at the same time of cleansing the filter from the impurities that in the ordinary course become lodged therein.

The method usually adopted is to erect vertical screens of fibrous material and to extend horizontally above their upper edges a copper pipe along the bottom of which a series of small, closely patched holes have been drilled. The pipe is then supplied with water whence it issues in a number of fine jets, which are directed upon the filtering medium. Experience has proved that neither has this method a fully-controlled effect upon the humidity of the air dealt with, nor have the jets of water the desired cleansing effect upon the filter.

Unfortunately ventilating installations in many large towns have to deal with a smoky atmosphere in which the chief solid impurity is soot. Minute particles of the latter become lodged in the interstices of the fibrous material, and require for their removal a far more drastic treatment than that obtained by the action of water trickling down the screen by the action of gravity.

Indeed, in order to lesson the high resistance of such filters to the passage of air, the filtering cloth is, where installed, woven with a comparatively wide mesh, and a very considerable percentage of solid matter passes right through the screen without becoming entrapped by the cloth. There is a practical difficulty in getting sooty particles to adhere to a wetted surface for a sufficient length of time to enable them to be flushed away. Any observant dweller in our industrial centres will have noticed small particles of soot dancing freely along upon the surface of water under the influence of small gusts of wind. It was the recognition of these difficulties that brought about the introduction of the revolving screen.

This type of filter, which has been made in units large enough to deal with 50,000 cub. ft. of air per minute, consists essentially of a hollow cylindrical framework, constructed of galvanized steel, the periphery being covered with the filtering cloth. The cylinder is mounted upon a shaft which revolves in gun-metal bearings, the bottom of the drum being immersed in a tank of water. The filter is driven by rope or belt, or by direct coupled motor.

Air, being drawn through the filter by the ventilating fans, enters the latter at the ends and passes out through the wetted material which covers the periphery. The humidity of the air is controlled by regulating the peripheral speed of the filter with reference to the radial velocity of the air, the greater the peripheral speed of the filter with reference to the radial velocity of the air, the more highly saturated the cocoanut fibre is maintained by passing more frequently through the water in the tank, and the humidity of the air is increased in consequence.

This filter, however, is subject in a less degree to the disadvantages of the previously mentioned types, and it has further been found in practice that a portion of the impurities deposited on the surface of the water is picked up again by the filtering mediums as it leaves the tank.

We may now lay down the five main points in an air filter essential to its scientific and commercial success :

(1) The air must be thoroughly cleansed. (2) Its humidity must be under complete control ; (3) the filter must be self-cleansing ; (4) offer small resistance to the passage of air and (5) require the minimum of running attention.

The fourth type of filter referred to at the commencement of this article is that which has as its characteristic feature the use of the spray nozzle. In this apparatus all fibrous material is dispensed with, the designers being firmly of the opinion that in all woven material, if the mesh is fine enough to be effective, not only is the resistance offered to the passage of the air considerable, but that it is practically impossible to properly cleanse the material from the solid impurities that become enmeshed therein. The air to be cleansed passes through a chamber in which are fixed a number of specially-designed nozzles. The water is forced through these nozzles at a minimum pressure of about 20 lb. per square inch, the path through the nozzle is spiral in direction and the water acquiring a circular motion accumulates sufficient centrifugal force to cause it to burst into a fine spray as it leaves the outlet. The water is thus so minutely divided that its intimate intermixture with the air is assured. Although the nozzles have an orifice of only about $\frac{3}{32}$ in. in diameter, each can vent between 3 lb. and 4 lb. of water per minute when supplied at the pressure stated. They are placed in the spray chamber at the rate of two nozzles for every square foot of cross sectional area, and as the usual air velocity through the washer is about 500 ft. per minute, the nozzles deliver about 1½ gallons of water per 1,000 cub. ft. of air per minute. Thus in a plant dealing with 50,000 cub. ft. of air per minute 200 nozzles would be required.

Fig. I. is a diagrammatic representation of the fan and filter combined. The water pressure required by the nozzles is generally set up by a small centrifugal pump which draws its supply from a circulating tank in which the level is maintained by a ball float and valve which regulates the influx of water in accordance with the amount absorbed by the air.

The effect of the almost impalpable mist produced by these spray nozzles in the path of the air is such that it is practically impossible for any solid impurities to pass through the spray chamber without being thoroughly wetted. The greater part of the water falls to the bottom of the washer, carrying with it a portion of the arrested impurities.

A certain portion of the spray, however, is carried along by the air current, and to prevent this free moisture from being borne into the building it is arranged for the air to pass through a separator before leaving the washer.

This separator consists of a series of galvanized steel plates about 18 B.W.G., in thickness, pitched about $\frac{7}{8}$ in. apart. The sheets are bent as shown in Fig. II., which gives a sectional plan of the sheets as arranged in the washer. They are generally fixed about four corrugations deep as shown in the figure. The spray following the direction of the arrows impinges against these plates, the surfaces being usually covered with a thin film of water as far as A, but the projections at A and B formed by the lap of the plates prevent the egress of free moisture from the filter. It will be noted that the wetted surfaces of the separator form an additional washing area equivalent to about 16 sq. ft. for each square foot of cross sectional area of the filter. The separator plates are, owing to the constant dashing of fresh water against them, self-cleansing ; but provision is made for flushing the plates at any time by the simple operation of turning on a water tap.

In the earlier designs of separators of this type ordinary sheets of commercial corrugated iron were used, but the form shown above is much superior.

This is a most ingenious air washer and humidifier, and has met with a large measure of success in difficult situations. The theory embodied in its design is, of course, that given a sufficient volume of water, broken up to the required degree of fineness, and provided that the velocity of the entering air does not exceed that arranged for, no particle of foreign matter can pass through the spray zone without being wetted. Those particles which do not fall to the bottom of the spray chamber with the water are carried along into the separator, and their inertia being increased by the added moisture, strike against the wetted plates and are washed down into the tank.

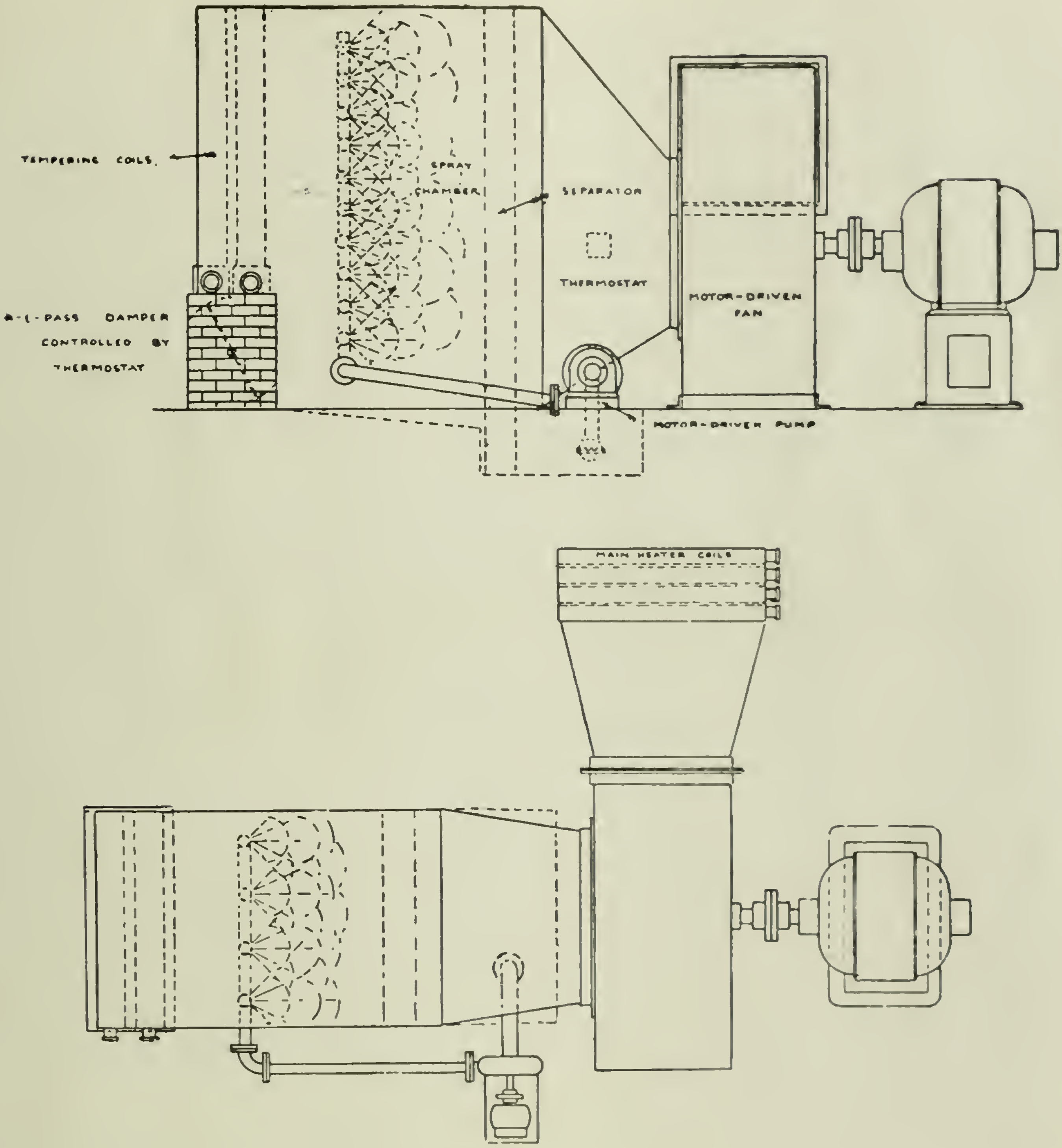


FIG. I.—Diagrams of Fan and Air Filter.

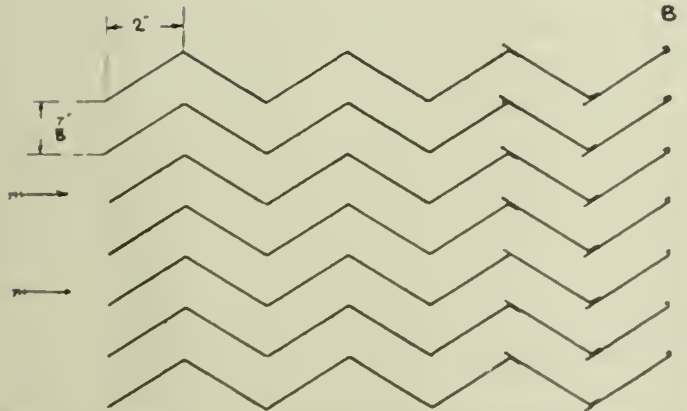


FIG. II.—Horizontal Section through Separator Plate.

There is, however, a danger in breaking up the water too finely. The celebrated 'London fog' consists of just such particles of soot impregnated with moisture as described above, but so small that the effect of their inertia towards causing them to strike against the separator plates when passing through it would be almost negligible. However, in tests with carefully arranged washers of this type, when an ample supply of water has been provided, 99% of lamp black, when thrown into the spray zone, has deposited in the washer.

The spray nozzle filter also gives a very efficient control over the humidity of the air handled. This is chiefly effected by varying the temperature of the air in the spray chamber, thus taking advantage of the fact that the capacity of the air for absorbing moisture varies with the temperature of the former. Thus, if air be raised in temperature from 50° F. to 70° F., its capacity for holding moisture is doubled.

Suppose that it is desired to keep the air in a building at 70° F., with 50% humidity, and that the outside temperature is 36° F., it would first be necessary to raise its temperature either in the spray chamber by using hot water in the nozzles, or by passing the air through steam-heated tempering coils before entering the washer. The air could then be completely saturated at 50° F. when it would be drawn through the main heater coils to raise its temperature to 70° F. Whereas air at 50° F. can only hold 4 gr. of moisture in suspension, when raised to 70° F. it can absorb 8 gr. Consequently air delivered into the building at 70° F. would have 50% humidity.

It might appear at first sight that humidifying the air might result in the condensation of moisture upon the window panes. All danger of this can be obviated, however, by maintaining the temperature in the spray chamber never higher than the mean between the temperature of the room and that of the external air. To take as an example the conditions assumed above, namely: inside and outside temperatures of 70° F. and 50° F., respectively, of the temperature of the spray chamber is less than 60° F. condensation on the windows is an impossibility, since the latter are themselves at a temperature which is a mean between the inside and outside temperatures, and is consequently unable to lower the air to its dew-point.

By control over the humidity of the air its temperature in summer can be lowered considerably, except on such occasions, rare in England, when the atmospheric heat is accompanied by excessively high humidity. It has been proved in practice that an absorption by the air at from ·8 gr. to ·9 gr. of moisture per cubic foot produces a lowering of the temperature of from 7° F. to 8° F.

In cases where excessively high humidities are met with the air can be thoroughly washed without increasing its moisture by using instead of moisture a solution of calcium chloride 28° Baumé. This solution will never give out nor absorb moisture.

If the strength of the solution be increased to, say, 40° Baumé, it will actually extract moisture from the air, and in difficult cases, such as low temperature drying, a strong solution of calcium chloride has been used. It should be added that in modern plants the humidity and temperature of the air are automatically controlled by thermostats suitably placed. The last type of filter enumerated at the commencement of this article is that in which the place of the nozzle is taken by a revolving spray wheel. A diagrammatic sectional view of such a washer is shown in Fig. III.

This washer was designed with a special view to preventing the fine mist in the spray chamber from being carried away through the separator along with the solid impurities that should be eliminated. It is obvious that the cleansing and humidifying effect of the spray becomes small as the latter tends to become stationary with reference to the air, which occurs when the water is carried along in the air current. The increased humidifying effect obtained by opposing the direction of discharge from the nozzles in the previous filter to that in which the air is moving is most marked. The strong current of air, however, soon reverses the direction in which the atoms of spray were originally projected, and owing to their lightness carries them in suspension.

When a fine spray is used for the purposes under consideration, it is necessary in the first place that the water be in motion with reference to the air; secondly, they must at any moment occupy the whole cross-sectional area of the washer in sufficient numbers to effectually cleanse the passing air, and thirdly the water must be broken up to just that degree of fineness requisite to produce the desired humidification. In addition to complying with the above conditions, it is claimed that the spray wheel, throwing off the water in a direction at right angles to the path of the air, gives the former sufficient momentum to prevent its being deflected from its path and carried away.

The tendency of a moving current of air towards entraining the finely-divided drops of water depends upon the pressure exerted upon them.

This pressure in kilo. under conditions of velocity and area obtained in practice is for plane surfaces $P = K\,W A \frac{v^2}{2g}$ where W = the weight of 1 cub. metre of air in kilos, A = the plane surface of the drop in square metre, v = the relative velocity between air and plane in metres, g = the acceleration due to gravity (9·81) K = a numerical coefficient.

According to Grashof, this coefficient is largely dependent upon the size of the surface and is for surfaces of

K=	1·86,	2·04,	2·18,	2·34,	2·51,	2·69
sq. m.	1·00,	0·25,	·5,	1·00,	2·00,	4·00

Thus K decreases with decreasing surface and consequently the more finely divided the spray, the greater is the entraining effect of the air upon it in proportion to the size of its particles.

Guided by recent results as to the resistance offered by the air to the passage of small spherical projectiles, it can probably be assumed that the value of

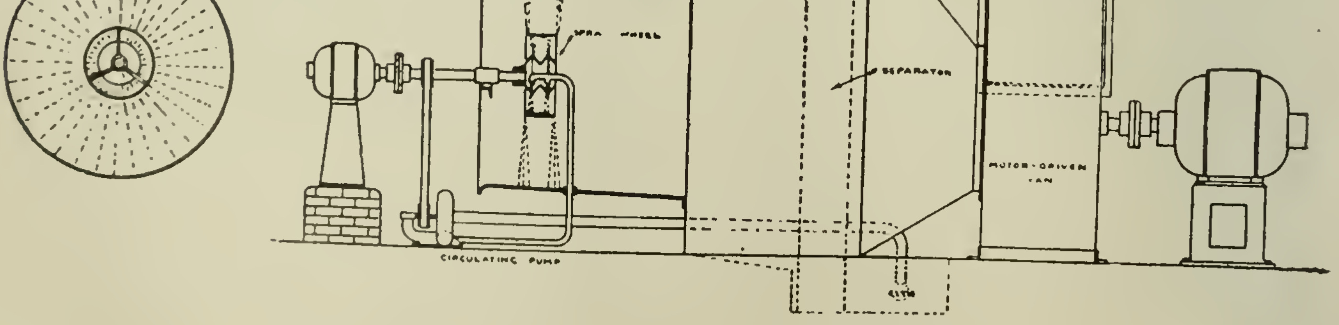


FIG. III.—Revolving Spray Wheel Air Washer.

$K = {\cdot}55$ when the sectional area of spherical drops is from ·1 to ·01 square millimetre.

The air pressure on the floating drops would then be $P = \frac{0{\cdot}55\ W.A.v^2}{2g}$.

The globule of liquid has been assumed to be spherical, but this is only the case when forces continue to act evenly upon it. Whereas the unbalanced air pressure soon flattens the side upon which the pressure is exerted, and by increasing its area increases also the value of K.

This circumstance, which is beyond ordinary calculation, must be neglected, although it tends to increase the pressure on the drop.

When a drop of water is loosened from a fixed point and falls, neglecting friction, its velocity v after a time t and a height h through which it has fallen are connected by the well-known equations $t = \frac{v}{g} = \sqrt{\frac{2h}{g}}$

When a drop of water falling vertically meets a horizontal current of air, the former is deflected from its vertical path. If the horizontal pressure on the drop, as expressed in equation (2), commences at the moment of its fall and is equal to its weight, the drop falls at an angle of 45° with the vertical, since its vertical and horizontal accelerations are equal. If under the same conditions the air pressure is several times as great as the weight of the drop, the directions of fall may approach the horizontal. Should the drop, however, have fallen vertically through some distance before meeting the side current, its deflection from the vertical is considerably less, since in equal time intervals the vertical velocity is greater than the horizontal.

Thus the danger of the drop being carried away in the air current is very much less. This is made clear by the diagram in Fig. IV. It is on the above facts that the design of the spray wheel air washer is based.

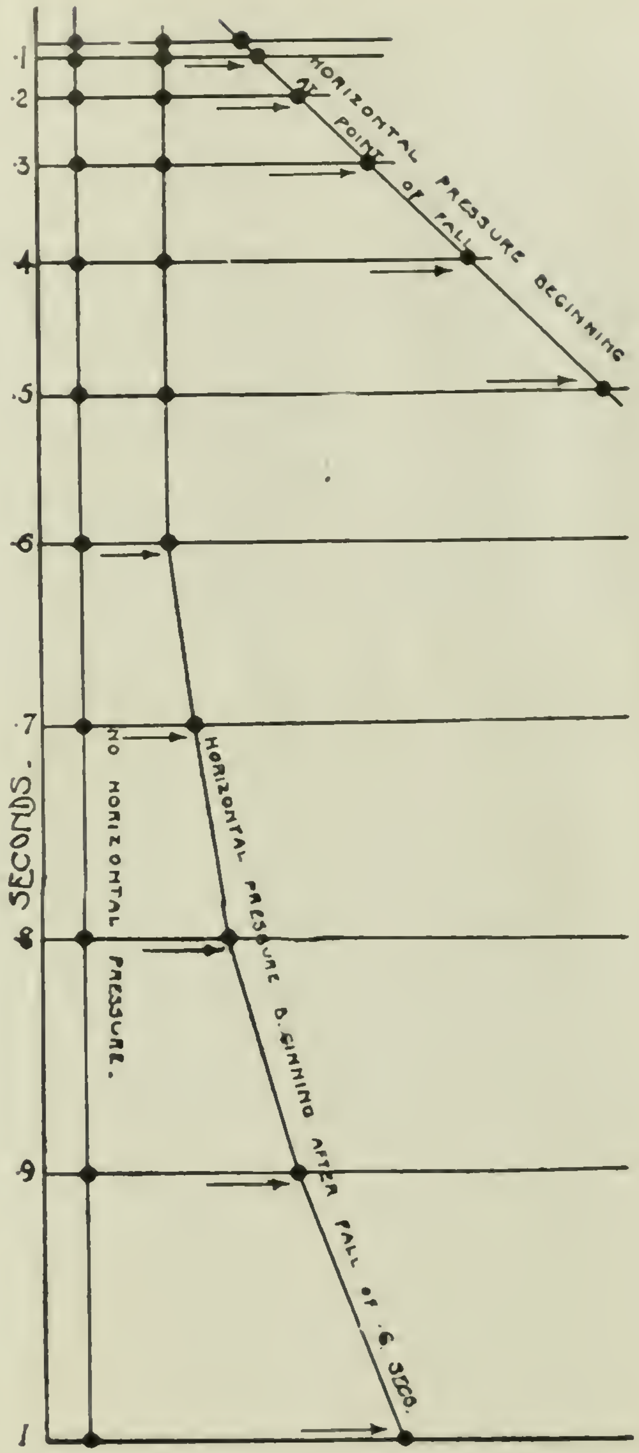
Referring again to Fig. III., the spray wheel is mounted on a turned shaft supported in two bearings within a tubular casing. The shaft is direct coupled to an electric motor, the speed of which can be varied by field regulation. The water supply to the wheel is controlled by a small belt-driven centrifugal pump which raises the water from the circulating tank. The water is thrown at a high velocity from the periphery of the spray wheel which travels at a lineal speed of not less than 6,000 ft. per minute, and consequently its deflection by the air current is practically nil. The 'carry over' of free moisture into the separator due to the water splashing against the sides of the casing is very small in amount.

Any floating dust or other impurity in the air, on entering the spray zone is dashed and hammered against the sides of the casing, whence it is washed down by gravity into the tank below.

The fineness of the spray, and consequently the degree of humidity given to the air is regulated by varying the speed of the wheel, and the quantity of water in circulation is controlled by a handcock.

The resistance offered to the air in passing through the washer is negligible, and its self-cleansing properties are obvious, indeed this air purifier conforms in a remarkable degree to the conditions laid down earlier in this article.

One of these washers was supplied recently in connection with the plenum ventilation plant for a large public building in Glasgow, where the problem of cleansing the air was aggravated by the necessity for taking in the air at the roof level. In less than five minutes after starting up the water in the circulating tank was discoloured and a film of soot was seen to be floating on its surface.


·1
·2
·3
·4
·5
·6
·7
·8
·9
1
SECONDS.
HORIZONTAL PRESSURE BEGINNING POINT OF FALL
NO HORIZONTAL PRESSURE.
HORIZONTAL PRESSURE BEGINNING AFTER FALL OF ·6 SECS.

Horizontal Pressure — Vertical Pressure.
FIG. IV. — Deflection of Falling Drops.

The introduction of plenum ventilation into England has caused the springing up of a new and important industry, and that one of its most serious problems has been tackled in a thorough and scientific manner, and with a growing measure of success has been sufficiently indicated in this article.

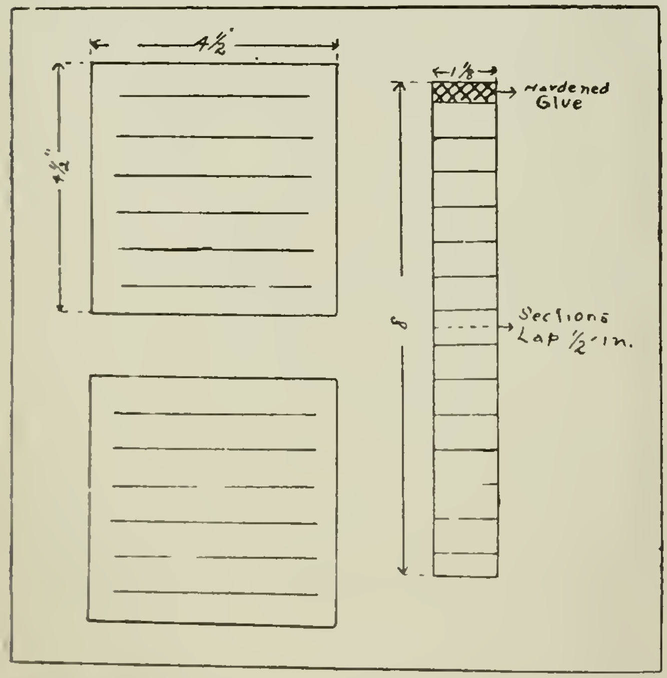
The introduction of pure, warm, humidified air into public buildings by the scientifically-designed apparatus of the present day, enables the city dweller to enjoy one of the delights of a country walk on a summer's evening, after a heavy downpour of rain has refreshed and purified the air.

In conclusion, the writer wishes to express his indebtedness to Mr. Julius Hanna, whose name is identified with original research work in the science of ventilation. It was owing to the courtesy of Mr. Hanna in his capacity as manager for Messrs. Musgrave & Co., of Belfast, that the writer was enabled to make a series of interesting tests of the 'Rainbow' purifier manufactured by his firm."—C. L. BROWNE.—*Cassier's Magazine*, Oct., 1910, p. 564. (R. A.)

ECONOMIC TAMPERS IN JOPLIN DISTRICT.—"An innovation in the blasting practice in the sheet ground mines of the Joplin district has come about with the introduction of the Economic tampers, a recent invention. These tampers have been used by a number of the mining companies with success, and they promise to effect a change in the mining costs, especially in the use of powder. Among the operators who have given them a trial are the Little Princess Co., the American, Genessee and Oakwood.

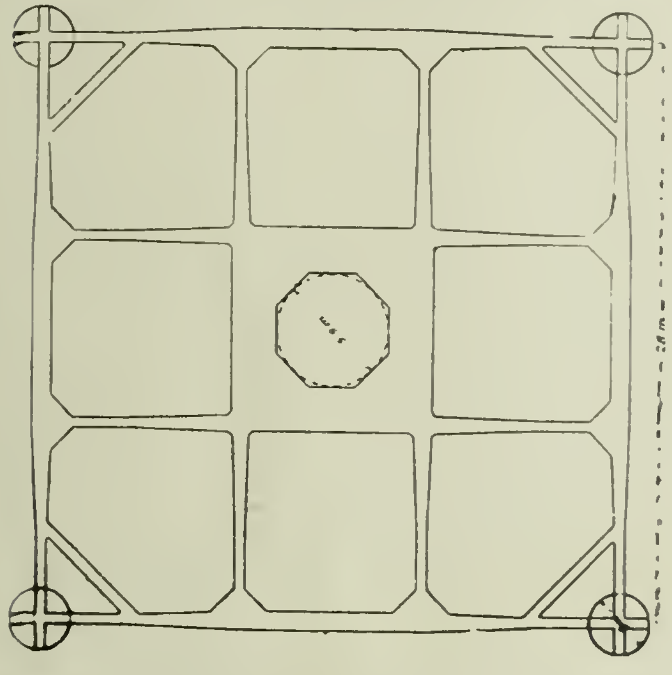
The construction of the tamper is extremely simple, consisting of a card-board tube covered with paper and filled with sand, the sand used being the fine material from the tailing mills, screened to a 10-mesh. The tube is made of two pieces of cardboard, each $4\frac{1}{2}$ in. square. Each piece is slotted and when bent to form a tube, the slots extend entirely around the tube. The two pieces are held in place by wrapping paper which is glued together, two pieces being used so as to form a weak place in the center of the tube, so that the cartridge may bulge and swell to fill the hole when pressure of the tamping rod is applied. The paper is crimped over the end of the tube to keep in the sand. After the tube is filled with dry sand, a thin layer of glue is applied, making a hard cake $\frac{1}{4}$ to $\frac{1}{2}$ in. thick, so that when the tamper is placed in the hole the tamping bar strikes the caked top and will not break the paper and lose the sand.

In order to make blasting effective, the powder must be confined in the hole, but Joplin operators have not been observing this requirement as carefully as they should, and hence have crowded in a greater amount of powder than was necessary, if proper tamping material were used. This greatly increased the amount of powder, which is already a large item of expense especially in the sheet-ground mines, where the ground is very hard, and requires much blasting. Hitherto the operators have been handicapped by the lack of proper material for tamping, the only thing at hand for use being rock and gravel, which are not only poor material, but very dangerous. Hence the operators usually preferred to ram the hole full of powder, thus using much more than was really necessary for breaking the hole. With the introduction of the Economic tamper a new and successful material has been found with which to fill up the hole, after sufficient powder has been put in, making it much safer, as well as more economical. A box of tampers costs \$1·50, while a box of powder costs \$5·25, saving thus on a box \$3·75. The use of tampers instead of gravel means fewer holes lost, caused by the fact that the fuse is never cut off by the use of tampers, which often happens in using gravel. The paper and fine sand packs the powder securely into the holes, and yet does not wear or cut the fuse. Danger of premature explosions is much less, as the soft paper and sand of the tampers cushions off any percussions of the powder.



One of the most fruitful sources of profit and success in the use of the tampers is in squibbing. After drilling, the holes have to be squibbed, three to five sticks of powder being commonly used for this purpose. Each hole is usually squibbed twice. By properly confining these sticks of powder with tampers it has been found necessary to use only two to four sticks of powder, effecting a great saving in each shot. By such practice the largest amount of ground is broken with the smallest possible use of powder and scarcely a hole is lost, whereas in the mines of the district where former methods were employed, it is common to see the face of drifts with holes one to three ft. deep, which have not been broken. Where every cost of mine management has to be carefully reckoned, as is often the case in the sheet-ground mines of the district, in order to make a margin of profit above mining expenses, this means a very considerable item of economy, equivalent in many cases to $2\frac{1}{2}\%$ reduction in royalties."—OTTO RUHL, *The Mining World*, April 23, 1910, p. 852. (C. B. S.)

SHAFT-SINKING IN DANGEROUS GROUND.—"In our January issue we gave some information concerning the sinking of a new shaft at a coal mine in the north of England through treacherous ground, where the support for underhung shaft-lining was provided by a ring of reinforced concrete. An even more notable case of building the support in this manner is to be found at a colliery belonging to the Société des Mines de Houille de Marles, in France. Forty years ago one of the shafts at this colliery was practically destroyed by an inrush of water, and until last year it was considered impossible to re-open it owing to the broken condition of the ground at the mouth. It was not until M. Coignet of Paris came forward with a proposition involving the provision of a ferro-concrete structure that the difficulty was solved. As a matter of fact the structure partakes more of the nature of a system of steel girders than of ordinary reinforced concrete. The structure is supported on four masonry foundations arranged at the points of a square. These are 3 metres in diam. and are situated 28 metres apart. Upon these rest four main girders, and upon the latter rest four other intermediate girders. The platform for suspending the shaft-lining is built round the centre opening. The masonry foundations were built sufficiently far from the shaft-mouth to ensure stability. The four main beams are buried in the ground, as also are parts of the intermediate girders, but they are all made strong enough to do without the uncertain aid of the earth support. The depth of the beams is therefore made greater than would otherwise be the case, the main being 3 metres and the intermediate 2·7 metres. The steel framework of the beams is composed of round bars varying from 21 to 23 mm. diam. There are 28 such bars and their ends are bent upward gradually toward the points of support so as to resist shearing stresses. The web of the beams is made in a manner similar to that of an ordinary steel lattice-girder. The internal diam. of the suspended shaft-lining is 5·9 metres. The total weight of the ferro-concrete structure is 300 tons, and the weight of the underhung lining is 200 tons. The small diagrammatic plan explains the shape of the structure.—*The Mining Magazine*, April, 1910, p. 293. (C. B. S.)



MISCELLANEOUS.

Protection against Rust.—"The clean iron surface is painted first with ferric chloride solution, then with a soluble aniline salt preferably the sulphate, which becomes intimately mixed with the basic oxychloride, and finally with chromic acid which produces an adherent film of aniline-black on the surface. The several ingredients may, however, be mixed together with linseed oil and the iron painted therewith." Prof. A. Lang. *Times Engineering Supplement*, Dec. 7, 1910. (J. M.)

South African Dredging. "The Mynpacht Gold Dredging Co has been formed in Melbourne to work an area of 200 acres on the Thabina River, Northern Transvaal. The company consists of 50,000 £1 shares, of which 30,000 are fully paid, and 20,000 contributing, paid up to 5s. The property is one of the options secured by Mr. J. C. Watson on behalf of the South African Dredging Syndicate. Mr. P. W. Tewksbury, whose name is well known in connection with bucket dredging enterprises in Victoria and New South Wales, has reported on the lease, and states that it is the richest alluvial deposit suitable for dredging he has ever examined. His tests gave the net monetary return per cubic yard of ground tried at 3s. 10d., as compared with 2s. 2d stated by Mr. Watson. The working costs are approximated at 4d. per cub. yard.

In order to ensure efficiency, the directors of the Mynpacht Company have decided to obtain the dredge locally, and also to send the whole staff from Australia.

The contract for the dredge, which will have a capacity of 30,000 cub. yards per week, has been let to Messrs. C. Ruwolt & Co., engineers, of Wangaratta, Vic. The pontoon will be of steel throughout. The dredge is to have a special turbine pump and tandem compound engines of the latest type, and the buckets are to be direct coupled, without connecting links. The whole plant will be fitted with electric light. The approximate cost of the dredge, f.o.r. Melbourne, will be £14,000. Mr Tewksbury, who is figuring upon the dredge turning over a minimum of 15,000 cub. yards of ground per week, is confident this plant will provide a useful object lesson to South African mining men, in showing what Australian manufacturers can do. The construction of the dredge should be completed in about six months."—*Australian Mining and Engineering Review*, January 5, 1910. (C. B. S.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

Ore Mining Methods. By W. R. Crane, with 56 illustrations. Demy 8vo. 219 pp. (Wiley & Son.) 12s. 6d. net.

"It is in the underground methods of working and handling ore, that great scope is at present possible for improvements and economies in many ways. Underground we are in the dark in many respects, we deal with unknown forces, hence we cannot exactly calculate the supports, etc., needed, too much money may be spent, or to little, thereby perhaps increasing the risk to life. The actual ore mining methods are essentially the practical side of mining, and for each separate ore body, the most suitable method of working, can only be determined by actual practical experience. Experiments in this department are therefore necessarily very costly, yet it would often pay if such were carried out. Miners are most conservative people, each mining district has its own method of mining which in many cases could be improved upon. Unfortunately there are no men, like some of the greater scientists, who take up the study of mining for a hobby, as amateurs, who would have the time and means of carefully studying and experimenting in the various mining problems in different parts of the world, at the mines themselves. The average mining man has therefore to gain, what information he can from books, and it is fortunate that he has such books as the one under review to turn to. In 'Ore Mining Methods,' the author has given excellent general descriptions of the various methods of stoping as employed, chiefly in the mines in America, with their relative advantages and disadvantages. Many really practically points are missed, *e.g.* under the heading 'Props,' p. 8, the author states that 'in order to increase the bearing surface caps are often provided, which consist of short lengths of plank placed between the ends of the props and roof or floor. This may be one reason for the use of head boards, but an equally important reason for their use is to prolong the life of the prop itself, and hence economise in timber. Again, under the heading of 'Methods of Handling

Ore in the Stopes,' p. 15, in speaking of chutes the author seems to be undecided as to what angle the broken ore will actually run down a curved sheet at metal chute. The angle depends to a certain extent on the size of the chute and nature of the broken ore, but under the most favourable conditions it will be found that the ore will not run at a flatter angle than 27 degrees. The author concludes with a chapter on 'Cost of Stoping" at various mines. For purposes of comparison these are useful, when the type of labour, the actual prices paid per lb. or ton of stones are given, etc., but are of little use for comparing results with other mines in various parts of the world when only the cost per ton of the various items of expenditure are given. On the whole the book will well repay studying, especially for students reading up for college examinations. (G. H.S.)

Abstracts of Patent Applications.

(C.) 546/09. Charles Harrison Dixon (1), William Beaver (2). Mechanical finger for receiving and holding gravity stamps when idle. 25.11.09.

This application is for an improvement in fingers for holding gravity stamps when idle, consisting of a separate end for the finger which is worked into an incline plane fixed in a casting in the end of the finger thereby raising the stamp high enough so that the cam will not touch it when the camshaft is revolved. This does away with the ordinary cam stick.

(C.) 36/10. Frederick George Fell. Stamp safety clamp. 4.2.10.

This application is for a toggle joint clamp for suspending stamp stems and the like, the clamp being hinged on one side and having a hook link on the other side for quick adjustment and removal.

(C.) 45/10. Frederick Benjamin Adolphe Kirsten. Improvements in apparatus for recovering diamonds. 10.2.10.

This application refers to the use of an endless belt, mounted after the manner of a Frue vanner, the surface of which is covered with a coat of grease. As is well known diamonds will adhere to this greased belt and means are adapted for continuously removing the valuable deposit and for supplying the grease required.

(C.) 60/10. J. R. Thurlow. Improvements in or relating to apparatus for classifying crushed ore products. 21.2.10.

This application refers to a device with various modifications for classifying crushed ore products by hydraulic separation of coarser or heavier particles from finer or lighter material.

The device consists essentially of a pipe connected to the bottom of an ordinary conical classifier, this pipe being perforated and water jacketed. The flow of the classified product is regulated at the bottom by a conical plug adjustable so as to govern the size of the annular discharge opening.

(C.) 69/10. C. F. E. O. Schmitt. Improvements in tube mill feeding means, also applicable in part to spiral pumps. 24.2.10.

This application for tube mill feeding means comprises in combination a casing providing a receiving chamber and an elevating chamber with the usual spiral vane. The receiving and elevating chambers are separated by a partition wall, having an aperture, and a vane set at an angle with the direction of rotation whereby the pulp feed is led into the elevating chamber. The whole apparatus revolves with the mill on its axis, and may be provided also with an independent central opening for feeding pulp directly through this axial aperture.

(C.) 247/10. C. H. Wilkins (1), J. Wilshir (2). Improvements in pipe and like couplings.

This invention relates to couplings for rapidly connecting lengths of pipe whether metal pipe or hose, or for connecting such pipe to a machine or for similar purposes.

The inventors claim, in the coupling, the combination of a male and female element, forming between them a longitudinally tapering annular chamber, with a resilient washer or packing ring positioned within the said chamber to make contact with the adjacent surfaces of both elements and to be movable longitudinally in the said chamber; the said washer to be also exposed to the pressure of the fluid passing through the coupling in such a manner as to force it towards the smaller part of the annular chamber and thereby cause it to make a tight joint between the coupling elements.

(C.) 473/10. E. H. Martin. Improvement in discharge means for stamp mills and other crushing machinery.

The device consists of horizontal bars (a kind of grating) traversing the discharge opening, spaced apart to pass material of the required size, their upper surfaces presenting a series of steps collectively inclining upwardly and outwardly from the stamp or equivalent.

(C) 489/10. M. de R. de Colombier (1), J. Clement (2). Improvements in separating metallic particles from their admixtures.

This application is for an apparatus for separating metallic from non-metallic substances by means of an electrical discharge passing between two metal plates arranged face to face, one being covered by an insulating materal on which the powdered ore is placed.

(C.) 535/10. George Mitchell. Improvements in and relating to process and apparatus for the treatment of ores. 29.10.10.

This application refers to a smelting furnace of the reverberatory copper type in which a portion of the hearth is replaced by a converter or converters, in order to utilise the heat generated during conversion to assist in the smelting of further portions of the charge. The smelting hearth is at a higher level than the converter and graded towards it, so that the metal drains to the converter as it melts. Details are given of its application in various forms, and with either carbonaceous or oil fuel.

(C.) 536/10. Ernest James Hume (1), Walter Reginald Hume (2). Improvements in concrete pipes, columns, girders, blocks, and other structural material, and method of and means for manufacturing the same. 29.10.10.

This invention relates to a method of and means for manufacturing concrete pipes, columns, girders, blocks, and other concrete structural material generally, whether reinforced or otherwise, but more particularly to the manufacture of pipes for water service, drainage and other purposes, as well as hollow blocks and other hollow structural details. The object of the invention is to more effectively secure the packing of the material from which the pipes and other articles are made and to reduce the cost of manufacture.

In this invention centrifugal force is made use of for packing the particles of the cement and other material together, the method of procedure being such that by making use of centrifugal force in the manner described in the specification, the outside or surface of the pipe or other article to be moulded is completed first, the inner portion being gradually filled in or built up until the complete pipe or other article is made, the reinforcement when used being first placed in position and then automatically bedded into the structure during the progress of manufacture.

(C.) 538/10. Hans Charles Behr (1), Charles Frederick Eduard Oscar Schmitt (2). Improvements relating to the sub-division of streams of pulp and other liquid. 1.11.10.

This application is for apparatus for sub-dividing a stream of pulp or liquid which consists in forming the same into a circular sectional stream, flowing it in a vertical direction whilst progressively reducing its cross sectional area and then dispersing it symmetrically from an axis to form a sheet, and splitting said sheet. This, in the apparatus specified, is accomplished by placing a conical vessel, with an open bottom, concentrically into a larger conical vessel, the liquid travelling downward in the smaller cone and then upward in the annular space between the cones, and finally falling over the level upper lip of the outside cone, with receptacles dispersed in a ring under and around the lip. The liquid may be mechanically stirred in the inner cone.

(C.) 543/10. Carl Nill (1), August Brandes (2). Improvements in and relating to the treatment of liquids. 5.11.10.

This is a process of treating water for boiler and similar purposes by passing it rapidly over a clean aluminium surface and so producing a coagulating effect from the aluminium removed or hydrate formed.

(C.) 546/10. Charles Cuthbert Freeman. A grinding pan discharge and classifier. 8.11.10.

This patent refers to the internal application to a grinding pan, of an adjustable outlet consisting of a pipe or tube which may be set at any required distance from the centre of the pan. Advantage is taken of the concentric segregation of particles according to their mass due to the centrifugal action produced by the working of the pan for obtaining the discharge of different grades of pulp.

(C.) 561/10. Reinhold Stemback. Improvements in grinding mills and the like. 17.11.10.

This invention relates to subdividing a tube mill into two parts by means of a vertical partition, into a preliminary grinding chamber and a finishing chamber. The connection between the two chambers is through peripheral openings round the end of the first chamber discharging into a circumferential chamber carrying a circumferential sieve. Large openings connected to the spaces inside and outside the sieve connect respectively to the the primary and finishing portions of the mill, the first for the return of the oversize back to the preliminary chamber and the second to conduct the undersize which has passed through the sieve to the finishing chamber.

(C.) 570/10. Peter Andrew Malone. Improvements in lubricating the bearings of cam shafts. 22.11.10.

This relates to a means of lubricating the bearings of cam shafts by means of a pad (or pads) consisting of a porous bag or envelope containing solid or semi-solid grease, these bags being merely laid on the journal, the lubrication being effected by the grease exuding through the pores of the texture without waste.

(C.) 575/10. James Valentine Coleman. Apparatus for separating mineral bearing matter from streams of water. 25.11.10.

This application is for an endless sluice, composed of pliable material provided with pockets and ridges and suspended between endless travelling belts to which progressive movement is imparted by means of carrying pulleys.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 562/10. Rollo Bowman Ballantyne (1), Martin Wagner (2). An improved sprayer for liquid solutions and the like. 17.11.10.

(P.) 563/10. Gaston Jacquier. Improvements in the precipitation of metallic oxides in solution. 18.11.10.

(C.) 564/10. Woldemar Hommel (1), Henry Thomas Durant (2). Improvements in or relating to the extraction of metals from their ores 18.11.10.

(P.) 565/10. John Wilson. A new and improved match (known by miners under the name of "cheesa stick") used for lighting fuse. 18.11.10.

(P.) 567/10. Bertram Poidevin Richardson. Improvements in pipe and similar couplings. 21.11.10.

(P.) 568/10. Henry Arthur Goddard. An improved mode of building in concrete and apparatus therefor 21.11.10.

(P.) 569/10. William Colin Hazelhurst. Tube mill and liner. 21.11.10.

(C.) 570/10. Peter Andrew Malone. Improvements in lubricating the bearings of cam shafts. 22.11.10.

(P.) 571/10. Sir Joseph Wilson Swan (1), James Alfred Kendall (2). Improvements in and apparatus for the manufacture of alkaline cyanides. 22.11.10.

(P.) 572/10. John Coutts Shaw. A new or improved device for detaching from their spindles the pointers of steam pressure gauges the hands of time pieces, and the like. 21.11.10.

(P.) 573/10. Ernest Holroyd Martin. Apparatus for sorting ore prior to milling. 24.11.10

(C.) 575/10. James Valentine Coleman. Apparatus for separating mineral bearing matter from streams of water. 25.11.10.

(C.) 577/10. Thomas Gaskell Allen. Improvements in or relating to acetylene gas generators. 25.11.10.

(P.) 579/10 Charles Graham Stone. Improved septic tanks for treatment of sewerage. 26.11.10.

(P.) 581/10 Joseph Gouldie Method of automatically collecting and extracting dust and explosives fumes from the atmosphere made by rock drilling machines and blasting underground. 28.11.10.

(P.) 582/10. Frederick Victor William Swanton. Improved apparatus to produce power by means of weighted levers and eccentric epicycloidal motions. 28.11.10.

(P.) 585/10. William Berthelson. An improved coupling for connecting hoses to pressure mains and the like. 28.11.10.

(P.) 584/10. Joseph William Poester (1), Edward Henry Johnson (2). Improvements in apparatus for separating the solids from the liquid of ore pulp. 23.11.10.

(P.) 585/10 George Anderson Robertson (1), Charles Thomas Stephens (2). Improvements appertaining to the mortar boxes of stamp mills. 29.11.10.

(P.) 586/10. Oswald Robert Rushton Hampson (1), Henry Algernon Hampson (2). Improvements in means for actuating shaking chute conveyors. 30.11.10.

(P.) 587/10. Otto Lenz (1), Joseph Amos Mills (2). Improvements in the fixing of hammer heads and other similar tools to their shafts or handles. 30.11.10.

(C.) 588/10. Charles Kratt. An improved method of and apparatus for heating water by electricity. 2.12.10.

(C.) 589/10. George Johnston. Improvements in or relating to crushing or grinding auriferous quartz and other hard ores or substances. 2.12.10.

(C.) 590/10. George Johnston. Improvements in or relating to filtering apparatus. 2.12.10.

(C.) 591/10. Harry Pauling. An improvement in the process of treating aqueous nitric acid and the like with sulphuric acid and other hygroscopic substances. 2.12.10.

(C.) 592/10. Harry Pauling. Improved method of concentrating aqueous solutions of nitric acid. 2.12.10.

(C.) 593/10. Nikola Tesla. Improvements in fluid propulsion. 2.12.10.

(P.) 594/10. Arthur Ernest Walker (1), James Cramond Walker (2). Safety device for mechanical haulages. 2.12.10.

(P.) 295/10. Henry Crawford Boyd (1), Robert Henderson (2). Improved concrete vibrating moulding machine. 2.12.10.

(C.) 596/10. Otto Stehang Tonnesen. Improved apparatus for sampling slimes and other liquids. 5.12.10.

(C.) 597/10. Louis Lumiere. Improvements in acoustic instruments 5.12.10.

(C.) 598/10. William Henry Leonard. Improvements in or relating to quick couplings. 5.12.10.

(C.) 599/10. Daniel Shaw Waugh. Improvements in or relating to rock drills. 6.12.10.

(C.) 600/10. Thomas Brown Baillie. Improvements in suction cleaning apparatus. 6.12.10.

(P.) 601/10. James Leslie Mitchell (1), William Wright Greathead (2). Improvements in safety catch mechanism for mine skip cages and the like. 6.12.10.

(P.) 602/10. Clifford Arnold Maclean. Contrivance for the prevention of miners' phthisis. 6.12.10.

(C.) 603/10. Frederick Stubbs. Improvements in safety amalgam receptacles. 8.12.10.

(C.) 605/10. Carl Ludwig Valentin Zimmer. Process for increasing the heat resisting properties of bituminous or tarry materials. 9.12.10.

(C.) 606/10. Arthur Thomas Collier. An improved manufacture of flexible material from india rubber and the like. 9.12.10.

(P.) 607/10. John Archibald Maclachlan (1), Andrew Trimble (2). A new sewerage apparatus for mines and the like. 9.12.10.

(P.) 608/10. Alfred George Newkwy Burden. Improvements in guides for the stems of the stamps of stamp mills. 9.12.10.

(P.) 611/10. Paul Arnold von Wildenrath. Improvements in rope grips. 10.12.10.

(P.) 612/10. Louis Lumiere. Improvements in acoustical instruments. 10.12.10.

(P.) 613/10. Paul Millington Jerrard. Improvements in and relating to flying machines, airships, and the like, and in apparatus therefor partly applicable also to other motor actuated machines. 12.12.10.

(P.) 614/10. Hans Peter Rasmussen. Composition for the use of electrical insulating and other commercial purposes. 12.12.10.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

Bass, K. C., *l/o* Louw's Creek; Rhodesia Esperanza G. M. Co., Ltd., Umsweswe, Rhodesia.
Bissett, A. C., *l/o* Scotland; P. O. Box 65, East Rand.
Cullen, W. *l/o* Modderfontein; c/o British S.A. Explosives Co., Ltd., 612 Salisbury House, Finsbury Circus, London, E.C.
Davies, Edward, *l/o* England; Antelope Mine, Matopos, Rhodesia.
Dodd, M., *to* P. O. Box 2486, Johannesburg.
Evans, Lewis. *l/o* Gwelo; 16 Lonrho Buildings, P. O. Box 547, Salisbury, Rhodesia.
Ford, S. H., *l/o* Bristol; 5, The Manor Way, Blackheath, London, S.E.
Gilbert, T. W., *l/o* Johannesburg; S.A. Tin Mines, Ltd., Doornhoek, P. O. Naboomspruit Station.
Isaacs, R. McI., *to* c/o C. Dixon, P. O. Box 305, Bulawayo.
Jack, W. D. *to* P. O. Box 54, Cleveland.
Jolly, T. S., *l/o* Jubilee Mine; City Deep, Ltd., P. O. Box 1447, Johannesburg.
Lindhorst, R., *l/o* Cleveland; Kingwilliams Town, C.C.
Mackay, A. N., *l/o* Mexico; Caley, Leopold Road, Wimbledon, London, S.W.
Mackilligan, H. R., *l/o* W. Australia: The Toll, Horeham Road, Sussex, England.
Mather, T. W., *l/o* W. Africa; 810, Earlham Street, Pasadena, Cal., U.S.A.
Morrison, S., *to* City Deep, Ltd., P. O. Box 1411, Johannesburg.
Mumford, F. T., *to* P. O. Box 244, Salisbury, Rhodesia.
Murdoch, R., *l/o* Johannesburg; Bantjes Consolidated Mines, Ltd., P. O. Box 2, Florida.
New, W. Stanley, *l/o* Germiston; Green Hill Park, Evesham, England.
Parsons, C. E., *to* P. O. Box 213, Bulawayo.
Pearce, John, *l/o* Devon; The Laurels, Willsbridge, near Bristol, England.
Robertson, J. R. H., *l/o* East Rand; Fern Cliffe, North Mount Vernon, near Glasgow, Scotland.
Robinson, C. A., *to* P. O. Box 27, Jeppestown.
Robinson, R. W., *l/o* Penhalonga; East Rand Propy. Mines, Ltd., P. O. Box 220, East Rand.
Sawyer, R. E., *l/o* East Rand; Cinderella Consolidated, Ltd., P. O. Box 75, Boksburg.
Stanley, Prof. G. H., *l/o* Johannesburg; 2 Highfield Road, Edgbaston, Birmingham.
Wade, R. A., *l/o* Bulawayo; The Dawn Mine, Turkois P. O., Gatooma, Rhodesia.
Wessels, J. J., *l/o* Nigel; Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg.
Wilmoth, L. J., *l/o* Barberton; c/o A. H. Wilmoth, Solicitor, Brunton Chambers, Collin Street, Melbourne, Australia.

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Proceedings
AT
Ordinary General Meeting, February 18, 1911.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the South African School of Mines, on Saturday evening, February 18th, Dr. James Moir (President) in the chair. There were also present :—

51 Members : Messrs. C. B. Saner, W. R. Dowling, Tom Johnson, E. J. Laschinger, H. A. White, W. A. Caldecott, E. H. Johnson, R. G. Bevington, A. McA. Johnston (Members of Council), E. L. Adams, S. Beaton, G. H. Beatty, W. Beaver, A. J. Bowness, J. Brown, J. Chilton, M. H. Coombe, W. M. Coulter W. J. Creasey, M. Dodd, M. J. Doyle, A. L. Edwards, R. Gascoyne, W. L. Hamilton, J. H. Harris, A. B. Inglis, A. J. Johnson, W. W. Lawrie, G. A. Lawson, H. S. Macgregor, L. Marks, H. Meyer, Prof. J. Orr, E. Pam, C. S. Parry, J. F. Pyles, E. Roberts, K. K. H. Sartorius, A. Schwarz, G. H. Smith, Ralph Stokes, J. A. Taylor, W. A. C. Tayler, A. Thomas, C. F. Thomas, F. W. Watson, J. Watson, E. M. Weston, and A. Wilkinson.

17 Associates and Students : Messrs. H. Abao, J. Gibson, T. W. Gilbert, J. S. Grace, A. King, H. G. Kirkland, L. T. Leyson, H. J. v. d Merwe. H. A. Pattle, F. J. Pooler, R. Sawyer, H. Stadler, A. H. Thomas, J. Thorlund, I Tom, E. J. Wiseman and A. L. Wright.

30 Visitors, including Sir Wm. Preece, K.C.B., F.R.S., and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as printed in the January *Journal*, were confirmed.

NEW MEMBERS.

Messrs. R. G. Bevington and Tom Johnson were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows :—

DICKSON, GORDON FRASER, London and Rhodesian Mining and Land Company, Limited, Lonrho Buildings, Salisbury, Rhodesia. Mining Engineer.

HAWLEY, THOMAS, Commercial Hotel, Salisbury. Rhodesia. Miner and Prospector.

IZOD, EDWIN GILBERT, Messrs. H. Eckstein & Co., P. O. Box 149, Johannesburg. Mechanical Engineer.

LAURIE, ROBERT ANTHONY, Crown Mines, Limited, P. O. Box 158, Fordsburg. Shift Boss.

LILLY, ALEXANDER, Princess Estate and Gold Mining Company, Limited, P. O. Box 114 Roodepoort. Mill Manager.

The Secretary : Since the last meeting of the Society the following have been admitted by the Council :—

As Associates.—

INNES, CHARLES WILLIAM ROSE, Knights Deep. Limited, P. O. Box 143, Germiston. Cyanider.

JAMESON, JOHN JULIUS IRVINE, P. O. Rooiberg, *via* Warmbaths. Mine Manager.

TAMPLIN, ERIC HORNBY, Ferreira Deep, Limited. P O. Box 5977, Johannesburg. Reduction Works Learner.

TAYLOR, MICHAEL, Princess Estate and Gold Mining Company, Limited, P. O. Box 114. Roodepoort. Cyanider. *(Transfer from Student Roll.)*

VAWTER, J. E., Knights Deep, Limited, P. O. Box 143, Germiston. Mining Engineer.

WELSFORD, HUBERT MEREDITH, Knights Deep, Limited, P. O. Box 143, Germiston. Cyanider.

WILSON, GEORGE CALDWELL, Mysore Gold Mining Company, Limited, Marikuppam, Mysore State, S. India. Chemist.

GENERAL BUSINESS.

The President : I have some more slides to show you in connexion with the mine-air question. They were done in order to see if the fog of upcast shafts (which is due to rarefaction) contained any silica particles. Mr Hildick Smith got me the samples by exposing glucose slides for 15 minutes underground near the upcast shaft of the Ferreira G. M.—a mine which contains much timbering. The top one shows no silica but a fungus filament and many oval spores and a few cocci, also a large flat cell which may be derived from skin or epithelium. The lower one shows a large piece of decayed timber about $\frac{1}{6}$ mm. long, also a number of flat cells and two pieces of quartz.

Organic Dust from Ferreira G.M. × 800.

From Air of Ferreira G.M. × 500.

The President introduced Sir William H. Preece, K.C.B., F.R.S., who, he pointed out, should be very well known to them all as one of the pioneers of electricity, particularly in wireless telegraphy, and who had an announcement to make as to a scheme for the advancement of mining and metallurgy, namely, the Bessemer Laboratory at South Kensington.

Sir William Preece, who met with a hearty reception, said: Gentlemen, it gives me great pleasure to renew my acquaintance with Johannesburg. I was here six years ago with the British Association, but I had not then acquainted myself sufficiently with the peculiarities of the atmosphere in the Transvaal. I was rather more interested in the new scenery, so much so that I was from 5 o'clock one cold morning until late in the day examining the beauty of the scenery from the platform of the train with the result that I caught a severe chill, but I managed to struggle one day into one of the sections and read a paper on "Wireless Telegraphy." I was subsequently taken all round about and I saw distinctly the nucleus of something that was going to be very big and very satisfactory. I come here now six years afterwards and I find that in every respect my anticipations have been realised. The place has grown immensely; the population is greater. Institutions for the promotion of the practical application of science are growing fast, and the result is, I find myself now in what I firmly believe to be the greatest centre of electrical enterprise and electrical energy in the British Empire. We have nothing so astounding to show the Britisher at Home as what you have to show him here. It passes all human understanding to see the rate at which things have grown. I am, however, bound to say that during the past six years we have not been quite idle at Home. One of the movements in which I have taken a very active part has been the work of the Standardisation Committee. I have been a member of the Electrical Standards Committee since its formation, about thirty years ago, and we have done well. Our electrical units are in universal use. We have established standards that are being followed everywhere, but we are not satisfied with the first and great point we started with. It was not so much to establish a system of measurements, but to start with firm decisive definitions which shall clearly express to everyone the properties of materials, the nature of operations, and the result of actions. In the same way, strangers like myself who come here would wish very much that we had a dictionary of the terms which have grown, flourished and stayed here, but are not very well known in distant places like poor old London. I would suggest that you should be asked to give us clear and well thought-out definitions of such things as slime, sand, concentrate, tailing, etc., things common to everyone of you, but things we know precious little about until we get here. Now that leads me to this, that I also notice the great strides that have been made in the advance of education, and it is in technical education, more than any other that we want to see progress made. We are doing very well in London. To-night the principal object of my remarks is to lead you up to the fact that we are establishing in London what I think will prove to be the most perfect laboratory for mining and metallurgy that is to be found anywhere. It arose in this way. A good many years ago it was decided to erect a memorial to Henry Bessemer, and a committee was formed to organise the collection of funds for the erection of this memorial. We did very well. Our late King Edward VII. laid the foundation stone of the building. We have every reason to hope that the laboratory will be opened by King George V., not very long after the Coronation. The building itself is complete. It is, as far as we are aware, the largest laboratory of the kind in any part of the world. It will be equipped with every possible form of apparatus that will contribute to the complete education of the mining and metallurgical engineer. We are promised help in all directions. We want the help and assistance of the British manufacturers. They are rather difficult gentlemen to move. In America if they want anything for a laboratory the American engineers vie with each other in supplying apparatus and never think of making a charge. In England, the first thing if there is some money in hand, and it is known that certain apparatus is required, is to be waited upon by some agent who wants to get a good order. That is not what we want. We want them to come forward and simply say "I will let you have this," and then we shall get our apparatus right up-to-date and ready for use without having to pay agents' commissions and manufacturers' profits.

Now there is an idea that we are rather behind in England in technical education. I do not think that is just. I do not know that there is any technical institute in this world that has been so successful in sending accomplished students out into the world as our School of Mines. Again, if you take Naval Architecture I do not believe there is any country that can show students of Naval Architecture who have acquired such positions and such great eminence. I do not think I should be very far wrong in saying that we are not very much behind in electrical engineering. I ought not perhaps to do that for I can only do it by making a boast, as I was the first to establish a school of electrical engineering. That was in Southampton in the early sixties. I turned out two or three good students. One of the earliest was Prof. Ayrton, who lectured to you when the British Association was here. Another one, who was by far the most able man I ever had with me, and who (very foolishly I think) dropped engineering, was Sir James Sivewright I only want to mention these facts to point out to you how interested I am in the advancement of education. I cannot suggest anything to improve the conditions of education in this country, especially when we are here in a building erected for the very purpose. We have nearly completed in London the Imperial College of Science and Technology, of which we shall be proud. I am not going to ask for anything, but I say this, that if I am spared I shall come here again one or two years hence, and I am sure I shall find in this very live place a constant steady improvement. I have visited America four times, and spent three months there on each occasion, and I have come back from America with the feeling that there is something in the electrical contact between Briton and American to the great advantage of the Briton, for he acquires by this contact a species of electrical energy. He is a better fellow to work with, and knows his work better. I believe the reason to be—not that there is any racial difference between American and Briton—that the former lives in an atmosphere totally different to that of London. There is an atmosphere—a champagne-like atmosphere in the States which makes one lively immediately one gets there. You never tire there. You work all day there. I come to Johannesburg and find that you are all imbued with the same kind of champagne effect. In your low atmospheric pressures, in your climate, in your surroundings, and in the busy world in which you live, there is not much difference between the life of the Johannesburger and the very active New Yorker. There were several young Britons who came out with me on the steamer. It is good for them to come to such a place as this, for we are kept down at Home by our atmosphere and climate. We have not the gorgeous sun you have here, we have not the air you breathe, and we are utterly deficient in that energy such as managers here possess, supported by the enormous capital you have at your command. When we in England want capital for any purpose we cannot get it. There is something in the company promoter which deters the public from subscribing. As an active exponent of new things electrical I have almost cried over the way our proposals are met by the financial members of the City of London. I will give you one case. I had a little apparatus that might be useful for anyone, for household purposes, for office purposes, and for many other purposes, even Government purposes. I only wanted £10,000 to start this thing, and I went to one of the leading men on the Stock Exchange, a personal friend. He was a very wealthy man and one who was connected with a great many successful undertakings. He said, "Well, what do you want?" I said "Only £10,000." "Pooh!" he said, "If you want £200,000 or £300,000 I could do something for you; but £10,000, we do not look at that on the Stock Exchange!" The result was I did not get my money from him; I had to get it myself! Gentlemen, I have had the great pleasure of coming here to-night and, whatever I have said has been with the firm desire that you will go on as you are going, and that in the immediate future I may come here again and find as great an advance in Johannesburg as in the last six years.

Mr. E. J. Laschinger (*Member of Council*): We very much appreciate Sir William Preece's remarks to us. He has taken the trouble to come here to see the Society at work, and to give us encouragement. I think it a great honour when a man of Sir William's standing comes here to say what he has said. He has told us about the marvellous developments which have been made in Johannesburg since he was here last, and expressed the hope that we will continue to expand and progress. Further, with regard to the mining and metallurgical laboratory which is being opened in London, we, as a Society, ought to take a lively interest in it as much as possible, and keep in touch with what is going on in the great financial centre of the world. It is to some extent perhaps a fallacious idea that the people in England are behind the times. It might have been true some time ago, but they have taken the challenge up, and we all know that a great deal of the best work in the mining and metallurgical world emanates from London to-day. This is because they have obtained the co-operation and assistance of various technical societies and workers all over the world. They have recognised the principle that science is universal, and that the best results are achieved by all working together, concentrating their ideas, and putting the best forward as a standard for those who come after to start from. I think I am voicing the opinion of members when I say that we owe Sir William a hearty vote of thanks for attending our meeting this evening.

Sir William Preece: I need hardly say I am very thankful to you all for the kind reception you have given my remarks. I wish they had been a little more carefully considered, but at the same time I think I have spoken rather more from the heart than the head. I have taken every possible opportunity to find out what you are doing here, and during the time I remain, I shall probably worry a few more of you to give me a little further information.

The President (*Contributed*): I have just received from Mr. A. C. Claudet an account of the Imperial College of Science and Technology, from which the following extract will be interesting:

The Mining Department.—"The main entrance to the mining section will be in Prince Consort Road, and the buildings in their entirety occupy a space of about 1½ acres, with a frontage to Prince Consort Road and to Exhibition Road. The buildings will accommodate the mining, metallurgical, geological and engineering departments—the three first named being to the west of the building and the engineering department to the east. The mining department includes on the ground floor, a large court, lighted from the roof, about 246 ft. long × 125 ft. wide.

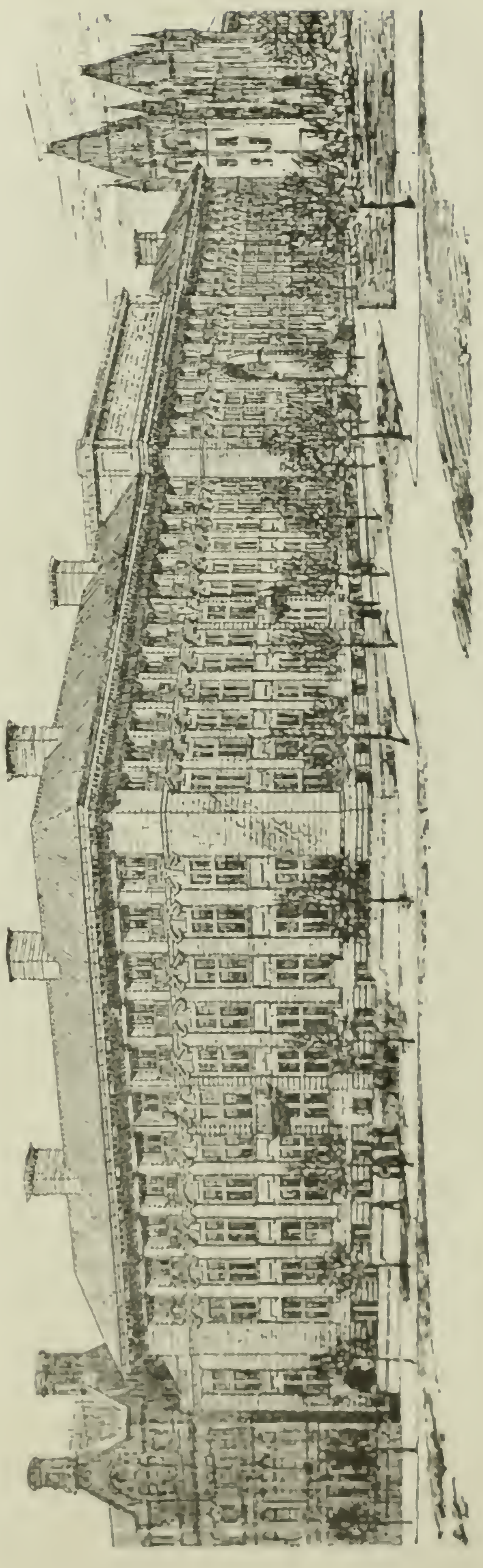
Another court adjoining will contain the roasting and smelting furnaces, in connection with which are the laboratories for wet and dry assay, with balance-room, lecture-room, laboratories for micrography, pyrometry, electro-metallurgy, the advanced course of metallurgical operations, and the equable temperature room. In connection with the mining department are two large lecture-rooms, museum and drawing office, machinery, assay, laboratory, and departmental library.

The geological department on the second floor contains laboratories for palæontology, petrology, mineralogy, economics and rock cutting, a museum 86 ft. × 40 ft., elementary laboratories, large drawing office, and research rooms.

All the buildings will be of the latest construction, fire resisting, amply lighted in both rooms and corridors, and laid out on simple and direct lines, with easy staircases and lifts at frequent points and good cartway approaches at the rear to all departments. There will also be large store rooms and separate cloak and lavatory accommodation for staff and servants. The building will be lighted throughout with electricity and heated by hot water and by steam, both electric current and steam being probably supplied from the central station

There is a plot which lies to the north of Prince Consort Road, and here the building to be erected will contain on the ground floor a lounge hall, large dining-room and kitchens, smoking-room, ladies', guests' and committee rooms. On the upper floor will be provided a large concert and debating room, to be used also as gymnasium library and reading-room, and rooms for staff, with a separate entrance.

The Bessemer Laboratory.—The Bessemer Laboratory is now being equipped with machinery.



Sketch of Proposed New Buildings at the corner of Exhibition Road and Prince Consort Road, South Kensington.

It is, as already stated, a top-lighted building about 240 ft. × 125 ft., with large upper ore sampling floor, an upper concentration floor, and a main concreted lower floor. All the usual type of wet and dry crushing machines will be installed with amalgamation, concentration, cyaniding and other processes, arranged for practical working scale tests of ores. In the metallurgical department there will be various types of furnaces to illustrate the teaching courses; the chief object of the whole arrangement being for the purpose of illustrating in a practical manner the teaching work of the professors.

The installation, though far from complete, already comprises a five-stamp battery, two-stamp battery, two crushers, a Huntington mill, two ball mills and elevators, all of full size such as would be found in operation at an ordinary mine. In fact the laboratory is on a scale commensurate with the working of a good-sized mine. It is quite correct to say that no such laboratory exists in any other part of the world.

No doubt seems to be felt that the mining building will be completed by the end of the year, from which time the training of the future mining engineer and metallurgist will be on an infinitely broader and more satisfactory basis than it has ever previously been.

How the Movement Originated.—The movement was started by the Institution of Mining and Metallurgy in 1901, and has been well described by Mr. McDermid, its secretary, as "for the co-ordination and re-organisation of higher technological education at South Kensington." The Government of the day has been most sympathetic, as no one can doubt who reads the speech of Mr. Haldane at the annual dinner of the Institution some eighteen months ago.

What the College will do for the Student.—Having thus described the Imperial College of Science and Technology in its relationship to the industries of mining and metallurgy, it is well to be precise in stating what it is calculated to do for the student in both these branches of his profession.

Mr. Claudet says: Let anyone go to South Kensington and examine what is being done there, and, if previously unacquainted with the subject, he will be astonished at what he sees. Practical metallurgical work is there to be conducted on a large scale in the new Bessemer laboratory. Much of the equipment which is of an elaborate kind is being provided out of the Bessemer Memorial Fund, and a large amount has already been spent.

The contention running through a recent article in the *Mining World*, entitled "How to Train Miners," was that the instruction given in the School of Mines (temporarily located in the old premises at South Kensington) is theoretical rather than practical, but this is not so, as I shall have little difficulty in proving. Students can go through a four-year course and obtain the advantage of a fifth year as Post-Graduates at mines in this country and abroad, under conditions I will presently explain. Hitherto, the Royal School of Mines has not been so advantageously placed for giving practical underground instruction as the schools in Cornwall, but it is obligatory under the present curriculum that a certain amount of practical work underground shall be done by the student during the long vacations. After the ordinary theoretical course, *plus* the underground experience gained during vacation time, any student who has obtained his associateship of the Royal School of Mines is eligible for selection for a Post-graduate course in Mines or Works, at home or abroad, inaugurated by the Institution of Mining and Metallurgy some eight years ago. Twenty or more graduates of the Royal School of Mines and other recognised colleges are selected annually by the Council of the Institution for these Post-Graduate courses, and those selected are subjected to thorough medical examination before their selection is confirmed. During the course (which is for one year, except in India, where it is for two years) the graduates are rotated through all the departments of the mines or works to which they are sent, and they receive from the mine owners remuneration which is more than sufficient to cover the cost of living. In addition to this, the Institution awards annually five scholarships of £50 each, and makes grants in certain cases from a special fund in its possession. The Council in their annual reports have frequently expressed their gratification at the results of these courses, which are far-reaching in their effect. It may be useful in this connection to quote what Sir Julius Wernher said about the Post-Graduate courses so recently as December last: 'I know of a good many Post-Graduate students who have been sent to South Africa and can say I have never heard a complaint about them. I believe they have always done their duty and will be a credit to your Institution in the future."

PROFIT PER FATHOM.

By R. E. Sawyer, A.R.S.M., A.I.M.M (Associate).

These are a few notes in favour of a partial adoption of Mr. H. F. Marriott's fathomage system,* bringing forward certain favourable points, which I do not consider he made the most of.

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* *S.A. Mining Journal*, March 12, 1910, p. 29; and March 26, 1910, p. 105.

With widely varying conditions, such as stope widths, nature of hanging, etc., it is impossible to say exactly what is the total cost (mining and reduction) of rock broken in any particular stope. The best we can do is to obtain a working approximation. The actual cost of stores and labour for any one stope is known accurately, but this charge amounts to less than one-fourth ($\frac{1}{4}$) of the total cost, in the case of large machine stopes.

The other charges shown in Table I. cannot be directly apportioned to each stope, but are distributed over hand, machine, small machine stopes, and the average for each class determined. All charges are distributed on a fathomage basis, with the exception of tramming, hoisting, reduction and residue. These latter are on the tonnage basis.

Sorting.—The value of sorted rock is often taken as one pennyweight. In a series of prolonged experiments I obtained a waste value of 0·4 dwt., when sorting from 10% to 20%, with very small variations in individual assays. It would seem, therefore, at first sight that a deduction should be made from the value per fathom to allow for the amount sorted. In the stope valuation, however, all waste rock is taken as being of no value, and all assays less than 1

TABLE I.

				SW"		SW"		SW"
Fathoms (measured) ...		2,568	1,190	55	1,140	122	238	40
Fathoms adjusted ...		2,758	1,310	—	1,200	—	248	—
Tons		59,476	18,000	—	39,000	—	2,476	—
Allotted.	Method Distribution.	Total Amount £	Hammer £	Shillings per Fathom.	Big Machines £	Shillings per Fathom.	Small Machines £	Shillings per Fathom.
Compressor Charges ...	F	1034·7	—	–	852·0	14·2	182·7	14·7
Rock Drill Maintenance	F	727·5	—	—	595·0	9·9	132·5	10·7
Machine Drill M & Sharp	F	593·6	—	—	485·0	8·1	108·6	8·7
Air Pipe Maintenance	F	95·4	—	—	77·5	1·3	17·0	1·4
Rigging	F	559·0	—	—	559·0	9·3	—	—
Hand Drill M Sharp ...	F	740·5	740·5	11·3	—	—	—	—
Central Pump ...	F	521·2	247·0	3·8	227·0	3·8	46·8	3·8
Tramming	T	3049·6	924·6	14·1	2000·0	33·4	125·0	10·0
Hoisting	T	1393·4	422·0	6·4	915·0	15·2	56·4	4·6
Timbering	F	664·6	316·0	4·8	289·0	4·8	59·6	4·8
Mine Administration...	F	783·1	372·0	5·7	340·0	5·7	71·1	5·7
General Charges ...	F	461·0	219·0	3·3	201·0	3·3	41·0	3·3
Sub Accounts	F	187·2	89·0	1·4	81·5	1·4	16·7	1·4
Underground Extras		10809·9	3330·1	50·3	6622·0	110·4	857·4	69·1
Stores and Labour ...	F	—	—	110·4	—	65·6	—	72·7
Total Underground ...	T	—	—	161·2	—	176·0	—	141·8
Reduction	T	9300·0	2820·0	49·1	6100·0	101·5	380·0	30·6
General Charges ...	F	1540·0	731·0	11·0	670·0	11·2	139·0	11·2
Residue	T	4360·0	1320·0	20·0	2860·0	47·6	180·0	14·5
		—		74·4	—	160·3	—	56·3
Total Deduction ...		—	—	235·3	—	336·3	—	198·1
Less Stores and Labour		—	—	110·0	—	65·6	—	72·7
Additions for Tables 2, 3 and 4.		—	—	125·3	—	270·7	—	125·4
Total excluding Residue		—	—	215·5	—	288·7	—	183·6
Cost per Ton ...		—	—	15·6	—	9·5	—	18·4
					Total cost per fathom mined			244·0
					Total cost per ton mined			11·7

dwt. are returned as a trace and calculated as of no value. No doubt some of the 0·4 dwt. of waste rock assay is due to mud derived from richer rock. On the other hand some waste rock, such as portions of "black bar" and poor looking bastard, assays considerably more than 1 dwt. It is, therefore, best to take the value of the waste as it is reckoned in the valuation, *i.e.*, of no value. So the value per fathom of any stope is unaltered by sorting. Sorting cannot raise the number of dwt. in a fathom, and its only effect is to reduce the total reduction charge. This charge, in a perfect method, should be distributed over each stope, according to the proportion which it is estimated could be sorted from the stope. In practice this is impossible, as anyone who has tried it will admit. The sorting charge has, therefore, to be put in with the other general charges. When, however, a stope is on the border line of unpayability, it should be examined to see if the average percentage sorted applies to it or not. If it is capable of a very large amount of sorting, special allowance must be made, which will increase its payability.

Fathomage.—A practical difficulty, that will arise on most mines, is that the tonnage as reckoned from the fathomage by the surveyors will be considerably short of that returned by the reduction department. In my experience, even this latter return is often materially short of the truth. Every effort should be made to secure agreement in the tonnage; stope widths must be checked, and allowance made for stripping of foot and hanging wall. The tramming returns

TABLE II.—HAMMER STOPES.

+125·3.

	Inch.	Dwt.	Fathoms.	Stores and Labour. £	Shillings. Stores and Labour cost per fathom.	Shillings. Value per fathom.	Shillings Profit per fathom.	Shillings. Loss.
10 W 1 E	45	7·7	59·7	351·7	117·5	364	121	
10 W 1 W	57·3	4·4	36·6	328·5	145·0	349	79	
10 W 2 E	65·4	10·4	8·4					
10 W 3 E	52·7	2 3	6·5	176·3	175	284	16	
10 W 3A E	55·5	6·0	10 5					
10 W 3 W	47·7	7·9	2·9					
10 W 4 E	79·8	4·9	22·5	293·8	180	388	83	
10 W 4 W	51·1	2·5	10·0					
10 W 5 E	38·1	10·1	86·1	396 0	92	405	188	
10 W 5 W	36·9	8·3	65·7	375·8	114	322	83	
11 W 1 E	44·0	5·0	44·3	205·0	93	231	13	
11 W 1 W	50 4	10·5	41·0	296 6	142	556	289	
11 W 2 E	47·3	2·3	14·2	171·3	98	376	153	
11 W 2 W	47·3	11·4	20·6					
12 W 1 E	40·4	7·0	32·5	348·0	95	416	196	
12 W 1 W	59·0	8·7	40·6					
12 W 2 E	41·2	5·1	45·5	298·9	131	221		35
12 W 2 W	51·1	6·9	51·7	313·5	121	371	125	
12 W 3 E	48·9	11·0	65·0	340·2	105	566	336	
12 W 3 W	50·1	6·9	39·2	360·9	90	412	197	
12 W 4 E	57·7	7·8	40·9					
12 W 4 W	39·1	17·7	28 9	136·5	94	726	507	
13 W 1 E	40·4	4·0	48·6	177·1	70	168		27
13 W 1 W	74·2	7·3	66 5	329·3	99	569	345	
13 W 2 E	65·8	8·1	59·4	318·9	107	560	328	
13 W 5 W	43·6	3·2	31·4	389 4	111	289	53	
14 W 1 E	43·2	10·4	11·4					
14 W 2 W	57·3	1·8	21·2					
14 W 2 E	56·5	3·6	5·8					

Average cost, stores and labour per fathom = 110·4 shillings.
Average stope width ... ... = 55 inches.

TABLE III.—MACHINE STOPES (LARGE MACHINES).

+270·7.

	Inch.	Dwt.	Fathoms.	Stores and Labour. £	Shillings. Stores and Labour cost per fathom.	Shillings Value per fathom.	Shillings. Profit per fathom.	Shillings. Loss.
1 W 2 E	143	2·4	31·2	102·7	65·6	360	24	
1 W 3 E	130	4·8	26·5	91·6	71·8	655	313	
1 W 3 W	133	4·4	20·7	96·2	92·8	612	249	
2 W 2 E	107	2·3	38·1	85·0	44·5	259		56
2 W 1 E	120	2·4	36·9	102·5	55·6	303		23
2 W 3 W	133	3·5	35·1	108·7	61·8	490	158	
2 W 3 E	162	1·8	26·2	98·5	75·1	306		40
4 W 2 E	55	3·3	36·5	109·5	60·1	189		141
5 W 3 E	116	4·2	25·2	91·4	72·5	512	169	
6 W 1 E	137	3·3	22·7	96·0	42·3	475	162	
6 W 1 W	132	3·6	20·6	49·5	48·0	499	181	
7 W 1 E	118	2·9	64·7	264·2	81·6	359	7	
7 W 1 W	120	3·9	26·7	74-5	55·7	492	166	
8 W 1 E	105	4·8	30·6	96·5	63·0	530	197	
8 W 1 W	88	2·4	37·0	106·6	57·5	222		106
8 W 2 E	85	2·4	55·0	138·1	50·1	214		107
8 W 2 W	112	2·6	31·5	109·5	69·6	306		34
8 W 3 E	100	2·6	35·4	90·5	51·2	273		49
9 W 1 E	120	2·6	36·5	128·8	70·4	328		13
9 W 1 W	143	2·6	19·1	88·3	92·4	390	27	
9 W 2 E	130	2·3	27·0	104·8	77·6	314		34

Average cost, stores and labour per fathom = 65·6 shillings.

Average stope width ... ... = 122 inches.

TABLE IV.—MACHINE STOPES (SMALL MACHINES).

+125·4.

	Inch.	Dwt.	Fathoms.	Stores and Labour. £	Shillings Stores and Labour cost per fathom.	Shillings Value per fathom.	Shillings. Profit per fathom.	Shillings. Loss.
1 E 1 E	43·7	11·0	26·0	145·5	112·0	505	268	
1 E 1 W	44·0	8·0	36·3	139·0	76·6	370	168	
1 E 2 E	52·4	5·4	19·9	79·2	79·6	297	92	
1 E 2 W	40·6	18·0	21·0	87·4	83·0	768	560	
1 E 3 E	39·3	8·0	25·6	83·6	65·2	330	140	
1 E 3 W	32·1	20·6	27·4	97·8	71·2	661	465	
1 E 4 E	26·2	12·7	27·5	75·1	54·8	333	153	
1 E 4 W	40·4	3·5	17·3	23·5	27·2	148		4

Average cost, stores and labour per fathom = 72·7 shillings.

Average stope width ... ... = 40 inches.

are only useful to determine the proportion of rock trammed from the three classes of stopes The total tonnage from stopes, after making all known deductions, is then distributed over hand, machine and small machine stopes in the proportion of the trammer's tally. The distributed tonnage is then converted into fathoms by using the known average stope widths for the various classes of stopes (hand, machine, small machine). The new fathomage is known as "adjusted fathoms," and is used for purposes of distribution. Having thus outlined the method, I advance the following claims for it :—

1. That the essential factor, gold contents per fathom, is independent of sorting. This factor in shillings is given by the formula : Reef width (in.) × assay value (dwt.) × 1·05. It will be noticed that stope width does not enter into this factor.

2. That the final figure, "profit per fathom," conveys more to the mine manager than any other figure. He has a very good idea of what fathomage to expect from any stope in a month ; he thus knows which stopes to push when the profit falls.

3. That it will help the manager to escape that terrible bogey "grade." The cry for grade is as pernicious as the shout for tonnage, it is only the happy combination of the two that spells profit ; and this is what the manager wants, or should want.

From the tables it will be seen that while it pays to work certain stopes at 2·6 dwt., other stopes are run at a loss although assaying 5 dwt. This shows how misleading the sampler's value may be, if considered by itself.

Sampling.—It is not always advisable to stop a stope that shows a small loss per fathom, as the calculations depend so largely on the sampling. In any doubtful case the last three month's assays should be taken into account and the assay plans consulted to see if there is no chance of better ore ahead.

The sampling of a large machine stope is a very difficult matter, and I consider that a section should be made of each stope face, showing drives, cross-cuts, box-holes, above and below stope and giving values. (See Fig. I.)

These sections are of great value in showing what reef should be stoped or left. They also ensure the sampler taking an intelligent interest in his work, as a stope carelessly sampled will never plot well on a section.

Pay Reef.—On some mines the percentage of pay reef mined is determined for each stope. I have drawn up Fig. II. so that a sampler on being told the total underground cost per fathom for any class of stope (in this case 161·2 for hand, 176·0 machine, 141·8 small machine) can at once see what is the pay limit for any stope width. To this must be added the value of the residue unrecovered.

Reduction of Tonnage.—In the hypothetical mine considered, let us say 60,000 tons per month are being crushed with 200 stamps. Suppose that all stopes which give a loss or a doubtful profit are stopped. This will knock off about 13,000 tons per month, and it will be possible to close down 50 stamps. The saving as shown by the sheets would be about £1,150. But it will be obvious that although we are reducing our tonnage by one fourth, the costs will not be decreased in the same proportion. For example, one quarter of the reduction costs will be about £2,710 ; practically the saving will not amount to more than £1,460. This difference must be borne by the remaining tonnage and cancels all that was saved by stopping these stopes. It appears, therefore, that no great advantage would accrue by shutting down all stopes that

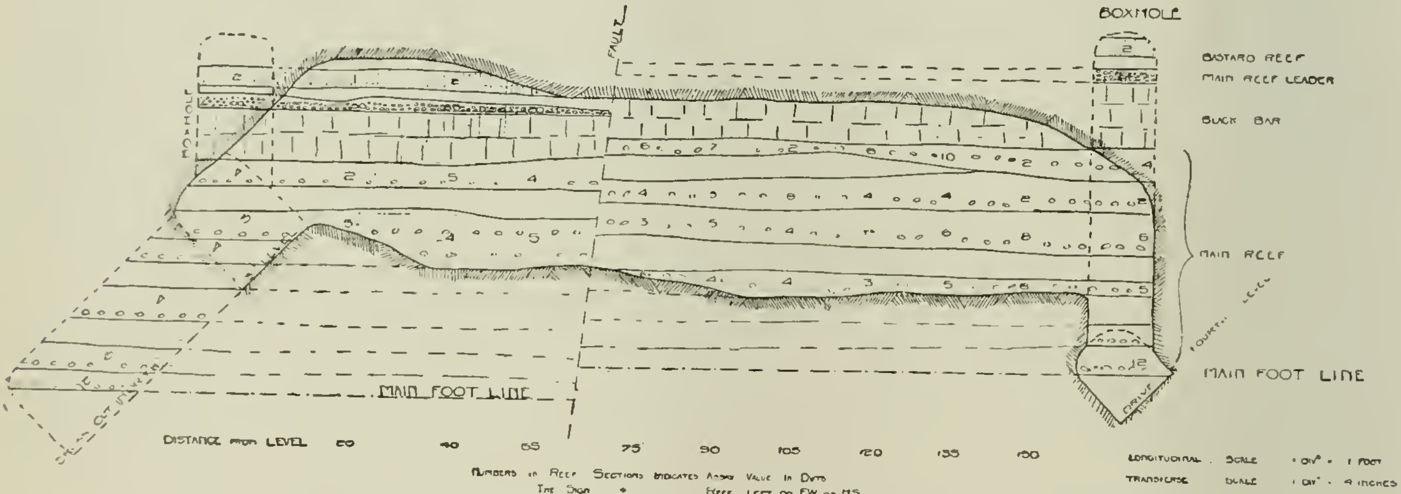
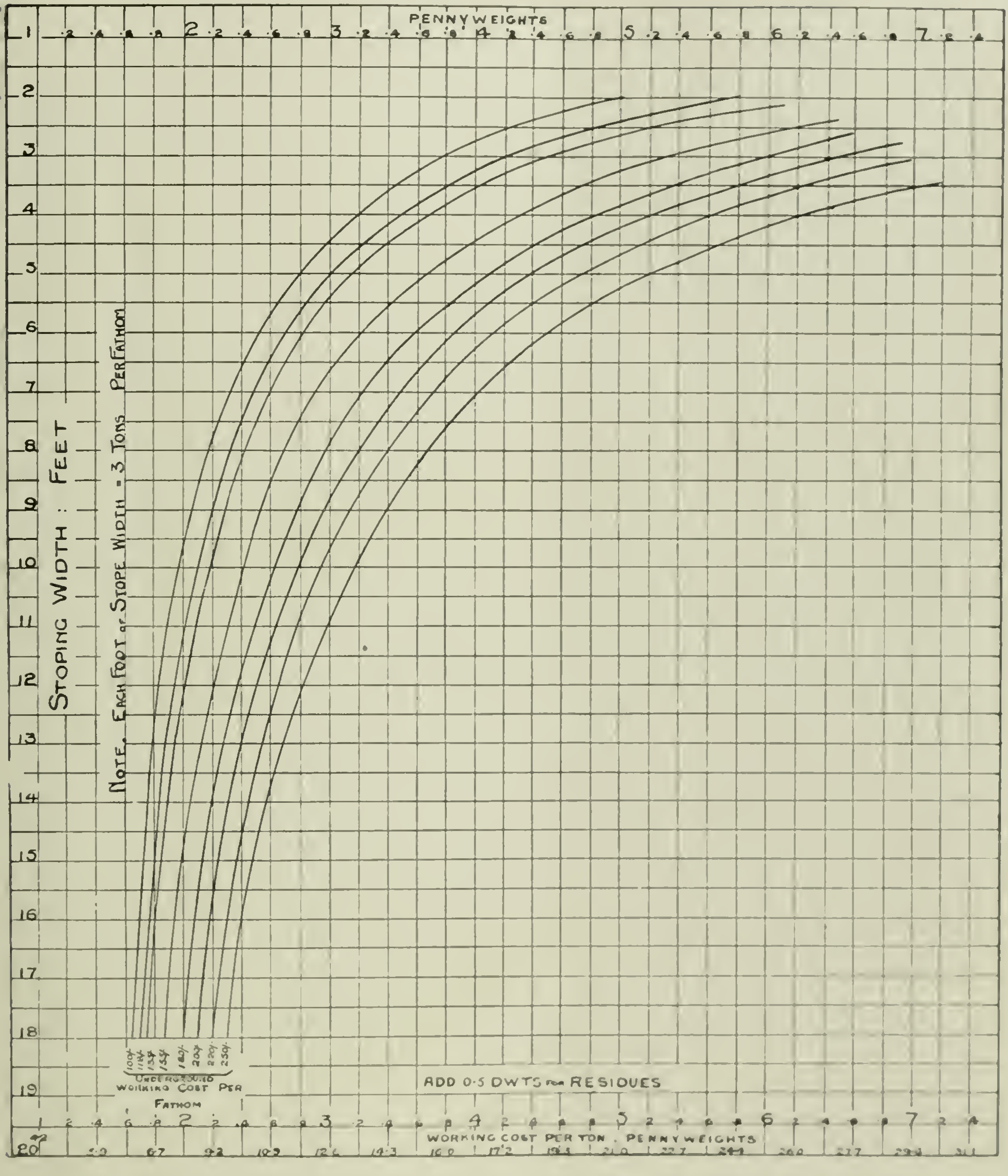

BOXHOLE
BASTARD REEF
MAIN REEF LEADER
BLACK BAR
MAIN REEF
MAIN FOOT LINE
FAULT
DRIVE
MAIN FOOT LINE
DISTANCE FROM LEVEL 20 40 65 75 90 105 120 135 150
LONGITUDINAL SCALE 1 DIV = 1 FOOT
TRANSVERSE SCALE 1 DIV = 4 INCHES
NUMBERS IN REEF SECTIONS INDICATES ASSAY VALUE IN DWTS
THE SIGN + REEF LEFT ON FW OR HS

FIG. I.—4W3E Stope : Numbers in Reef Sections indicate Assay Value in Dwt. The Sign + indicates Reef left on F.W. or H.S.



Curves showing Limit of Payability for Varying Stope Width and Cost per Fathom.

Reduction Cost per ton taken at 4s. 6d. per ton mined.

General Charges .. .. 6d. ,,

Sorting 10 per cent.

(Development Redemption excluded.)

FIG. 11.—Working Cost per Ton (Pennyweights).

show a loss, the correct course being to stop only those stopes which show a heavy loss, and put the machines or hand labour into better stopes.

On a large mine, that is divided into sections under the control of mine captains or shift bosses, it is advisable to call for a certain tonnage of a certain grade from each section, and the grade called for by the stope assays should be worked out daily for the whole mine. This should agree very closely with the screen assays. I would go further and say that while still working out the daily grade for the whole mine as a useful check, the daily profit from each section should be called for, and let the grade and tonnage take care of themselves.

A sheet would be prepared showing name of stope, stope width (in.) and a constant. These will not be varied during the month if pssible. The constant for any one stope is:

$$\frac{\text{Capacity of truck (tons)} \times 4 \times \text{profit per fathom}}{\text{Stope width (in.).}}$$

The daily entries will be number of trucks trammed, and the number of boys or machines working at that point. The daily profit for each stope would be calculated as follows:—

$$\text{Trucks} \times \text{constant} = \text{profit for the day.}$$

The profit for each section can then be obtained by addition. The calculation is simpler and takes less time than that of the average grade.

A careful study of the results in the tables will repay those interested. A noticeable point is the efficiency of the small machines.

The figures given are compiled merely as examples.

For the sake of argument a proportion of the large machine stopes have been shown as of low value, so as to come out unpayable.

NOTE.—All the formulæ quoted above are worked out on a basis of 12 cub. ft. of rock, in the solid, = 1 ton (2,000 lb.)

$$\text{A square fathom} = 36 \text{ sq. ft.}$$

$$\text{Hence } \frac{\text{Fathom} \times \text{SW (in.)}}{4} = \text{tons}$$

Also 1 dwt. has been taken = 4·2 shillings.

NOTE.—In Tables II. and III. the figures, over the column headed "Stores and labour, cost per fathom," represent the total addition to be made to this heading to give the total deductions.

The President: I have much pleasure in moving a hearty vote of thanks to Mr. Sawyer for his paper, and hope someone else, in whose province it is, will discuss it.

Mr. Chas. B Saner (*Vice-President*): With pleasure I will second the vote of thanks. The paper contains a lot of controversial matter, but as I have only just seen it, I will not criticise it now.

PRACTICAL NOTES ON COAL.

By MICHAEL DODD (Member).

In contemplating the uses of coal and the history of coal mining, I am driven very forcibly to the conclusion that this invaluable, and exhaustible, commodity has probably been handled in a fashion more extravagant and wasteful than has any other article of the world's great natural wealth. Coal has often been mined by methods which have given little heed to the future; the economy and the profit of the moment were the only considerations. Secondary seams have been sacrificed by the adopted mode of working the best one, so that in almost every coalfield there are left behind millions of tons of valuable fuel which it will never be possible to reach again

In consumption, too, there has often been a measure of ignorance and indifference such that an infinitely greater tonnage of fuel was constantly being used than was justified by the amount of work got out of it. Probably this was evinced more at the coal mines themselves than anywhere else. In my young days it was estimated that a colliery might be expected to consume in its own boilers about 10% of its total product. But in less than a generation an enormous change has come about: Scientific education, the wider study of all economic problems, the improved mental equipment of engineers and of business-men, the advance of discovery, the crying demand for the cheapening of the cost of power, have all contributed to bring about this change. The engineer of to day is never worth his place unless he knows a great deal more about the character, the source, and the cost of his power supplies than the colleges were in the habit of teaching five and twenty years ago.

That which prompted me to bring forward these few notes was one of the questions put to candidates in a recent Examination for Mine Manager's Certificates. The question was the eminently practical one: "If you were called upon to make a contract for supplies of coal, how would you set about it?" And this question shall be borne in mind in subsequent remarks. Leaving out such questions as the origin of coal, the distribution of it, and the mining of it, I propose to discuss—(1) its composition, (2) the different classifications, (3) the uses of the different sorts, (4) the question of selection

(1) *Composition.*—Without going into the question of ultimate analysis, as being outside the scope of a purely practical paper, the composition of coal may be put down as consisting of volatile hydrocarbons, fixed carbon, sulphur, ash, water.

The first three of these represent the combustible portion, sulphur being invariably present as pyrite. In small proportion, up to 1%, the presence of sulphur may be of little consequence, but, if this limit be exceeded, the sulphur is a source both of trouble and danger; trouble, because, given the presence of some impurities in the ash, sulphur induces clinker in the process of burning, and danger, because of the tendency to spontaneous combustion. Fires, both in the hold and in the bunkers, have all too frequently occurred in sea-going vessels.

Water.—All coals contain a small proportion of water both free and in chemical combination. As a rule, deeper seams contain less water than shallower ones; and generally, also, the older the geological formation from which the coal is produced the smaller the percentage. To both these rules, however, exceptions are common.

The effect of moisture in the coal is simply to cut down, by whatever proportion may be present, the total percentage of combustible—indeed, the effect is slightly more than this, because a little of the combustible is used up in evaporating the moisture.

The ash represents the total of the solid incombustible constituents of the coal. No coal is entirely free of ash, though some come very near to perfection in this respect, for analyses of British coals with so little as 2%, are by no means uncommon. Yet the most wonderful sample which has come to my notice was of South African production: it was a bit of freak coal from the Karoo, which contained only ·2% of ash.

With not more than 1% S in the coal it may safely be carried oversea: 2% is a distinctly dangerous figure; 1½% would give pause to a buyer.

Classification.—All classifications of coal are based on the same principle, *i.e.*, on the relative proportions of volatile gases and fixed carbon. No two authorities agree in their classification, but a simple method is to write the sorts down as:—

Highly bituminous,	40% and upwards	of volatiles
Bituminous,	18 to 40%	,,
Sub-bituminous,	12 to 18%	,,
Anthracite,	less than 12%	,,

The coal known as cannel, and some gas coals, are of the first named. Household coal, according to *taste*, may be chosen from any of the four. Coking coals may be either the first or second, whilst steam coal of the most useful quality belongs to the second and third classes.

The uses of Coal.—(1) For Gas-making.—The introduction of electricity as an illuminant has made a vast difference to the consumption of coal gas; still in Europe, and especially in Britain, there is an enormous production of gas, both for lighting and heating. The most suitable coal for the manufacture of gas is one which is high in volatile hydrocarbons, and in which the relative contents of the various gases are such that the gas produced will be high in illuminating power. A high-class cannel coal, with volatile contents of probably 45%, will yield 13,500 cubic feet of gas per long ton, of illuminating power of 36 candles; a good Durham gas coal, volatiles 35%, about 12,500 cubic feet, and illuminating power of 16 candles. Selected Witbank District coal, with 33% of volatiles, gives over 11,000 feet per ton, and an illuminating power of slightly over 14. The long ton is here meant, as it is the British standard.

For Coking.—In the selection of coal for the purpose of coke-making, it would appear reasonable to assume that with the scientific knowledge of to-day, an expert ought to be able to put his hand on analyses of coal and declare off-hand which were or were not suitable for coke-making; but this does not seem possible. Some of the best British cokes are made from coal whose volatile contents exceed 30%, while some of the best American are from coal whose volatiles fall short of 24%. There are unexplained mechanical or structural differences in coals of similar chemical composition such that, while one of these coals will produce an admirable coke, another will not do so, failing, it may be, to give either cohesion, hardness, or porosity, each of which qualities is an essential of good coke. In addition, a good coking coal must be low in ash and sulphur contents.

Steam Coal.—It is, in this country, to a very small extent only, that we are interested in the manufacture of gas or coke. Coal to us is important only as fuel used in the generation of power. This phase of the subject calls, therefore, for more detailed consideration. For use in steam boilers we seek, of course, to get full advantage of all the combustible contents of our coal, whether in the form of volatile hydrocarbons or fixed carbon. In the process of burning, the fixed carbon remains on the fire grate until it is consumed, so that, roughly speaking, all the value is extracted therefrom. The volatile gases on the other hand, are set free by the heat of the fire, and immediately start off for the smoke stack, and in every case a proportion escapes. The gases are elusive, and some power thus escapes. At first it would appear from this that the greater the proportion of fixed carbon in your combustible the more economical the coal. Given a forced draught and an excessive grate area this, in certain circumstances, holds good. But under ordinary conditions of firing, non-volatile or anthracitic coal burns so slowly as to render it impossible to get out of the boiler the amount of work which ought to be expected. A coal, therefore, for steam purposes should be one which contains a sufficient proportion of hydro carbons to give flame enough to burn the whole with such rapidity as to get the best work out of the boiler, and yet a proportion sufficiently low to prevent undue loss of combustible by the chimney. The vital question in determining this is: "What is the percentage of volatiles in the best and most economical steam coal?" The question is not an easy one to answer, because the figure must, naturally, vary with varying conditions. The stronger the flue draught the less assistance does the furnace need in the shape of gases. The higher the percentage of ash in the coal the greater the need for flame to hasten combustion The so-called smokeless coals of South Wales contain 10% or 11% of volatile, but these in consumption call for a special draught. Welsh coals are best for steam purposes, under ordinary conditions of firing, when they contain from 13% to 17% of volatile, but such coals will contain probably from 2% to 6% only of ash. Natal coals contain from 9% to 12% of ash, and probably the best steam producers amongst them are those which contain from 18% to 21% of volatile. Amongst Transvaal coals we find that the best Witbank coals contain from 13% to 16% of ash. These in consequence may be expected to give the best result when the volatile contents are from 22% to 24%. If we recognize these figures as being approximately reliable, a very natural enquiry is, how much we should write down the value of a coal for steam purposes because it contains a higher percentage of volatiles than the figure which is considered to be the most economical? This is hard to answer, since the reply depends so much on other questions, as for instance: Are the boilers hand fired? What is the space between the bars and the boiler shell? But with the average boiler plants in use on the Rand in mind, I should be disposed to hazard the opinion, that as good results may be expected from a Witbank District coal whose percentage of volatile is 24%, and whose calorific value is 12·6%, as from one whose volatile contents are 29%, while giving a calorific of 13%.

Anthracite.—The world's supply of anthracite is comparatively limited in comparison with the abundance of bituminous sorts, yet the occurrences are far from being unimportant. This class of coal is extensively used in the large towns of Europe and America because of its smokelessness. It is used for steam frequently enough when its cheapness, as compared with other coals, is such as to justify the provision of forced draught to aid combustion, whilst it is of great service to the suction gas engines which have come to the fore so rapidly during the past decade. At the present moment the makers of suction gas engines ask for a coal containing not more than 10% of volatile matter, though there is little doubt that in time they will succeed in adapting their generating plants to the use of bituminous coal. As yet it would appear that the tar generated in the fusion of this latter class of coal calls for too frequent stoppages for cleaning, and an excess of hydrocarbon gases in the explosive mixture increases the danger of pre-ignition

Calorific value —I must now refer to a point in reference especially to steam coal which, perhaps, should have received notice at an earlier stage in my notes, *i.e.*, the estimation of the relative heating power of coals. This is usually spoken of as "calorific value," and calorific value means the number of units of water a similar unit of coal will convert from boiling point into steam. Thus, when the calorific value of a coal is said to be 12, we understand that one pound of such coal will, in the laboratory under ideal conditions, convert 12lb. of water into steam, after it has first been heated to boiling point. The instrument by means of which such a test is made is termed a calorimeter. Of these there are several in use, of which the best is the "Mahler Bomb," the standard instrument in use on the Rand. Another form of expressing the calorific value of a coal is in terms of the "British thermal unit." The value of a coal in British thermal units is the number of units of water which one unit of coal will heat from 60% to 61% Fahr. The number of British thermal units is arrived at by multiplying the calorific value by 966—the figure which represents the latent heat of steam. The calorific value of a coal is, of course, its heating power under ideal conditions, *i.e.*, its theoretical power. That which is obtained from it in actual boiler practice is a different matter. This latter—the percentage of efficiency—depends on the character of the boiler plant, and on the amount of care displayed in the management of it. There are mine plants on the Rand to-day which do not probably give more than 45% efficiency, while others do as much as 60%. In South African railway practice, a figure of 55% of three or four years ago has improved, in the newer locomotives, to over 60%. In the more modern and best equipped stationary boilers 70% has certainly been left behind; and with the most approved methods of mechanical stoking it is claimed for some plants, that the high figure of 80% is now being attained

This brings us to the consideration of the very interesting question referred to earlier in this article, *i.e.*, the question which was set in the chemistry paper at a recent examination for mine managers' certificates: "If you were called upon to make a contract for coal, how would you set about it?" The following suggests itself as a practical sort of reply, viz.,

1. Call for tenders, to be accompanied by certificates of analysis and calorific value.
2. Rule out such tenders as are manifestly unsuitable, either from excessive price or known unsuitability of the coal offered.
3. Having reduced the choice to two or three, have these carefully sampled, and the samples analysed and tested for calorific value.
4. Decide on purchasing the coal which gives its heat unit at the lowest cost, provided:
    a. It does not clinker;
    b. It is neither too high nor too low in volatile contents;
    c. It is uniformly well separated and cleaned.

Naturally, the contract subsequently entered into must provide securities for the due delivery of coal of sample quality.

The United States Government buys a large tonnage of fuel. Some time ago the Minister responsible appointed a special committee of experts for the purpose of laying down on a scientific basis the conditions of purchase. A bulletin has been published explaining the conditions which have been adopted, and this contains some very interesting information. Both anthracitic and bituminous coals are purchased. In the case of anthracite, purchases are made on the basis of the percentage of ash contained in the dry coal, which means that all moisture is driven off before the test is made. A tender having been accepted, sampling is systematically carried out by a responsible official The samples taken are in quantities of not less than 100lbs., and these are carefully quartered down for analysis. Payments for the coal are made on the results of these analyses, credit being given the supplier if his coal shows a lower percentage of ash than that specified in his contract, while he is debited in the case of any excess Credits and debits are calculated on a definitely fixed scale. When bituminous coal is purchased, the coal is rated on the ash factor, plus calorific value, and in such cases it is specified that the percentage of volatile contents shall not exceed a certain fixed figure. Such conditions appear sound, and are certainly effective, but they strike one as very elaborate, involving a labour and expense which would only be justified in the case of very large purchases. It is an easy and *inexpensive* matter to ascertain calorific value, while an analysis is a much more cumbersome piece of work.

Too much stress cannot be laid on the importance of correct sampling of coal if reliable results are sought for. I have known coal sampled by one man to be tested to a calorific of under 11, while a sample of the same bulk, taken by other hands has shown 12·5. The rough and ready way of sampling is to pick up a chunk of coal, good or indifferent, according to the result desired, and send that to the laboratory. A test of a sample so taken is valueless. Sampling must be done by taking from the various parts of a heap a large number of small portions, and the aggregate of these should be carefully mixed, crushed, and quartered down if sound results are sought for.

As a dissertation on coal, the foregoing is very incomplete and scrappy. The object has been to give prominence only to some of the main practical factors which are worthy of note in the consumption of coal, and especially in consumption for the generation of steam.

The President: I am sure this is a most excellent type of paper, and one which, in agreement with the title, is intensely practical. I think also we ought to congratulate Mr. Dodd on his excellent delivery, for I fancy we very seldom get such a good speaker.

Mr. A. McArthur Johnston (*Past-President*): In seconding the proposal to move a hearty vote of thanks to the author, I feel that we ought to congratulate ourselves on having such an authority to speak on this subject. Mr. Dodd, as we all know, has been long connected with the collieries of South Africa, and more especially of the Transvaal, and is therefore most competent to help us in estimating the value thereof.

With your permission I would take this opportunity of criticising the author's paper, because he is not always with us and I am anxious to let him see that though we may criticise severely in this Society sometimes, we do so, as one of our Past-Presidents used to say, *suaviter in modo, fortiter in re*

Generally speaking, I am disappointed in the paper. From our knowledge of the author's attainments and of the store of figures which he must have by him pertaining to Transvaal coals, we were entitled to have expected from him information bearing on the local supplies. Such would have been beneficial not only to the gold industry, but to the coal mines and the country generally as advertising the class of coals which can be obtained here. His remarks are, too, very indefinite in many places, and seem to point to the intention of not giving too much away.

For example, he talks about the moisture in the coal as cutting down simply the total percentage of combustible—or slightly more than this, since the moisture has to be evaporated. Now, I am in agreement with this remark when referring to Middelburg coals, which as a rule give a moisture content of under 2%, but I decidedly object to coals containing 8% or even 10% of moisture,* such as are on the market here, being placed under this category. This quantity of moisture will have a material effect on the value of the coal.

Again, I am disappointed at the author not giving us some information on the coking capabilities of the Middelburg coals. I had hopes that he would have placed before us some data showing how by picking or by mining one particular seam, with subsequent washing, a high class coking coal could have been obtained, on which future smelting works in the Transvaal might rely. We know that in the past a little good coke has been produced in the Transvaal, but we are anxious to hear from an expert what can be done, and what possibly will be done, now that the huge coal industry is under such powerful and concentrated control

I note that the author gives us a definition of the British Thermal Unit. As a practical definition I have no quibble to find therewith, but as a scientific definition I am anxious to have the point made clear. Consulting most of the textbooks at hand I find quite a diversity of opinion. Carpenter[1], Cremer and Bicknell[2] and Jamieson[3] define the B.T.U. as the heat required to raise 1 lb. of water 1° F. at or near the temperature of maximum density 39·1°; Lunge[4], Schwackhofer and Browne[5] and Phillips[6] give as the temperature 32° F. to 33° F.; Roberts Austen[7], Lewes[8] and Poole[9] state 1° F. without mentioning the particular range; Sexton[10] affirms 60° F. to 61°, whilst Deschanel[11] states "there is not at present any very precise convention as to the temperature at which the cold water is to be taken. If we say that it is to be within a few degrees of the freezing point, the specification is sufficiently accurate for any thermal measurements yet made." Since such a diversity exists, it seems we had better refer this to the Standardization Committee. The point is interesting.

Another point worth noting is the pressure used in the Mahler bomb during combustion of the coal. In the report issued by the United States Geological Survey on the coal testing plant at St. Louis, 1904, it is stated in Part I., page 180, that a pressure of 18 atmospheres of oxygen was used in the bomb, except in some duplicate determinations, in which 25 atmospheres were used. As these gave practically identical results, the lower pressure was deemed sufficient. In December last Mr. E. A Allcut* contributed an article on the effect of varying pressures on the calorific power obtained. Calculating at 25 atmospheric pressure of oxygen as giving the maximum heat value, he tabulates descending pressure results as follows:—

Oxygen pressure (atmospheres)	25	20	15	13	11·3	9	7	5	3·2
Heat value (per cent. of maximum value.)	100	99·7	98·6	97·7	97	95	90	71·75	59·6

Personally speaking, my experience favours, when burning the coal in powder form, that of the U.S. Geological Survey, but when doing test or check work it is our invariable practice to ignite with a pressure of 25 atmospheres.

Mr. Dodd bases much of his paper on the question "If you were called upon to make a contract for coal, how would you set about it?" To my mind this is a question set by examiners to find out how deeply the answerer has thought of, or even read of, such a subject, and I view the author's answer as being insufficient. He evidently thinks so himself, as he proceeds to tell us about the buying of coal by the United States Government, and the conditions laid down by them. I personally feel that he would have been quite justified by including such remarks in his answer to the question, and it would have been an excellent ending to his paper, had he drafted for our perusal a specification, under the terms of which he would have been pleased to sell coal.

If I may, I would like to mention two points, not very clearly brought out by the author. He quotes that credit is given the supplier if his coal shows a lower percentage of ash than that specified in the contract, while he is debited in the case of any excess. This fails to recognise that steadiness in the quality supplied is very material to good firing, and I should certainly advocate that coal falling below a certain definite value, be returned to the mine at the supplier's risk and cost. Again, I am in complete agreement with the author in emphasizing the care required in sampling, and would only point out the absolute necessity of sampling immediately after weighing and putting the sample obtained in an airtight box or jar, since loss of moisture very quickly occurs in the boiler shed, with, of course, consequent loss to the buyer.

I trust that these remarks will be taken by the author as fair criticism, and that they will stimulate him to help us in threshing out this important subject.

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* *cf.* A. F. Crosse, this *Journal*, Vol. X., p. 311.

[1] Experimental Engineering, p. 311.

[2] Metallurgical Handbook, p 150.

[3] Steam and Steam-Engine, p. 32.

[4] Chemists' Handbook, p. 60.

[5] Fuel and Water, p. 12.

[6] Fuels, p. 25.

[7] Study of Metallurgy, p. 172.

[8] Liquid and Gaseous Fuels, p. 59.

[9] Calorific Power of Fuels, p. 3.

[10] Fuel, p. 35.

[11] Natural Philosophy, p. 311.

*Engineering, December 2nd, 1910.

Mr. E. M. Weston (*Member*): Mr. Dodd, in suggesting his answer to the question regarding selecting coals has, I think, overlooked one practical point which cannot be neglected, especially here on the Rand where some of the coals sold contain such a high percentage of ash. Should not one always hold a practical test of any coal with regard to the type of furnace and draught employed, and the boiler it must be burnt under? If one burns one of these high ash coals in an ordinary grate with natural draught under a water tube or tubular boiler, even if one burns nut-size coal only, one finds any laboratory tests of calorific power are useless and misleading. These tests may show a value of 6,000 or over in B.T.U.'s: but if one expects to get an evaporation of water anything like one might expect in accordance with that, one would be disappointed. Owing to the large percentage of ash an examination of the ash dump would probably show that only 60% to 70% of the carbonaceous matter had really been consumed, the unburnt carbon having gone to enrich and enlarge the ash dump. If, however, such a coal could be burnt in specially designed furnaces or mixed in proper proportion with a richer coal, better results might be obtained.

Mr. E. J. Laschinger (*Member of Council*): This subject of coal presents a very large field for investigation and discussion. There is one thing with regard to specifications in general. For whatever purpose material is required, I am a great believer in the fact that the only way to buy materials is to buy them on the specfication of their physical properties alone. The main disputes on contracts, especially coal, arise in this way. Coal is bought under certain guarantees of performance, and as soon as the guarantee is not fulfilled, the coal supplier is notified accordingly. He replies that the coal is not used properly, and so the dispute goes on, and there is nothing definite by which either party can come to a real settlement. The question as to what is a suitable specification of the physical and chemical properties of coal has been exercising the minds of most engineers who are responsible for the large power plants for the last 10 or 15 years, and more especially the last 5 years. A great deal has been published on the subject, and the only way to deal with it in regard to this discussion, is to get some of our members who know more or less about coal, to give us the salient points that should be included in the specification on coal.

Another point of scientific interest may arise from a discussion of the paper. The question is: does the Mahler bomb give a true idea of the heat value of coal? It is said that a great deal of heat which gas is supposed to give out in combustion is absolutely unaccounted for—15 to 20% of the power does not appear. We know that experiments have been made by well-known scientists recently, and it has been found out that, during the explosions of gas, there was energy transformed which escaped altogether without doing useful work of any kind, not even appearing in the form of heat.

Mr. M. Dodd (*Member*): I do not know whether it is desirable that there should be a reply to such points as have been raised immediately, or whether it is desirable they should form part of a subsequent discussion?

The President: As a rule a reply is given four months hence, but if there is anything urgent, you may as well give it to-night.

Mr. M. Dodd: There is nothing urgent, but there are points which I think properly belong to discussion later on.

Mr. F. J. Pooler (*Associate*): As a teacher, I should be glad if we could get this question of the British Thermal Unit fixed. In the metric system the unit is fixed as that quantity of heat that will raise 1gm. of water from 0°C. to 1°C., but in searching for a definition of the B.T.U., one finds different text books giving different temperatures and we have given up the attempt to fix this unit definitely, as a bad job, *e.g.* Clerk Maxwell avoids the difficulty by defining the unit as "that quantity of heat which, if applied to unit of mass (one pound) of water at some standard temperature (that of greatest density, 39°F. or 4°C., or occasionally some temperature more convenient for laboratory work, such as 62°F. or 15°C.) will raise the water 1° (F. or C.) in temperature." This being rather indefinite, I sought elsewhere and found the temperatures stated as

1 "at or near 39·1°F.,"
2 "from 39·1° to 40·1°F.,"
3 "from 39° to 40° F.,"
4 & *5* "from 32° to 33° F."

Definitions *1* & *4* were by the same author, and *4* & *5* the only two in agreement, differ from Clerk Maxwell.

It is, of course, not a matter of much practical importance, for Callendar & Barnes (B.A. report, 1899), state that the specific heat of water varies between 1·0054 at Fr. Pt. and 1·0074 at Bl. Pt., with a mean value of 1·0000 at 68°F. and a minimum value of ·9982 at 106°F., showing a maximum variation of less than 1% for the whole range, so that a calorific value returned as 10 might be either 9·91 or 10·09, not a very serious error either way, yet one that might make a fairly big difference in using large quantities.

In regard to the question of coal washing before coking is undertaken, a considerable amount of work has been done at Sweetwaters in Natal by the Maritzburg Iron Company, Ltd., and coke is now being turned out from there yielding 10% of ash and 1% of sulphur. The results were not so satisfactory till recently, but the ash per cent. has been lowered lately by washing the small coal, twice on a jigger-washer with a sieve provided with a felspar layer. Further analyses of coke from this process may yield interesting results, as our South African coals are generally thought to be poor for coking purposes. One would also like to know whether there is any more of that Karroo coal with 0·2% ash, for much that I have seen contained nearer 20% of ash.

Mr. W. Laurie Hamilton (*Member*): I would like to suggest that some of our engineers might give us a paper on the proper use of coal, and the better management of the boiler. That is the more important point I believe on the Rand to-day. I will give you two instances. There are two mines of 200 stamps each. One used better class coal than the other. Yet the cost per ton crushed in one case of the inferior coal was 10d. and in the other 3s. 6d. The general manager investigated and he found this. That in one case 50% of coal went on the ash dump and in the other 11%. In a case where there was only 18%, the ash was riddled and part re-used. The main cause was this. The one with the high consumption had better boilers than the other, but they were neglected. They had not a spare boiler and were driving those in use all the time so that they could never get the boilers cleaned out or repaired. In the other case they had a man who made a pet of his boilers, and I say that this has more to do with the consumption of coal than any other point on the Rand. Keep your boilers clean, use the coal suitable to them, and it will reduce your costs not only to 10d. but even lower.

Mr. Tom Johnson (*Member of Council*): I would like to bear testimony to the truth of Mr. Hamilton's remarks, though I believe at the same time the cost has gone up at the mine with the low costs through changing their coal in the same boilers.

Mr. M. Dodd (*Member*): The only point on which I think, at the present moment, it is necessary to remark on is the question of the British Thermal Unit. Of course 39·1° F. is the temperature of the greatest density of water, but to my mind there should be no difference between taking it at 39·1° F. and taking it at 60° for the reason that you do not need to take the density into account at all. You only take into account the weight of the water.

The President: Of course there is also the possibility that the specific heat of water changes when it is heated. I am pretty sure it does.

FAULTING PHENOMENA IN RAND MINES.

By G. H. Beatty (Member).

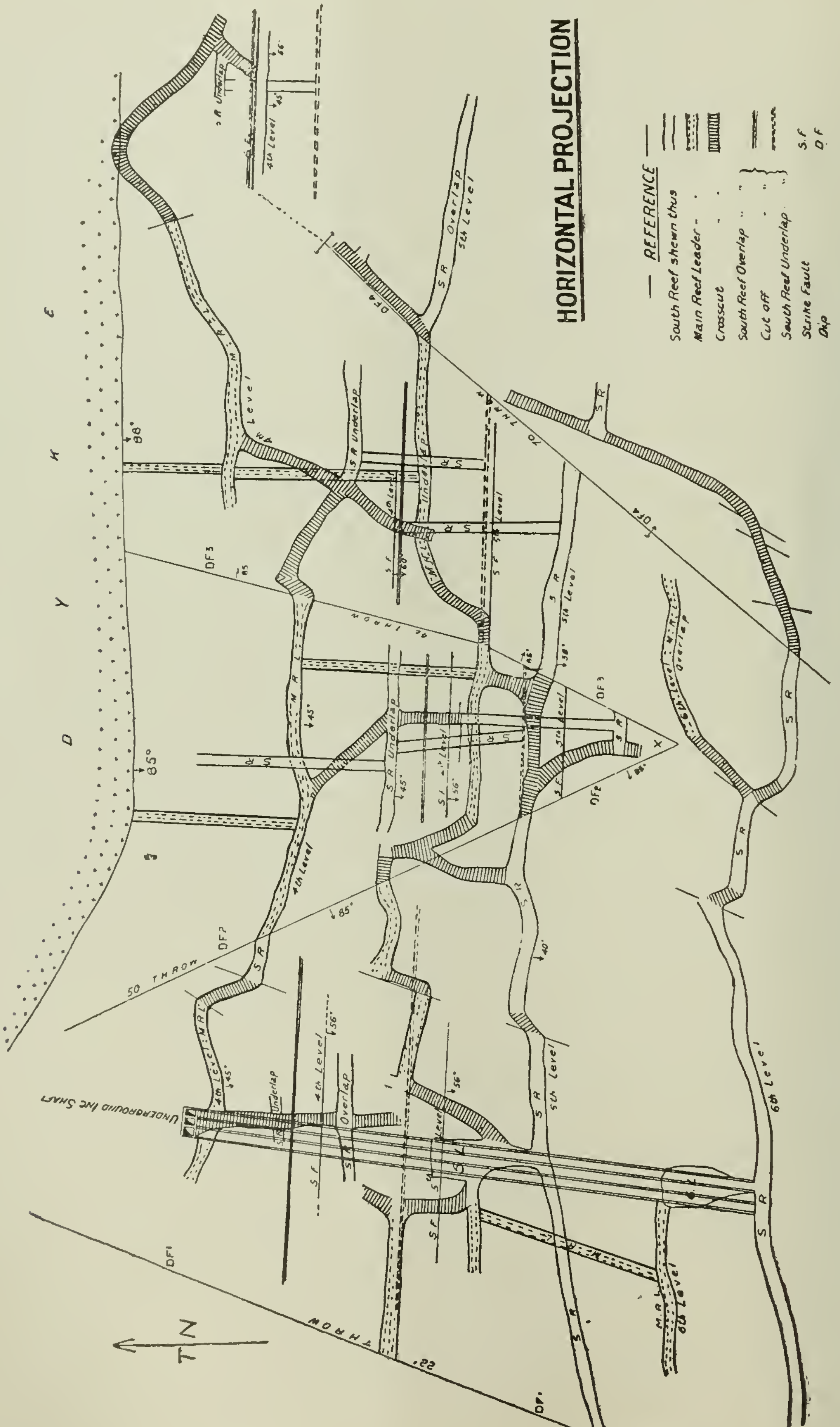
The subject matter of these notes is very well known, but in virtue of its important bearing on efficient development, I venture to bring it before the Society. I propose to open the discussion by an exposition of and some comment upon certain examples of complicated faulting which have come under my notice.

There is a lamentable looseness in the use by miners of expressions applied to faulting. For example, such general terms as "upthrow," "downthrow," "normal fault" and "reverse fault" are frequently used with little respect to their true application. To illustrate this, suppose in a level being driven east on a reef dipping south, that a dip fault is encountered dipping west; the average miner would speak of this as an upthrow of the reef, because, in driving through the fault he found the reef thrown south and upwards into the hanging wall. The fault is, however, a normal fault, *i.e.*, movement has taken place on the hanging wall (or western) side of the fault in a downward direction, and in effect, the reef on the western side of the fault along which a level has been advanced eastward, is properly regarded as the dislocated or downthrown portion, and the portion of the eastward side, which he calls upthrown, is really the reef in its normal position. A normal fault is a downthrow, and is accompanied by a loss of reef-bearing ground. The converse is true for a reverse fault, which is an upthrow and shows an overlap in horizontal projection. For all purposes it can be assumed that normal faults owe their existence to gravity, and reverse faults to pressure. Some evidence supports the belief that normal faults are not always due to subsidence, but the exception is rare and of no practical interest. In Rand formations, as a general rule, reverse faults are older than the normal ones. When the reverse fault so happens to be a strike fault, I have come across no important exception. It seems quite likely in many cases that the normal faults occurred when the lateral or tangential pressure which caused the reverse faults was relaxed. It follows that normal faults being younger displace the reverse faults. The appearance of a fault is frequently misleading as to the extent of its displacement. I know of one instance where the minimum movement along the fault plane was at least 1,000 ft., and yet the fault itself could not be distinguished. One meets with other faults which, though strongly defined, have little or no throw, but, of course, the actual movement may have been considerable. One meets the same difficulty in attempting to find any relation between the amount of throw and the relative dips of the fault and the reef, as these seem to bear no discoverable relation to the extent of the displacement of the reef by the fault. When it is necessary to locate a reef thrown by a single unknown fault, such indications as known strata, fault-rock, slickensides, folding of the strata on either or both sides, should be looked for and carefully examined. Failing any information from such sources, the dip and strike of the fault must be carefully taken, and the fault treated as one having a normal throw, the prospecting cross-cut being carefully examined after every blast for indications. When two or more faults have competed to dislocate the reef, the case becomes more interesting and calls for closer study. A valuable aid to the solution of problems arising from the disturbances due to faults and dikes, is the constant cultivation of geometrical perception. The full significance, however, of plans and sections is more clearly revealed with the aid of a model. Either one of two faults may cut out or throw the other, the former condition being rare. Almost every fault will be found to have an increasing or diminishing throw, ultimately dying out entirely in both directions. Two faults of the same system (reverse or normal) dipping in opposite directions may, upon approaching each other, coalesce, and terminating as individual faults, continue as one with a diminished throw. Again, if both dip in the same direction and coalesce, they will continue as one fault with an increased throw. Faults rarely unite, it need hardly be said, unless they have a similar trend, and probably belong to the same period. The direction of movement on a fault plane should, whenever possible, be determined. When one plane has cut and displaced two other planes of dissimilar dip, the direction of movement on the one can be determined graphically. When the average dip, strike, nature, and direction of movement of a fault have been determined, its effect upon faults, dykes, and reefs, with various dips in other parts of the mine can be ascertained with reasonable accuracy. Speaking generally, faults in Rand mines show a marked consistency in strike and dip. There are, of course, many interesting exceptions to this rule. Very often a big fault is accompanied by smaller sympathetic faults of similar trend and dip, causing troublesome stoping in the vicinity. An important fault is often possessed of peculiar characteristics, and its individuality admits of its easy recognition in various parts of the mine.

The principal feature of the section shown on the plan (Fig. I.) is a true strike fault having a reverse throw, causing an overlap or duplication of reef of 100 ft. on the west, and diminishing to the east, where it dies out in a monoclinal fold, some 500ft. east of the workings shown. This fault is readily distinguished on the western section where it carries an inch of fault-rock, but east of the shaft shown, its presence is only minutely revealed by a close examination: in fact, it is only noticeable where it comes in contact with a reef, and for development purposes its position was calculated. Considerable development was done in this neighbourhood in the nineties, at which time the faults shown were little understood, the existence of the strike-fault passing unrecognised. Recent development was, therefore laid out with a view to taking the best advantage of the old work. The plan further shows a series of dip faults numbered DF1, 2, 3 and 4, all of which have down throws, and were caused by movements subsequent to the lateral pressure which caused the strike-fault. Therefore, the strike-fault is displaced by all the dip faults. Upon examination, fault No. DF2 was found to have a down throw (vertical displacement) of 50 ft., and the direction of movement on it was clearly shown by joining, in cross-section, the apices formed by the reefs and strike fault. This direction proved to be nearly horizontal. In my experience this is the exception rather than the rule, for generally the direction of movement approaches the vertical.

The nature of fault No. DF2 being determined it was reasonable to assume that the characteristics of the other dip faults, which probably belonged to the same system, were similar. An examination of faults Nos. DF3 and 4 on the plan will show that this was nearly the case. As the movement on the dip faults was roughly horizontal the relative positions of strike fault and reefs remain unaltered on either side of the dip faults, on the same horizontal plane. Had the motion been vertical the strike fault would of course have, on the dropped section, approached the overlap and receded from the underlap portions of the reefs on the same level. Faults Nos. DF2 and 3 are especially interesting, they have down-throws of 50 ft. and 42 ft respectively, and dipping in contrary directions, unite at the point x, appearing in the lower level as one fault with a throw of 8 ft., this actually being the difference of their throws as individual faults.

The sketch will now, I think, explain itself. At the point 0 it will be noticed that the overlap portion of the south reef does not reach the 4th level. The available data showed that it could not be far away, and the strike fault not being visible at this point, a rise was put up from the 5th level, the last round in which blew through into the cross-cut. The cross-cut is old work,


HORIZONTAL PROJECTION
REFERENCE
South Reef shewn thus
Main Reef Leader
Crosscut
South Reef Overlap
Cut off
South Reef Underlap
Strike Fault S.F
Dip D.F
D Y K E
85°
88°
DF1
28' THROW
DF2
50 THROW
DF3
DF4
10' THROW
40' THROW
UNDERGROUND INC SHAFT
4th Level
5th Level
6th Level
S R
M R L
S R Underlap
S R Overlap
S F
45°
56°
40°
38°
X
N

probably driven to pick up the south reef—the underlap portion of the same reef having been mistaken for the main reef leader. The strike fault has, of course, exactly the same overlapping effect upon the main reef and main reef leader at a lower level. These reefs have been omitted from the plan, wherever possible, for the sake of simplicity. The dyke, which forms the north boundary of the section, is not displaced by the dip faults. Its dip is roughly vertical, and remembering that the direction of movement on the faults was horizontal, it follows that the dyke was subsequent to them. Had the movement been vertical, similar to the dip of the dyke, the dyke might have been faulted without showing any horizontal displacement. This horizontal movement further eliminates the possibility that the faults were subsequent to, and terminate on the dyke. The section of ground shown in Fig. I. is itself an overlap, probably caused by the dyke rather than by any later movement on it; firstly, because the dyke face shows no evidence of movement, and secondly, because such movement could only have been caused by a return of pressure after the dip faults had been developed. The thickness of the dyke is about 100 ft., and its underlap reef is also the overlap of another dyke to the north. In consequence of this the cross-cut from the vertical shaft to the incline shaft shown, exposes the reef three times. The two dykes mentioned are separate, and not one dyke faulted, as might be inferred. They have a similar but somewhat irregular strike. They meet in both directions, encircling an oval shaped piece of ground, 250 ft. wide at the widest point. This section of ground is, curiously enough, entirely free from faults of any description. In addition to the overlaps formed, there has been a westerly movement of several hundred feet on the same dykes, proved by their displacement of an old vertical dip dyke to the west of the sketch. It would be expected to find on the underlap portion corresponding to the section in Fig. I. dip faults similar to those of the overlap As mentioned above, however, the underlap portion is free from faults.

Fig. II. will serve to show in cross section the positions of the apices formed by the strike fault and reef at the cessation of movement on faults DF2, 3, and 4 (Fig. I.).

The above example and remarks may prove of interest and use to those who have broken country to deal with. Each faulted section of a mine, however, has to be developed from data gathered on the spot, and no amount of general remarks and rules will prevent mistakes.

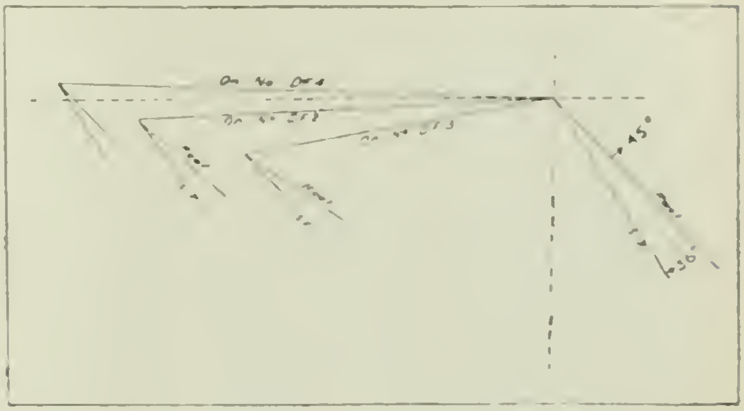


Fig. II.

The next step—after it is considered that the country is well understood—is to lay out a scheme of development which is at once economical and complete. By this I mean that the value of gold recovered from the section must show the maximum profit over the combined development and working costs. This may seem to be an obvious remark, but I have often noticed a penny saved on development cost, increased the mining cost by two pence. The converse is true—it is, in fact, more general to overdevelop than underdevelop.

The cause of much inefficient development is due on a milling property to a natural desire to keep the mill running. The development in consequence does not receive its share of attention. Straight drives in the footwall with boxholes to the reef or reefs, could very often be driven to great advantage in country where the general strike and faults are known. There is a great desire to be always on reef which admits of better sampling, but more often the reason is that the powers that be are rarely averse to a few hundred tons of doubtful development rock for the mill. The result is that future tramming and mining costs do not receive their due consideration. The reef actually broken by driving on reef is not lost if the drive be in the footwall, but can be far more neatly stoped and profitably milled at a future date. Straight driving from point to point has a decided tendency to reduce the total necessary footage. Moreover, every 500 ft. of straight, well-equipped, laid and kept track represents one penny per ton trammed saved over the usual prevailing conditions, namely tortuous drives. Another point which tends to inefficient development is the short and uncertain tenure of office held by the average mine official. An official reaps the immediate benefit of good stoping, but good development work often militates in a small way against present success. An official who would kick at a wasted sleeper has been known to wink at 100 ft. of useless driving—the sleeper is against costs, but the 100 ft. is footage.

It is bad policy to intersect a reef from the hanging side, causing in the future expensive mining and the leaving of valuable pillars. It is generally better, when feasible, where a fault of known displacement has thrown the reef into the footwall, to go back in the drive, and from a point determined in it, to interesct the reef and fault contact on the further side of the fault. The surveyor has in the course of his work the best opportunity of studying the reefs, faults, and dykes in a mine, and should with care and observation make few mistakes. The observation of some underground phenomena has led me to think that there is still a tendency to movement along some fault planes. Allowing that radical dynamic agencies will never again disturb the strata, they may still contain unreleased strains, which, together with the pressure due to great depth, will be released by stoping operations. The pressure at, say, 5,000 ft. will probably cause a virtue to be made of a necessity, by causing driving to be done in the footwall.

In conclusion, let me say that these notes are in no way intended to be exhaustive ; I have, in fact, only touched upon a subject on which a volume might be written. I am hoping that many criticisms and additions will be forthcoming before the last word is said.

Mr. E. Pam (*Member*) : This is a very valuable paper, and I have much pleasure in proposing a vote of thanks to Mr. Beatty. It is impossible to grasp all the points after one reading, and I feel diffident about making any remarks without further thought. The improvement in locating the throws of faults does exist, but I would point out that part of the credit for this is due to the large amount of development done on the Rand, and the improved way in which our records are kept, so that every fault of any importance can be referred to at once, and the reef can usually be located without difficulty. Compilation plans of the districts are also very helpful as by means of them the experience of one mine is of material assistance to its neighbour.

Where working in virgin country such as our deep deeps a model is very useful, and a proposal was made some months ago to build a complete model of all the mines on the Rand, to be kept in a central position for reference. It would be costly I know, but the clear idea which can be got from a model would save much expense in laying out and development work.

Mr. J. Cellier (*Member*) : As regards the models of some of these mines, in many cases the damage is done before the model is made. There are some mines at present that would come out at a loss if they had to make models.

Mr. M. H. Coombe (*Member*) : This is a very interesting paper, a paper which has evidently called forth a great deal of observation on the part of the author. I think we have all need to be thankful that such papers are brought before this Society. We are far more advanced to-day in our knowledge of faults, dykes, second throws etc., than we were some years ago, and I dare say that on the Rand in our practice to day, the general underground man is more conversant with throws and faults than miners in any other part of the world. The reading of this paper puts me in mind of the old days—25 years ago, when we first started the outcrops along these reefs, and when we had no data whereby we could decide whether there was an upthrow or a downthrow. If we were driving east and encountered a fault dipping west, we knew almost to a certainty that we had to go in the hanging wall. I claim that the old miner did very good work in that respect, and the data known to-day for the deeper mines have been compiled from the practical work done by the old miners. They may have thrown away a good deal of money in cross-cuts, etc., but they found it. There is a lot of money needlessly thrown away to-day in these big mines cross-cutting, etc., in impossible directions for the reef ! I know of one special case in one of the leading mines of the Rand where they went 200 ft. into the hanging wall looking for a reef that was 5 ft. in the footwall.

Mr. Chas. B. Saner (*Vice-President*) : As the hour is late I propose further discussion of papers be postponed.

This was agreed to, and the meeting closed.

Proceedings
AT
Extra General Meeting, March 4, 1911.

An extra general meeting of the Society was held in the Lecture Hall of the S.A. School of Mines, on Saturday, March 4th, the President (Dr. Moir) in the chair. There were also present:

36 Members : Messrs. W. R. Dowling, R. Allen, Tom Johnson, E. J. Laschinger, A. Richardson, G. O. Smart, H. A. White, A. Whitby, James Littlejohn, A. F. Crosse, W. A. Caldecott, E. H. Johnson, R. G. Bevington, A. McA. Johnston (Members of Council), W. Beaver, B. V. Blundun, A. J. Bowness, J. Chilton, A. A. Coaton, J. M. Dixon, R. Gascoyne, J. Gray, A. J. Johnson, J. Lea, G. Melvill, P. T. Morrisby, E. Roberts, W. H. Roe, S. Shlom, G. Hildick Smith, W. H. Smith, W. A. C. Tayler, A. Thomas, E. M. Weston and A. Wilkinson.

12 Associates and Students : Dr. J. L. Aymard, Messrs. F. C. Carbis, W. M. Chester, J. Cronin,

H. J. Filmer, J. Gibson, B. W. Holman, A. King, F. J. Pooler, H. Stadler, P. A. Tucker and A. M. Wilcox.

12 Visitors, and Fred. Rowland, Secretary.

GENERAL BUSINESS.

The President : I have first to ask you to accord a very hearty vote of thanks to Mr. Pearce and the other gentlemen who organised that excellent visit to the Crown Mines last week, one which we all so much appreciated. I have also to announce that Mr. M. Torrente has been kind enough to arrange a visit to the Roodepoort United Main Reef Mine for us on March 25th. We have also very great pleasure in welcoming to this meeting Principal Thompson, the new head of the S.A School of Mines. I think it is very kind of him indeed to come to our meeting only two or three days after his arrival here, when he must be worn out with seeing people and trying to remember who they are.

NOTES ON BATTERY PRACTICE.

(*Read at September Meeting, 1910.*)

By A. R. STACPOOLE (Member).

REPLY TO DISCUSSION.

Mr. A. R. Stacpoole (*Member*) : I am sorry my paper missed one of the principal objects that prompted me to write it, which was, to draw the mill men of the Rand into a general discussion.

Mr. E. H. Johnson points out that the reason for removing battery plates from the front of mortar boxes to the foot of tube mills, is to prevent scouring by coarse grit when coarse screens are used. This is a very sound reason, and one which we all admit ; but has the point ever been decided that the gain by this change is greater than the loss in not being able to catch the free gold until it has passed into the launders, pumps and tube mills? Certainly the results from the new plants, which were recently started on the Rand where amalgamation takes place only after the pulp has passed the tubes, leave room for doubt on this point. With more care and time given to the plates which are in front of the mortar boxes, better results might be obtained and less amalgam and mercury would be found in the launders and tube mills.

Mr. Crosse states : "I am afraid that some of our members will consider the loss of amalgam mentioned is much over-estimated." If he is of this opinion, I would ask him to obtain figures as to the quantity of mercury drawn from the mine store for use in the mill for a period of, say, six months, and I think he will agree with me that at least, in many cases, the quantity of mercury lost per ton of ore milled, or per ounce of gold recovered, is excessive. There can be little doubt that where mercury is lost, gold goes with it.

I said : "The mill manager and amalgamator are rapidly having their power and responsibility usurped by the reduction officer and cyanide manager," but Prof. Stanley put my meaning in a nutshell, when he said : "I think if the mill men would make themselves heard a little more they would get on a little more. It is generally the cyanide manager who rises to be reduction officer. Judging from Dr. Caldecott's remarks it was he who originated the post of reduction works manager. I am not prepared to say whether these officials have been a success or otherwise, but what I do know is, the inauguration of this new post has been the means of making all the ambitious mill men invade the cyanide works, and the cyanide men are to be found in all the mills, each gaining knowledge of the others' work with the object in view of qualifying for the much coveted post. This has brought about a better understanding between the two departments.

Mr Alexander states : "I cannot agree that this paper is really representative of common battery practice on the Rand." I also agree with Mr. Alexander on this point ; but I am sorry he did not see fit to take the hint thrown to him by Dr Caldecott, and give us a description of his up-to-date methods of handling amalgam and black sand, etc.

Mr. J. Hayward Johnson is under the impression that my paper was more directed to mill practices in outside districts than to mill practices on the Rand, but this is not so. My experience is that the outside districts pay more attention to the plate and mill recovery than is the case on the Rand. In speaking of sampling the pulp as it leaves the lip of the mortar box, he said : "Owing to the plate being close to the boxes it is very difficult to take an accurate sample." In this I do not agree with him. A reliable sample may be taken from the screen, if each screen has a strip of thin sheet iron inserted in the frame at the bottom, 3 in. wide, and running the full length of the screen ; this will form a sill for the pulp to run off and the sampling vessel can be passed under it the full length of the screen thus enabling a representative sample of the pulp to be caught.

Mr. F. W. Cindel gives the percentage of gold recovered in his mill, but it would have pleased me more if he had given the quantity of amalgam taken from the launders and tube mills, which he naively omitted to do. I hope his general manager will take his hint, and not whisper down the back of his neck, "send a few more ounces of gold in the residue to the dump, but increase the tonnage." Mr. Cindel might find a trolley useful to bring back the "few more ounces of gold" (which he speaks so lightly of) from the dump; he could even employ the much talked of horse to draw the trolley, if he was successful in his search. My critic asks, "what will become of the free gold which concentrates in launders?" A part of it will ultimately find its way to the tube mill plates, and the balance would be absorbed in crevices in the launders, loss in handling when cleaning up the launders, and also when renewing tailing pumps, and tube mill liners, as well as offering a temptation to thieve.

Mr. A. King remarks: "If, however, the tube mill circuit is properly arranged and is provided with the requisite number of cone classifiers with diaphragms, there is no chance of any amalgam overflowing with the cyanide pulp." It is evident from this remark that Mr. King wants to convert the tube mill into an amalgam trap. A large proportion of the amalgam and mercury entering tube mills is ground to floured amalgam, which will float over the plates, and which can be found at water mark adhering to the sides of launders both in the sand and slime plants; it is even questionable if a portion does not float away on the water from the slime plants. My critic is nothing if not emphatic; he settles one point after another in a decided and definite manner, even the knotty problem of tube mills, which is occupying the attention of the leading Rand engineers at present, as to whether tube mills are not more expensive to work than stage crushing by stamp mills. His method of collecting black sands, I will refer to later. I agree with his method of treating black sand residue in a Brown air lift vat by cyanide.

Mr. D. J. Pepler states: "The author must be an absolute stranger on the Rand, for he seems not to have the remotest idea of the methods in vogue in mills here." I may say that it is sixteen years ago since I first worked in a mill on the Rand, and have worked in one of the large mills there as recently as six months ago. Since I first knew the Rand I never lost touch with it, and paid regular visits to it every year or eighteen months, always making a point to see any new machine or method adopted in connection with the reduction works. During my last visit I went to a mill in the Germiston section, one in the central section, and one on the West Rand to see what was new in milling, and it struck me then that the care and time given to the plates and in dressing them was not as great as was bestowed on them some years ago. I have seen, morning after morning, eighteen plates scraped and dressed within forty-five minutes. Does Mr. Pepler think this is sufficient time to scrape, collect black sand and dress eighteen plates in? I know a change for the better has recently taken place in this mill, and that it is to-day one of the most carefully run mills on the Rand. The mere fact that the mine manager requires a large tonnage, or that coarse crushing is carried on, or that the cyanide works are more perfect and capable of better work now than in years gone by, is to my mind no reason why the plates should not have care and time given to them. The objection he brings against cyanide managers becoming reduction works managers is: that they cannot supervise the maintenance of the mill. There are a number of mill managers who hopelessly fail on this point, and a visit to some of the mills will convince one that their maintenance is not a particularly strong point with the mill manager. I cannot agree with his method of collecting black sand; he hangs up the stamps, eight seconds later the water is shut off, and then he states, "by that time most of the tailings will have disappeared, while the black sand will remain on the plate." I would like to know what retains the black sand on the plate? If the plate is examined with a magnifying glass, it will be found that a quantity of black sand is held on the plate by large bits of sand, and if the plate is hard and dry, particles of free gold may also be seen which have not amalgamated; the moment the large particles of sand are disturbed, the gold and black sand can be seen moving down the plate, and are washed away with the tailing. I want to ask Mr. Pepler what would happen if the flow of water is too strong, or if the valve is stiff and cannot be shut off for half a minute, or a minute, instead of being shut after eight seconds? Will the black sand be still on the plate, or will the plate be washed bright and free of the greater part of the black sand? He gave the following figures:—"Tube mill plates, black sand gave 59%, scrapes and other sources 41%; in the battery, black sand 42%, scrapes and other sources 58%, which should be considered satisfactory." Why should these figures be considered satisfactory? Is it because 5, 10, 15%, or even more of the black sand and free gold which had rested on the plates are washed down into the tailings launder? Mr. A. King advocates Mr. Pepler's system of collecting black sand, but I consider it risky and haphazard, and when dealing with a concentrated product which yields 50% of the total mill gold, no chance should be taken.

Mr. G. A. Robertson remarks; "Instead of the author then showing us some method whereby a saving might be made in the free gold, mercury and amalgam when they find their way into the cyanide works, he proceeds to tell us how to collect sand from plates, &c." My paper was not written with this object, but with the object of preventing free gold, black sand, mercury, and amalgam from entering the cyanide works, which can be best done by paying great care to the dressing of plates, giving sufficient time to each plate, careful collecting of all the black sands, and by not using mercury to excess. An ounce of mercury should be to the amalgamator what a pound of cyanide is to the cyanider. The cyanider is never tired of trying to curtail the quantity of cyanide, and not only every cyanider on a mine, but every mill-man can tell you the quantity of cyanide used per ton of ore treated; yet how many amalgamators can tell exactly the quantity of mercury used per ton of ore milled, or per oz. of gold recovered?

If the practices of each mine manager were more generally known and criticised, each would be able to discard his own weak or bad points and adopt the better ones of his neighbours, which would be a distinct advantage to the industry. Mr. Robertson quotes from authorities on the loss of mercury: "15% of the entire mercury fed is lost; and it is well known that the amount of gold carried away and totally lost by this means is enormous." These quotations go more to strengthen my remarks on this point than to weaken them. These losses are admitted by the mill men on the Rand and accepted as inevitable, as not one of the mill men have come forward to tell us what is being done to prevent this loss of amalgam.

I am sorry Mr. Robertson did not give us figures as to the percentage of mercury lost, instead of the percentage of mercury recovered by riffles; it appears that what is recovered is history, but what is lost is mystery, and judging by the strain of the criticism on my paper, the mill man is determined that it will remain a mystery.

The President: This seems to have produced a very profitable discussion, and I hope our friends the mill men will give us something more this year to keep the interest up.

THE CLASSIFICATION OF TAILING PULP PRIOR TO CYANIDING.

(Read at October Meeting, 1910.)

By Edward H. Johnson, M.I.M.M. (Past-President).

DISCUSSION.

Mr. J. Vine Jory (*Member*): In commenting on this paper I do not propose to deal with the theoretical considerations of the classification of tailing pulp, but with a few mechanical points of obtaining classification in this type of plant, which have developed during the experience gained during the last nine months in the continuous operation of the apparatus.

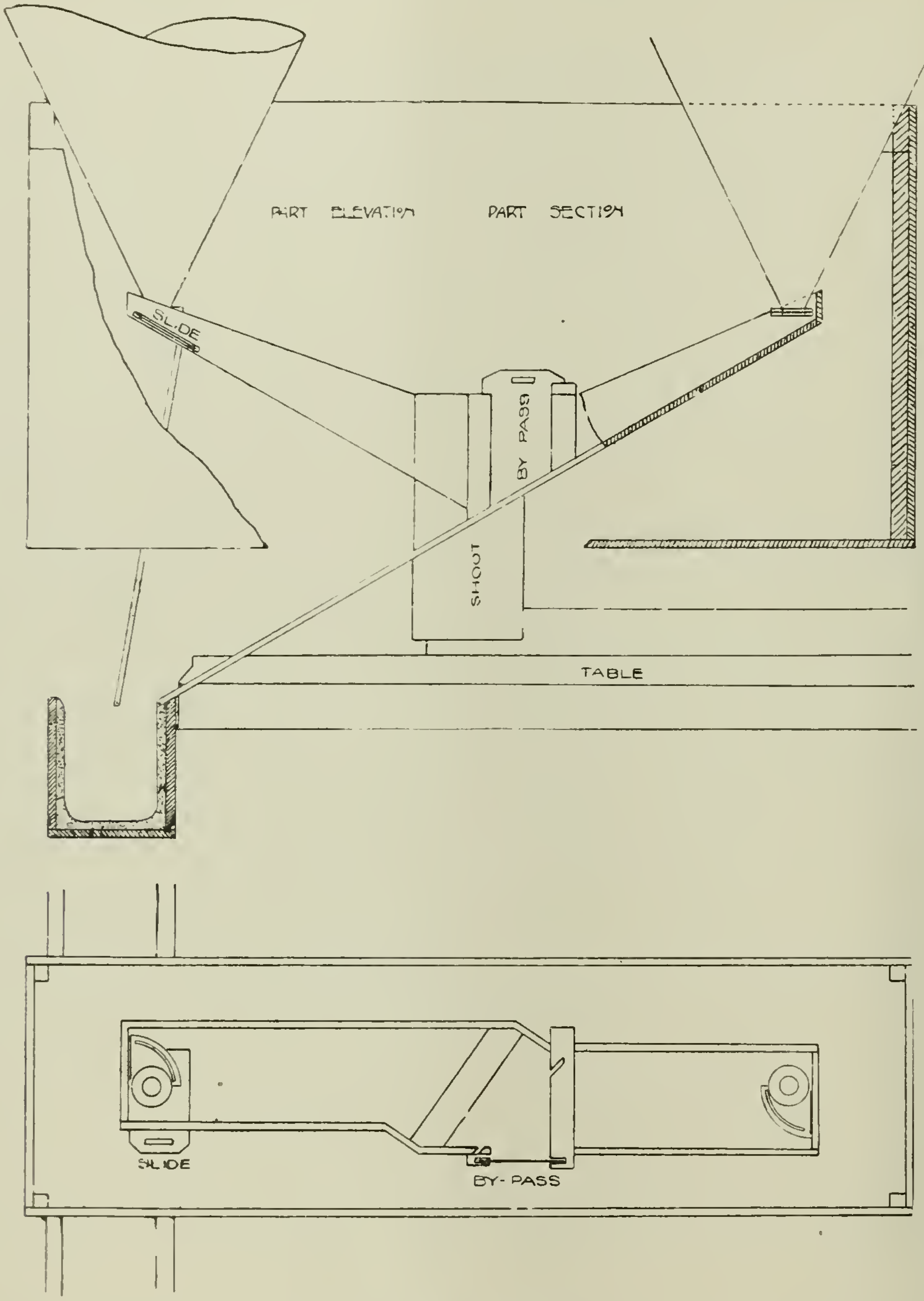
Primary Cones.—As in any settling apparatus, or differential settlement as in the case of classification, the amount of agitation caused by the feed of fluid pulp must be controllable. There should, therefore, be as little fall as possible between the outlet of the feed launder and the surface of the cone. Also provision should be made to eliminate the occluded air in the pulp at the centre of the cone, otherwise coarser material than is desirable will overflow. This point was probably emphasized in our case by the pulp elevation being by air-lift. A wide sleeve surrounding the sub-level inlet and carried slightly lower and above water level will correct the agitation effect of the entangled air. Similar provision was also made in the return sand cones with a reduction of +200 product in the slime from 4% to under 1%.

Secondary Cones and Vacuum Tables.—The vacuum tables have given little trouble, but we have found it advantageous to substitute loose jute cloths for the finer calico originally used. Although the time taken for dressing a table is very small, the shutting off of the underflow of the cones during this operation interferes with that continuity of action of the cones which makes for good work. For this reason the by-pass shown in the accompanying sketch was introduced and has proved itself extremely useful. The introduction of the underflow of the secondary cones direct into the stream of solution for the short period necessary during dressing tables has not caused any accumulation of solution to take place. This underflow of secondary cones carries between 26% and 28% of moisture, the average of the table moisture being 12%.

Sampling.—The continuous sand plant lends itself to very accurate sampling of both sand and slime products, as neither is complicated by the return-sand spitzkasten as is usual in pulp sampling.

The extraction results obtained during the past six months may be of interest to members:—

Tons sand treated 152,930 = 47·97%, of tons milled
Average grading + ·01 in. 12·1% + ·006 in. 32·3%
− ·006 in. 55·6%
Average original value 5·118 dwt.
Average residue value 0·595 dwt.
= 88·37% extraction
Average final draining 0·06 dwt.
Tons slime treated 165,298 = 52·03% of tons milled
Average grading + ·003 in. 1% − ·003 in. 99%
Average original value 3·076 dwt.
Average residue value 0·214 dwt. = 93·04%
Average washed residue value 0·08 dwt.



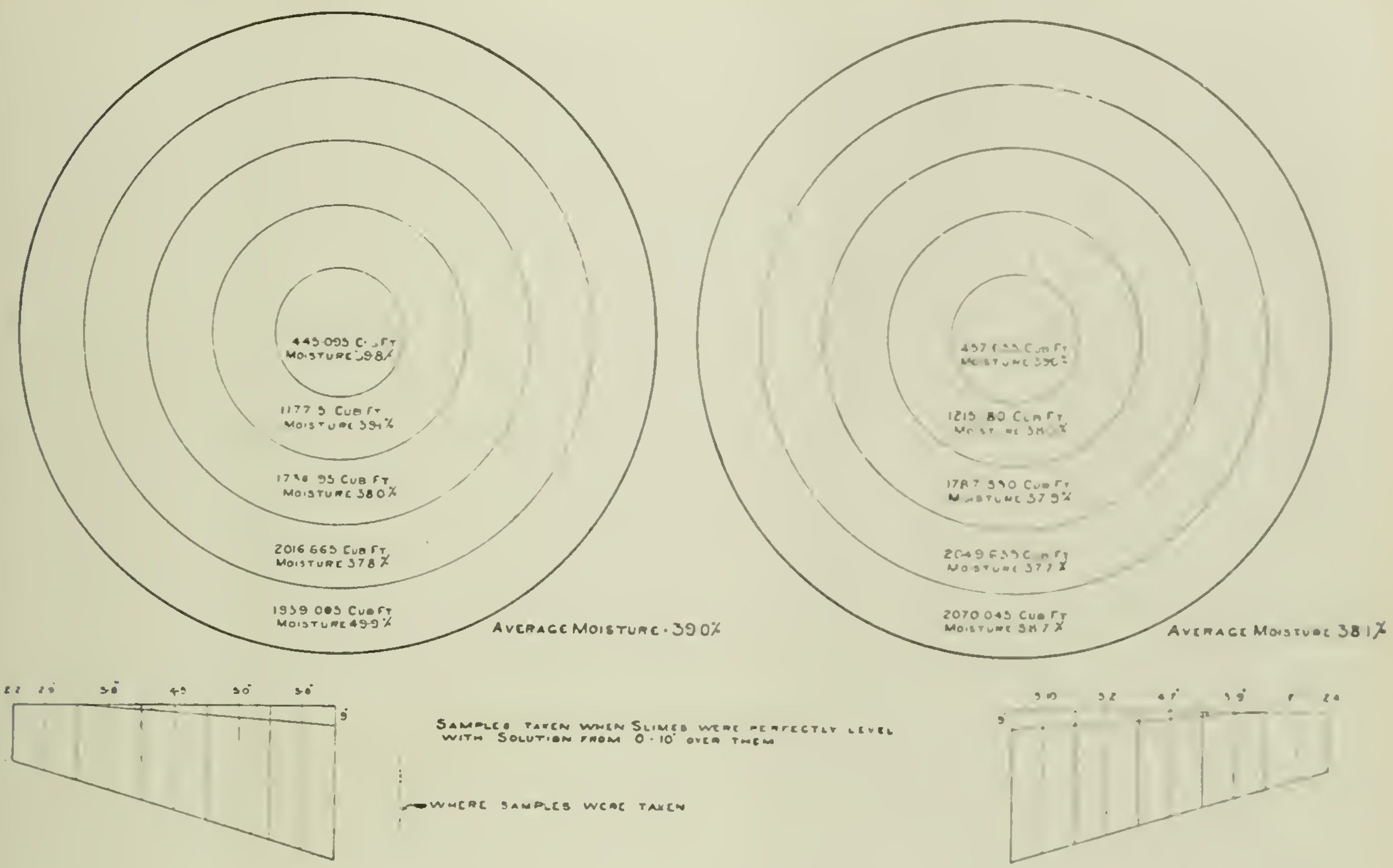
The cost of operating the continuous sand plant, including all white and native labour, power, maintenance and re-elevation of all the sand and solution 48 ft. has worked out over the tonnage dealt with at 1·159d. per ton.

I am indebted to Mr. Penlerick, general manager East Rand Proprietary Mines, for permission to publish these details of extraction.

REPLY TO DISCUSSION.

Mr. E. H. Johnson (*Past-President*): The discussion elicited on this paper has been almost entirely of a confirmatory nature, and I have to thank Messrs. A. McA. Johnston and G. O. Smart for the very valuable data they have adduced, and for the leniency of their slight criticism. It is somewhat surprising that, although at the present time, some quarter of a million tons per month are locally being classified by the means described, or approximately $13\frac{2}{3}$% of the total Rand tonnage, only two members have contributed to the discussion on the classification aspect.

Mr. McArthur Johnston confirms so very closely the data given in my paper that there is little to reply to. I think, however, he perhaps over-emphasizes the increased length of time of treat-



ment due to the East Rand single treatment, as compared with the double treatment, and overlooks the relative initial values of the material dealt with, viz., 6·00 dwt. at the East Rand as against 2·88 dwt. at the Simmer Deep. It is fair to assume that a little longer treatment would be necessary for the higher value. The slightly longer treatment is, however, I think more than compensated by avoiding the cost of the double handling needed for the double treatment, as with the perfectly clean sand obtained the same ultimate residue is attainable. His later reference to the experiments he made of dropping the transferred sand into solution is confirmatory of this, as they very closely imitate the conditions of single treatment.

Mr. Smart's data on the results obtained from single cone classification are extremely valuable as showing what can be done where the plant does not lend itself to the elevation required for the installation of the secondary cones, but where treatment of the sand as collected is practised, the installation of secondary cones is preferable. I I must thank Mr. Smart also for his grading of battery feed, which more or less confirms that I quoted. The proportion of fine in battery feed will necessarily be a fairly variable quantity according to the nature of reef and mining condition, not to mention, as perhaps treading on dangerous ground, the economy or otherwise (principally otherwise) in the use of explosives.

I was particularly pleased that Mr. Smart, through the instrumentality of Mr. McLean had been able to practically confirm my experience of the changed conditions that the better classification has produced on the density of settled sand and slime. Mr. McLean's work in this direction is a model of careful determination and could very well have formed the text for a paper dealing with the subject of tonnage measurement of moist settled solids. The difference of 4% higher moisture content of the Simmer and Jack settled slime as compared with East Rand Proprietary Mines is quite comprehensible when the difference of 12% and 52% respectively of the mill product collected as slime is considered, the latter naturally containing a higher proportion of the finest sand.

Replying to the queries asked:—

(1) The long 1 in. pipe was at right angles to the cone of the vat, and when falling ran along the guides at that angle.

(2) The terms 1st, 2nd and 3rd sections are, perhaps, East Rand Proprietary Mines localisms, for collecting, 1st wash and 2nd wash sections of slime plants.

(3) The impact of the falling cylinder was found to displace a little of the fluid surface slime, which contained approximately 60% moisture. The correction, however, did not affect the total determination.

Regarding the difference Mr. McLean shows between the moisture content of the peripheral and central samples of a settled slime charge I am indebted to Mr. P. R. Nethersole for the accompanying diagram showing that these differences do not occur to the same extent on the East Rand Proprietary Mines as on the Simmer and Jack. The reason is due to the concave cross section of the settled charge and the relative lengths of decanter arms. The East Rand Proprietary Mine decanters are 12 ft. long therefore do not reach the concave centre (a defect I confess), consequently the supernatant solution in the centre corrects the greater density of the settled solid.

Mr. Tonnesen's contribution is particularly valuable, as giving us, what many of us have sought, a convenient and accurate core sampler for settled slime. His apparatus has been in daily use on the East Rand Proprietary Mines for some time now, and has proved itself a great improvement on the cumbersome device used in the original determinations. I should like, not only to congratulate Mr. Tonnesen on his successful solution of what has hitherto baffled most of us, but also to thank him for providing us with such a neat and convenient tool.

Mr. Jory's somewhat belated contribution contains several notes that I hope will prove of service to those operating this type of plant. I should like to take this opportunity of thanking Mr. Jory and Mr. Lloyd, who are in charge of the respective East Rand Proprietary Mines continuous sand plants, for the enthusiasm they have shown in their working, which has contributed so largely to the success this type of plant has achieved on the East Rand Proprietary Mines.

The President: As I have said already, it is most gratifying to find our Past-Presidents taking such an interest in the Society, and I am sure we all thank Mr. Johnson very much for his interesting reply. As an example of the activity of our Past-Presidents I may point out that at this extra meeting we have no less than five of them present, practically all who are still resident in Johannesburg.

NOTES ON THE OCCURRENCE AND CONCENTRATION OF TIN ORES.

(Read at October Meeting, 1910.)

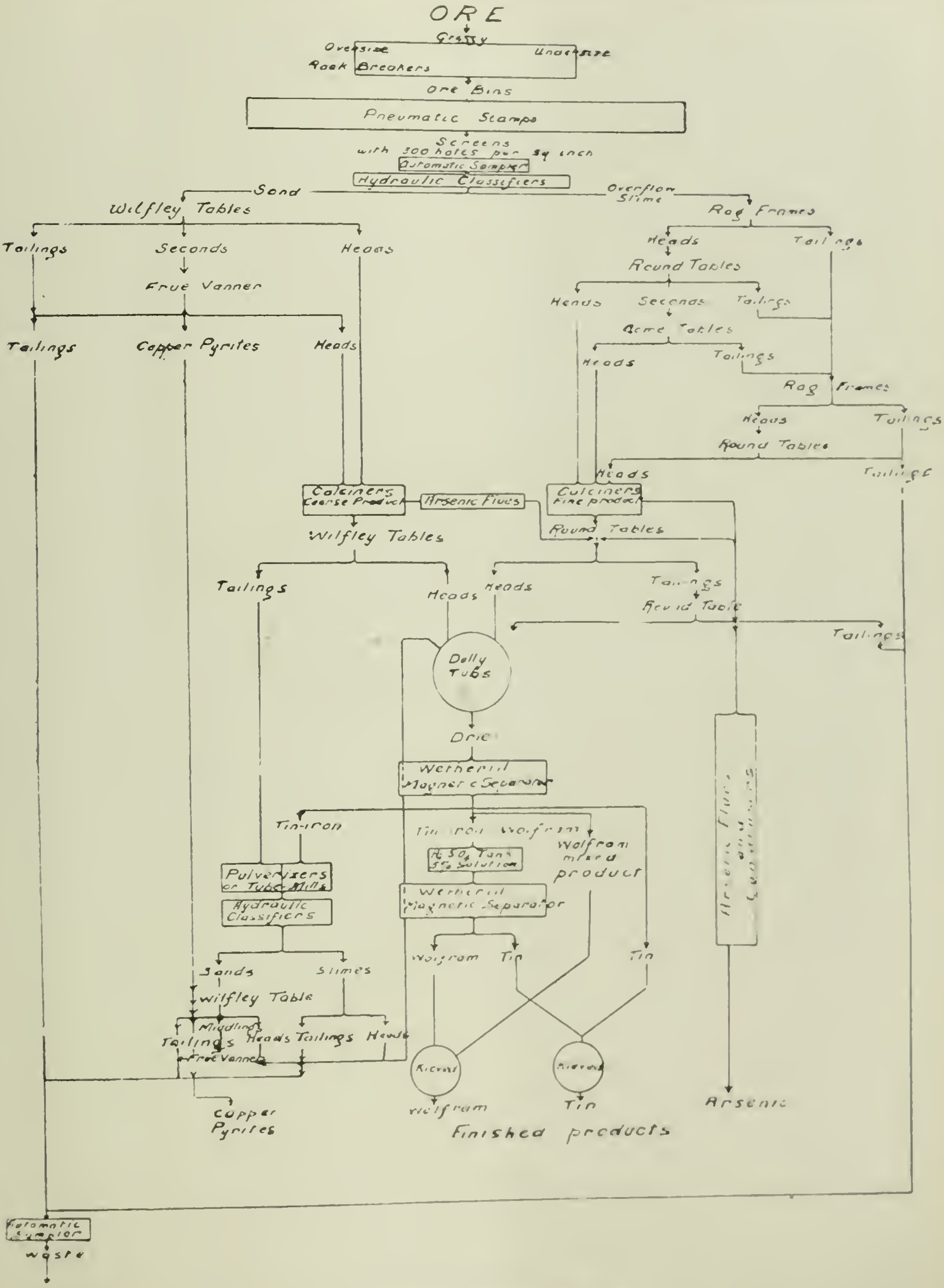
By C. Fred. Thomas (Member).

DISCUSSION.

Mr. A. Treloar (*Member*): If Mr. Weston could not congratulate Mr. Thomas on his paper on "The Occurrence and Concentration of Tin Ores," I am afraid I cannot congratulate Mr. Weston on his contribution to the discussion. It is always an easy matter for anyone to write sarcastically on any subject, provided he has sufficient sarcasm in his nature, but sarcasm and criticism have an altogether different value.

With the concentration of tin ores, as well as with most scientific treatments, it is wise to observe with an unbiassed mind and to take a broad view. Perhaps the author did try to cover too much ground in his paper, and it may seem as if some of his statements are rather vague; at the same time there is much truth in what he says, and seeing that he is the first contributor of a paper on tin concentration read before this Society, I think he is to be congratulated on the result, in spite of the fact that he happens to be a Cornishman. Mr Weston evidently speaks under some provocation when he says "that the Cornish tin expert is prone to strut around the world posing as the one and only authority on the ore and its concentration, and inflicting out-of date Cornish dressing devices on young and unsophisticated tin industries in other countries." Might I ask, where this has been done? And does he not know that in South Africa, and in many other parts of the world, the tailing from some of the mills (erected *not* by Cornishmen) is richer than the ore of the average Cornish mine, and that if the same percentage of loss were escaping in Cornwall the tin streamers on the Red River would sell, not 30 tons of tin monthly, but nearer 100 tons, and that if the Red River were red through ruby tin escaping it would then contain enough tin to cause it to stand still.

In order to discuss percentages of recovery in tin concentration it is necessary to see the samples taken and also to know the assay value of the pulp from the stamps. A Wilfley concentrator if properly erected and adjusted loses practically the same amount of tin in the tailing from an ore containing 1% as it does from an ore containing 3%, and, naturally, the percentage of recovery in the first instance would be much less than in the latter—a fact often overlooked. To provide for the irregularity of the percentage of an ore it is necessary when laying down a plant to add an extra concentrator for the specific purpose of using it when the value of the product goes up. Supposing the average loss from a concentrator of the Wilfley or Buss type when doing good work on sand to be 0·2% metallic tin, then if the sand pulp contained 5% and 0·2% was lost, the result would be a recovery of 96%; if however the sample assayed 1% and 0·2% was lost, it would mean a recovery of 80%; thus in the case of two mines equal work might be performed and yet the results widely differ. The above



Flow Sheet for Complex Tin Ores.

is an illustration of sand concentration; and of course a further loss would occur with the concentration of the slime, still further reducing the percentage of extraction from the original pulp sample It is customary to think that with high values you get high losses, but that is not true when comparing values with losses. A machine works irrespectively of values, and however well it is erected and adjusted there must and ever will be a loss, but if the load is regulated according to the value the loss is fairly constant.

It is scarcely necessary for anyone with even a small knowledge of mining conditions in Cornwall, to hesitate to reply to Mr. Weston's remarks as to why the mines do not put down a succession of rag frames. I will, however, answer by informing him that, unfortunately, nearly the whole of the mining area is owned by one owner, and he, for personal reasons, lets only a portion of land to the mine, generally not nearly sufficient for the plant required, and then lets the adjoining portion to a tin streamer, giving him the right to come close to the mine discharge, the streamer paying a toll in the form of one-eighth of the gross return. The price of land to the mines, if purchased, works out at £100 per acre, or over £200 per morgen, not 5s. per acre as in South Africa, or a comparatively small price as in Australia. Mr. Edward Walker (whom I have the pleasure of knowing) ought not to ask why more rag frames are not put down after spending so much time as he did in Cornwall, and anyone conversant with Cornish mining should not have to ask for information seeing that the question of "Lords's dues" has for the past five years created such a stir in Cornwall, and especially during the past three months. I may say, to give credit to two of the owners, that in the case of the Carn Brea and Tincroft mines, during the three months mentioned, they have given up their dues and added an installation of plant to assist the mine, but this was only done under pressure and when the capital of the mine was practically exhausted. Might I also inform Mr. Weston that in every case I know of the slime is treated twice, and often three times if space permits, and that the average percentage of slime caught on a rag frame is not 5% of the product fed on, but nearer 65%, as he will discover if he makes a proper test.

For the last century the tin mining world has been waiting for a slime concentrator, and it still waits in spite of the Lührig and other machines; and if Mr. Weston has had the privilege of testing many of these slime concentrators he could only have come away with a sad heart. Personally, I have found that with a wooden bed set at a good angle and the surface kept roughened, a revolving table is the most effective machine, and with three or four times treating I have got a higher percentage than from any other type. The Lührig, Wilfley slimer and Frue vanner, are only effective on certain material, but for a very fine product, such, for instance, which will pass through 20,000 holes per square inch, the results they give are not satisfactory. I might also say with reference to Cornishmen refusing to adopt up-to-date methods, that these (so-called) up-to-date methods have done Cornwall irreparable injury during the past five years, and that out of a nominal capital of £750,000 spent there is nothing to be seen to-day but machinery standing idle. The good work done by the Germans at Clitters resulted in the payment of a dividend and the stoppage of the mine after a nominal capital of £160,000 had been spent. A new company was formed in 1908 with a capital of £15,000, and this company is now in liquidation, and, as far as I know, never paid a dividend. I could mention many others where the results have been similar, and one, an American company having the North Crofty mine, spent an enormous sum installing a plant of their own design, with disastrous results as far as outputs and dividends were concerned. If by these things Mr. Weston considers Cornwall owes a lot to either Germans or Americans I must beg to differ. I grant that Germans are, as a nation, extremely clever in scientific research, and that they introduced magnetic separation into Cornwall, which has helped several mines considerably. It is generally known, however, that these revolutionists have returned home with much less money and a great deal more experience. It is a well known fact that experience is necessary when dealing with any particular ore, and the treatment which does well in Australia is not workable in South Africa, and the same applies to Cornwall, and it behoves any mining engineer to first consider and experiment before making a definite statement.

I agree with the author that the average extraction does not exceed 80%, considering that in many mines where the ore is poor, it does not reach that. There are to-day, when tin is fetching nearly £200 per ton, many idle mines in Cornwall with ore of an average value of 1%; although mining, with crooked shafts and over 2,000,000 gallons of water, from some mines, to pump daily, with "Lords' dues" and a complex ore, is done for an average cost of 23s. per ton. It should then not be difficult, if such marvellous results can be obtained in Australia and elsewhere, to make these mines pay. If the cost of tin mining in South Africa could be obtained, the information would be most valuable to a new industry, and would brush away some of the misceptions prevalent.

I append a flow sheet of a plant treating a complex ore, where the slime is treated three times and where the extraction is equal to 82% only. By referring to the sheet it will be noticed that four products are sold, viz., copper pyrite, wolfram, tin and arsenic. The pulp sample assayed 0·01% for copper, 0·015% for wolfram, 1% for tin, and 1·35% for arsenic. As the flow sheet clearly shows the treatment it is unnecessary to furnish any explanation, except as regards that pertaining to the copper pyrite, which being simple might be useful to someone running a small plant, possibly at their own expense where copper pyrite is present. It was found that the waste oil from the bearings of six Frue vanners, treating middling from twelve Wilfley tables, got into the boxes with the water used to wash the concentrate off, and this water with the very light skim of oil carried with it a large percentage of copper pyrite. To collect this the water was allowed to overflow into another box and from that into a launder 30 ft. long and 1 ft. wide. A piece of perforated copper grate was fixed across the launder at the end, with a piece of cloth placed across it to act as a filter. So well did this answer that practically all the copper pyrite was collected in the launder and the water which passed through the filter was quite clear. About 30 tons of copper pyrite were collected yearly by this means, assaying on an average 17% copper, which were sold to a smelter. The wolfram, as the sheet shows, was extracted by magnetic separation and assayed when finished 69% tungstic acid. After the concentrate was roasted and the wolfram extracted, the tin was treated in the usual way by dollying, etc., and assayed 65% metallic tin. The arsenic was collected in chambers and assayed when sold from 90% to 94%. I might add that the water used for treating the roasted concentrate was allowed to flow over a series of settling tanks into which was thrown scrap iron, and a little copper was collected by precipitation, but owing to the largest percentage being saved in the launder referred to, previous to the concentrate being roasted, the percentage of copper from these tanks was very small.

I very much question if with a complex ore it is possible to obtain a much higher percentage of extraction than 82%, although with an ore free from impurities of the kind mentioned above a little higher percentage might be recovered, but that would depend on the constitution of the ore, and whether fine crushing was used to eliminate the tin from the waste.

Mr. E. M. Weston (*Member*): As Mr. Treloar's contribution to this paper is mainly a criticism of my criticism, the President has allowed me to supplement my remarks. Regarding my attitude Mr. Treloar does not understand the tradition of this Society. The more pungent and sarcastic criticism is, as long as it is fair and impersonal, the more members, and in most cases the person criticised, enjoys it. I think the atmosphere is a healthy one, and Mr. Treloar will have to try and get acclimatised. If Mr. Treloar had understood this he would not have been so simple as to vehemently deny the fancy derivation I invented for the origin of the name Red river.

Regarding rag frames, Mr. Treloar has quite missed the point of my criticism, which was not why more frames were put down, but why the Cornishman used so unsuitable an appliance.

If Mr. Edward Walker did not know it was impolite to ask awkward questions after visiting Cornwall, I am not prepared to apologise for him. I would draw Mr. Treloar's attention to the remarks of the Cornish correspondent of the *London Mining Journal* of a recent date, who wants to know why one tributor easily draws 10 tons a month from the Red river with primitive appliances. And in connection with losses I would draw his attention to the prospectus of a company to treat the Carn Brae dump of 500,000 tons, containing 17 lb. of tin ore per ton giving a recoverable value of 5s. to 10s. per ton. If, as Mr. Treloar says, a rag frame extracts 65% of the slime the three treatments he mentions would recover 95% of slime values, so I am afraid there is a mistake somewhere; for the tin is in the Red river, and it comes from the mines. I note in the Rooiberg plant he has installed callow screens thus justifying my criticism on the paper.

Mr. Treloar favours slime tables of the Acme type and has installed them at Rooiberg. Tables of this class were installed 25 years ago at Mount Bischoff and do splendid work in series. I believe I am correct in saying that the metallurgists there are replacing them by machines of a more modern type which they have found better. Clitters I believe failed not because the ore dressing was bad, but because the Clitters mine was worked out, and there was more water than ore in the lower levels of "Kingston, and this has been largely the reason that these revolutionists returned home with much less money and a good deal more experience." They put up expensive plants to treat ore that investigation proved existed only in the imagination of the gentleman who tempted them. We all know in every country the fairy tales about the rich ore "the old men" left and the marvellous lode at the bottom of the flooded winze, and if the foreigners failed, not to get the tin from the ore, but to get the ore from the mine, apparently Cornishmen, at least, did not suffer.

And now having said most evil things I could think of about Cornishmen and their methods, I should like to add that I could say most of it about the men who have managed mining in every field in the world I know of, including the Rand and Australia. Mistakes have been made and money squandered in vast sums everywhere, and if some of us got the chance we would, no doubt, make some large and healthy mistakes ourselves. So honour to "the old men" Cornish or otherwise, for they were strong, brave men, and did their work in the world with their might which is more than can be said of some of us though we have better knowledge and better tools.

REPLY TO DISCUSSION.

Mr. C. F. Thomas (*Member*): I may say that the concentration of cassiterite is a widely discussed problem, and plants must vary according to the particular ores to be treated. If the ore should be chloritic and contain sulphides, float slimes are to be expected.

Mr. Walter McDermott's paper on "The elements of slime concentration" read before the Institution of Mining and Metallurgy in April, 1910, together with the discussion and contributed remarks thereon, contains valuable information. Mr. W. S. Welton gave some interesting figures on float tailing with a concentration to only four times the value of mill feed, proving great losses on silver, copper and lead ores. Mr. Weston is amazed to learn that wet concentration for the extraction of the sulphides of copper is unsatisfactory. It is a surprising confession to make, and a careful study of the matter might prove there is much to learn on the subject. It is necessary to carefully note to what extent concentration has to be effected. To bring a copper ore to four times its value as a mill feed is not to be compared with a concentration of a cassiterite ore from 1% to over 80% cassiterite. There is a peculiar affinity between certain sulphides and oil. The Elmore Vacuum Process has done good work in preventing such heavy losses as have been often proved in the wet concentration of sulphides. Broken Hill, in Australia has had great difficulties in wet concentration. Broken Hill, in N. W. Rhodesia, has proved that with a complex ore the use of wet concentration is a failure.

It is unwise to make rash statements in a criticism unless one has definite proofs, and I regret having to show that Mr. Weston was extremely rash in many of of his remarks. "The extraction of mispickel is low by wet concentration and selection previous to milling is advisable." Mr. Weston evidently does not understand the meaning of this statement by me and then endeavours to prove it incorrect. Having considered he had proved the statement wrong, he says "the author really should try and realize Cornwall is not the universe or even the hub of it." Mr. Weston has mentioned Clitters and I believe he visited these mines about 1907. This plant erected bv Germans, which "gave the Cornishmen ocular demonstration in the art of ore dressing under modern conditions," is now idle. They selected mispickel previous to milling, roasted it, and then sent the residues to the mill, to increase the extraction. The Luhrig vanners were not successful on this particular ore. The loss of mispickel through float tailing was high. The Chairman of the Company was an Englishman, the Consulting Engineer was an A.R.S.M. and received his first practical experience in Cornwall The manager was a Cornishman. The jigs were rarely at work; the Buss Tables made by the Luhrig Co. gave most excellent results, and great credit is due to the Company for introducing the Wetherill magnetic separator into Cornwall.

Stannite is a comparatively rare mineral found near the contact of slate with granite. Mr. Weston misquotes and says "it is not always found in slate near granite." I agree with him on this point.

Production. — "Australasia, Cornwall and Africa produce 12% to 13% of the worlds production." In regard to my quoted statement Mr. Weston says "the author appears ignorant of the fact that very extensive deposits of alluvial tin have lately been discovered in the Niger Protectorate" which bid fair to compensate for the stationary or declining output of the Straits Settlements and Australia. The Niger Protectorate is in Africa and I did not consider the output of sufficient importance, to mention under the heading of production. Bolivia was mentioned and its output has rapidly increased. One Bolivian mine is producing more tin than all the mines in Africa. The Province of Yunnan in Western China is producing approximately 4,000 tons of tin per annum. The Niger Protectorate may rapidly increase its output, but recent discoveries of cassiterite have not been considered of such importance to effect the market, and the price of tin metal has risen substantially in the last two to three years.

The Editorial Committee is next blamed for my using the expression "fluidity of the tin product." This seems unreasonable. It was stated that this particular concentrate was almost free from impurities which might have a hardening tendency on the metal. The so-called English tin is several pounds less in value than that from the Straits Settlements. Before the Bolivian concentrates came on the home market, English tin was higher in price than that from the Straits Settlements. If one were to ask a smelter the reason, he might have something to say on fluidity.

"Sand from a modern plant with good classification should not be worth further treatment (or the tailing from the tables in front of the mill can be made too low in grade for further consideration). The object of wet concentration is to eliminate the maximum amount of crushed ore as waste in the first concentration.." This quotation from my paper should be plain enough to be understood by one who has even a small idea of wet concentration. Of course Mount Bischoff takes a middle head from the tables for regrinding, and also sends a sand tailing to waste.

Ilmenite is often associated with cassiterite, and Mr. Weston makes a bold statement when he says it calls for magnetic separation methods. If the concentrate is fine, I venture to say, leave magnetic separation alone until a better method for separation is found. The magnetic separation of slime has not been a success up to the present. Again, ilmenite is often present and is not liberated from the cassiterite, even after fine crushing. A test made after roasting a concentrate from a 30 mesh screen gave the following results:—The concentrate from the furnace assayed 51% Sn. Re-concentration on a No. 5 Wilfley table gave at the head 54% Sn, and at the tail 46% Sn. I would like to ask my critic whether there is any reasonable excuse for calling ilmenite, cassiterite.

It has been proved that with depth in granite, sulphides and wolfram decrease and the cassiterite becomes finer grained. "Sizing for fine grained cassiterite is unsatisfactory." In stating this I, according to Mr. Weston, show my ignorance of modern practice. The notes define fine grained cassiterite as yielding a concentrate of which over 90% will pass a 150 mesh screening.

The Callow screen is doing excellent work on the discharge from a 30 mesh screening, but what a suggestion to size for almost a slime!

In my paper I mentioned that "Elevators for pulp should be avoided if possible." Mr. Weston says it is not at all necessary to avoid pulp elevation. Who would not avoid elevating pulp containing 5% to 7% cassiterite, if possible?

Great attention has been paid to the slime question in wet concentration throughout the mining world, and particularly in Cornwall because the ore slimes to a great extent. Some mines are filtering the overflow water from the final concentration of slime through coke, etc. Such firms as Messrs. Fraser & Chalmers, Luhrig, Humboldt, etc,, are keenly interested in the matter. Mr. Walter McDermott says:—"In considering the progress of concentration methods for the last 30 years, it appears possible to draw certain conclusions which may be considered as elements governing the probable developments in the future. To those who have been in touch with this branch of ore treatment, it is very evident that there has been nothing revolutionary in the best practice for the period mentioned."

Mr. Weston must have solved the concentration of the slime, as he states "to me the explanation seems quite simple." I grant the Red River is returning 10% to 12% of the value that the mines are selling. It is probable this river is sending into the ocean 7½% to 10% of the output of the various mines. The river and its banks are a mass of slime concentrating plants for over 8 miles in length. All those who may be interested in the wet concentration of slime, will be anxious to have this explanation.

I purposely refrain from replying to certain remarks of Mr. Weston, and regret that prejudice is distinctly shown in the remarks I refer to.

Replying to the remarks of Mr. Amos Treloar, I may say that my paper purposely covered a wide area, and I hope it will lead to some papers on the concentration of tin ores. Recent experiments on slime containing 0·5% cassiterite prove a fair recovery of tin metal by means of electric smelting, but from an economical standpoint they have proved a failure. The percentage of recovery in tin concentration may not have received the attention it deserves, but it is difficult to obtain reliable information. Chemical analyses on low grade tailings are not always satisfactory. Not only the tin mining world is interested in water concentration, but the mining world has long been waiting for an efficient slime concentrator. It is probable that the cost per ton mined on the tin mines in Africa will prove exceptionally high.

Replying to Mr. Weston's further remarks, I may say that the recoverable value of Carn Brae dumps may average 6 lb. of cassiterite per ton. My sampling of these dumps gave an average of nearer 10 lb. per ton, but some waste burnt leavings exceeded 17 lb. of cassiterite per ton.

The Transvaal Consolidated Lands quarterly report to December 31, 1910, under the heading of "Groenfontein," states:—"It has been decided to re-treat the sand residues, which are estimated at 11,000 tons, assaying 1·29% metallic tin."

Other "waste" tailing dumps in the Transvaal will pay to re-treat, although a considerable loss has occurred through slime. I still venture the opinion that the average extraction obtained from mined cassiterite does not exceed 80%.

THE MINE DUST PROBLEM.

(Read at January Meeting, 1911.)

By Dr. J. L. Aymard (Associate).

DISCUSSION.

Mr. G. O. Smart (*Member of Council*) read the following contribution to this discussion :—

Mr. A. W. Stockett (*Associate*) : The model exhibited shows an improved method of manufacturing Dr. Aymard's collaring iron. In the original design this is made from a piece of steel plate, cut as the development of a frustrum of a cone, and then lap-rivetted, entailing considerable work. The present model is made from a piece of solid drawn boiler tube, which is cut to the required length with a saw, and drawn cold to the conical shape on a mandril by a single blow under the steam hammer. The cost of the material is about 6d., and the cost of manufacture is reduced to a minimum.

Mr. Tom Johnson (*Member of Council*) : Dr. Aymard in his paper seems to have used a terminology of his own, which makes it rather difficult to criticise his paper.

From what I can gather the author seems to think there must be some difference in principle between the ventilating of a gold mine and a coal mine. There should not be—at least, not in the sense that the author means. Of course there is a difference, for, as a rule, coal mines are well ventilated artificially, and gold mines mostly ventilate themselves, which is certainly no credit to anyone. Nothwithstanding the fact of these terrible disasters we hear of, coal miners understand ventilation, and for the author to say that they must stop these disasters before they can teach us ventilation, is like a medical student telling the doctors he is studying under that he can learn nothing from them, until they can guarantee the life of every patient they accept, which would be rather absurd.

The author understands very little of mine ventilation, and is not in a position to judge of other people's knowledge of the subject. For instance, in the exhibited model of a portion of a mine; the upcast headgear was bratticed or closed in ; this he said contained a small fan, but it was not an exhaust fan. Now I should like to know what kind of a fan it was. Then he speaks of the practice of leading the foul air through the whole workings. I should like to know where this happens, as it would be rather a curious thing to see on the Rand in any of our self-ventilated mines. After all his condemnation of sending the foul air all through the mine, however, he calmly commends the Village Deep installation, which is simply passing the foul air of the Village Deep through the whole workings of the Village Main. There is some qualification needed somewhere, else he is straining at the gnat and swallowing the camel. I think he has missed his vocation in trying to teach us ventilation.

Again, he asks us who is to be told off to regulate the regulating doors at all times of the day and night. I think I will be a Scotchman for once and answer his question by asking him another. If he had a patient with a wound in which he, the doctor, had inserted a drainage tube, would he plant a nurse there to watch the drainage tube the whole time, when he knows from his experience that the tube would be all right for a number of hours without attention? The author should have read some elementary work on ventilation before writing his paper, and he would have found that there are thousands of regulating doors in use with no one sitting beside them.

In another place he speaks of "short circuiting the air." What does he mean by the term? He does not use it with the same meaning as it is generally used: I think he means *splitting*. Most of the trouble of poor ventilation in our mines is caused by short circuiting of the air. Then again he says, "without seriously interfering with the system of natural ventilation existing on many mines, I think the more extensive use of return foul air pipes along the roof of the drives would be beneficial." What does the author mean by this? Does he not know that the system of ventilation, natural or artificial, makes no difference ; the means of coursing or guiding the air are the same, *i.e.*, how the air is produced has nothing to do with how it is used.

Natural ventilation is one of the systems of production and belongs to the *science* of ventilation. Air pipes, doors, etc., used in coursing or guiding the air belong to the *art* of ventilation.

Who told the author that the ladderways were the most used path for travelling? In most mines they are not, so one can only judge whether a mistake has been made by making the ladderway the return, after knowing the conditions.

The author disagrees with the M.R. Commission for wishing old stopes to be closed off, then calmly recommends the same thing, and I think for the same reason, viz., to prevent them being made into latrines.

Although I cannot admit that the author understands ventilation, I do admire his perserverance in trying to help us with the dust question. I heartily agree with him that this question has not advanced much the last ten years. At the Geldenhuis Deep in 1901 we tried sprays, atomisers, jets, etc., and found that washing down the place before blasting was the most effective method for allaying the dust raised by blasting, and that a small jet playing on the rock under the hole whilst drilling dry holes was good.

I have kept dust down very well with the ordinary milk tin, that is, keeping the face wet, but I did not use a $2\frac{1}{2}$ in. or 3 in. bit to start with. I used a 2 in. bit which made much less dust. This is one of the points that needs looking after—the less dust there is made the less trouble to prevent it getting into someone's lungs.

Under the problem politic the author uses his new terminology again. The machines that are used three on a bar are percussion machines and the others must be air hammer machines. The author speaks of the difficulty of getting men to carry out the laws, but the difficulty is more apparent than real. If an Inspector of Mines took it into his head to visit, say, the development work on any mine, he should, on visiting the various places, be able to judge if the rules and regulations were being carried out, and if not, who was to blame. I think one great reason that things are not better is that *we* officials are much too easy, and are as much in fault as the men. If we only did our own duty, and made the men keep their places damp, or summons them for not doing so, we should make a great stride forward towards knocking out the dust question. It would cost fewer pounds in fines than we have lost lives through it.

The President: Dr. Aymard has a new device to show us: it has nothing to do with dust, though it comes under increased efficiency.

Dr J. L. Aymard (*Associate*) here exhibited a patent leg protector for the use of natives in the mines, saying:—"The hospital statistics show that injuries to boys' legs are very frequent and quite apart from the number of boys who go into hospital there must be a large number of boys who receive injuries to their legs, and who are working partially incapacitated. Boots and leggings have been tried but had to be given up, as the boys would not use them. A boy cannot walk up a slippery stope with boots on, so I have devised this modification of a boot. The idea is for the boy to wear out his own sole, and as ordinary leather would be perfectly useless in a short time I have substituted raw hide and turned the hairy part inside. I have made it perfectly flat, and turned it in so as to protect the toes. The boy simply slips his foot into it. It will probably last for a very long time as raw hide can be used in water for a long time before it becomes soft. I have had them placed on a boy's legs and thrown rocks at him without his feeling any hurt. The only question is that they are rather expensive, because raw hide is different from ordinary leather, and you can only make about eight pairs of these out of one hide."

Mr. G. Hildick Smith (*Member*): The following are a few points which have occurred to me in reading the author's interesting paper on "The Mine Dust Problem." With regard to his remarks on mine ventilation in which he says that it is the idea of promoting further discussion on the subject, which has prompted him to include it—ventilation—in his paper, I think a full discussion on the problem of ventilation could with ease be carried on *ad infinitum*, and at the finish there would still be a doubt as to whether the correct solutions had been arrived at. What is really wanted, in order that the question of ventilation may eventually be thoroughly worked out, is actual practical experiments carried out by competent men in a thoroughly scientific manner from the results of which reliable formulæ can be deduced and standard methods adopted. All through the underground work in mines there is too much guessing, and this is a point which cannot be too strongly emphasised. It is due to several causes:—

(1) The great cost which would be incurred in making underground experiments. To commence with a State mine is required, either an old mine, which has been practically worked out, or a mine abandoned for various reasons would have to be acquired to start with and run as a combined training and experimental mine by the Government in conjunction with the Chamber of Mines. The whole might be controlled by a Central Committee consisting of representatives of the various mining houses and the Government, the mine itself being under the management of a thoroughly trained engineer. Or an alternative method to this would be the appointment of an underground experimental committee, who would appoint suitable men to experiment on various subjects underground on any of the mines of this field, permission having first been granted by the management of the mines for the experiments to be carried on, expenses being defrayed by a general experimental fund. We on the Rand like, at any rate, to pride ourselves that we are working on the most efficiently new and up-to-date mines in the world, therefore let us show that we are.

(2) Underground work is not an ideal occupation, it is too dirty, unhealthy and altogether too much against the grain to suit many men. Yet it is where the real spade work is done, and therefore where constant attention is most needed. There is much too much of the "look at the plan" game and tell them what to do about it

still. Experiments in chemical, mechanical and metallurgical laboratories are a very nice form of interesting amusement when one can sit on a stool in the fresh air and smoke and watch a beaker full of water boil, etc., but when it becomes necessary to spend every day of a week underground, for example, in order that a given experiment might be carried out, it becomes a different matter.

Ventilation can be taken as the first item on the programme, which has reached a stage where experiment is now needed and not discussion.

As to the "medical aspect" put forward by the author. We are here again guessing at present. The writer would like to suggest that experiments should be conducted on mine rats. It would be necessary first to determine whether the lungs of the rats living in the mines are affected with silicosis, if so, then healthy rats or other animals should be obtained, kept underground, some continually, others during the various shifts only. Some in cages or what not, so that they cannot inhale dust, and others along side them so that they can inhale dust, and after various periods post mortem examinations should be carried out on their lungs, etc. The smaller animals theoretically should be affected in a shorter time than human beings, due to their quicker breathing. Much could possibly be learnt in this way.

Coming now to the question of the respirators and their uses. I must first thank the author for saying that his work in this direction is entirely due to the suggestion made by me at one of our previous meetings. Not having been at the January meeting, on reading through the author's paper I was suddenly confronted with this announcement, and received at first a slight shock, but on second thoughts a further stimulus to work in this direction, in which I am firmly convinced that we have a sure preventative for phthisis for those who really are anxious to have one. The baffle principle can hardly be termed new, when in reality it is the principle, although on a much smaller scale, that prevents dust from passing through a damp sponge or two pieces of damp sponge superimposed through which all air inhaled has to pass in a zig-zag course before entering the mouth, depositing on the way any dust particles. I have here the type of sponge that I find handiest after having for several months experimented with all kinds. A rough practical test as to whether any dust particles pass a respirator can be carried out by means of some substance, such as snuff or pepper, which may be dusted around the outside of the respirator when in position, and deep and quick breathing resorted to, care being taken to protect the eyes. I have not tried this test with the author's respirator yet, but I think from an inspection of his respirators that the baffles are not close enough together to prevent air carrying dust from passing obliquely through the passages without striking any part of the baffles. Another point to be raised against respirators large enough to cover the nose and mouth is the difficulty of obtaining a good fit against the face and also the difficulty of being able to see over them when looking downwards. With a respirator such as I have here it is necessary to use small pieces of cotton wool in the nostrils, but a good fit against the face is assured and also there is no difficulty in being able to see just as well as without it in any direction. In my work as a shift boss I find no difficulty whatever in breathing through my respirator even after having run up 100 ft. of vertical ladders in order to test it.

There are just a few remarks I should like to make with regard to shift bossing. The author is correct in stating that it is the most arduous work on a mine, but he is wrong in stating that shift bosses to-day are called upon to do more work than formerly, the opposite being the case in all the well organised mines, especially in the older mines which require the most careful supervision to ensure safety. Here again we have an instance where it would be possible to adopt a proper standard in underground work. In very many cases the shift bosses finish their shift, finished physically, and when the dose has been repeated for many months, even when strict training has been resorted to, in order that a constant fitness for work may be kept up, yet eventually the result is, that the majority of shift bosses become so continually tired that they drift into nothing more or less than walking machines, not having the time or inclination to think. The remedy for such conditions is more shift bosses with better supervision and fewer accidents. The standard round of a shift boss should be set by the possible comfortable walking distance for underground, so that undue physical strain can be avoided. The standard distance allowed would vary somewhat as to whether a mine was a new developing mine or an old mine; in the former case the time taken up in walking, will be a greater percentage of the total time spent underground than in the latter case where more time has to be spent in more careful supervision. I would suggest as a standard day's walking underground a distance of about three miles with supervision of about 25 white men and 250 boys as a good day's work for a shift boss.

Mr. E. M. Weston (*Member*): I have marked a few passages that seem to call for comment.

Personally, I have always had the idea that in trying to shut-off disused stopes and other places, the current would lose its velocity and become short circuited. Now, dealing with this matter of miners' phthisis, I should like to give you my recollection of some interesting conversations with a medical man in this town, which seemed to me to throw a good deal of light on the subject. It has already been noted that the amount of silica in the lung does not bear that proportion you would expect to the tissue of the lung when the lung is examined after death. I really forget what proportion of quartz was found in some of the lungs incinerated, but it was something very small indeed, and the slides which Dr. Moir showed us recently, raise that question again. It is probable the size of any of these pieces of quartz shown in them would be something smaller than 1/100th millimetre, and you would have noticed that the area taken up by the quartz was really very small indeed. To come back to my conversation with a medical man, I said to him: "Is it not true that the *small* proportion of solid matter existing in the air, which each one of us—without reference to miners—each one of us living on the surface must breathe into our lungs would suffice to choke them up entirely in the course of time if there were not some provision for removing matter reaching the lungs, as well as for preventing it from entering them!" The doctor replied: "That it was very well known, that Nature had several methods of removing ordinary foreign matter from the lungs." He said "the white corpuscles of the blood had the power of carrying away foreign matter from the lungs: that there were two glands where this foreign matter was deposited; that if we examined these glands (in the case of a person living in smoky cities) we would find them full up with smuts!" He said "that in the case of people who are breathing quartz dust, nature is continually at work removing these grains of quartz, perhaps as fast as they enter the lung:" but he said that "in the case of miners, the sharp character of the dust wounds the tissues of the lung, and that the first stage of miner's phthisis was generally the irritation caused by the presence of these wounds, caused by the particles of quartz. These particles of quartz may be removed after they have wounded the lung, but a certain scar remains as the effect of the wound." He went on to say: "that the second stage of progress of the disease was caused by fibrosis, due to these wounds. The formation of these scars or fibroid growths is a gradual process, reducing the area of lung available for oxidizing the blood, and if the scarring is extensive enough, may cause death, without any complications due to the presence of germs." That seemed to me a clearer explanation of the disease than I had ever heard from anyone else. No doubt learned members of this Society knew all about it before, but I think Dr. Moir will tell you that what we do not know is just the different effects on the lung of different sizes of quartz particles. Dr. Aymard says that many of the drives on the mines have a very peculiar stalactitic appearance, and that they ought to be nicely trimmed down. Well, unfortunately, mining is not an exact science, and we can scarcely guarantee that our drives shall always be as nicely and cleanly excavated as Dr. Aymard would wish. I do question the advisability of his recommendation, when he says that the whole of the drives in the mines should be systematically washed down once a week with some suitable disinfectant. Considering that in the Village Main Reef there are 20 miles of drives, it seems to me, one would have to increase the labour efficiency to a very large extent. I think Dr. Aymard is to be congratulated on the work he has done in attempting to stop the dust at the mouth of the hole. I think this question of miners' phthisis is largely a question of education. If we could drill into the managers the loss they suffer from the destruction of their most skilled and efficient labour: if we could only drill into the workers what they are exposed to, and how to avoid these dangers, good would come. The process may seem sometimes heartbreaking, but I agree with Mr. Johnson, that it is a process which, in the long run, will bear fruit. I think, also, that Dr. Aymard's suggestion that hose pipes should be armoured with wire, covered by tin, is a very good one.

Mr. R. Gascoyne (*Member*): I think it would have been better if Mr. Weston had followed his own advice to Dr. Aymard and confined himself to things he understands, and that his lecture on the medical aspect might have been left alone. The author, in referring to Mr. Penderick's paper on East Rand ventilation, criticises that paper, but has evidently forgotten that the paper was devoted exclusively to the East Rand Proprietary Mines system of ventilation and was not intended to apply to any other mines. I wish to say that it is a very good system, and whatever fault there may be in carrying out such a system, it will be more in connection with the details than the general system. The author also criticises coal mine ventilation systems, but I agree with Mr. Tom Johnson when he said that it would have been much better for the author to have studied some elementary work on ventilation before tackling the subject. The author is not aware that this dust question in coal mining is one of the most serious questions we have to deal with, far more serious even than in the case of metalliferous mining. Explosions do not happen through lack of ventilation in coal mines as generally supposed, but through the presence of coal dust, and great attention is therefore given to the removal of this dust. Coal dust is formed in a much different way to the dust in gold mines. The coal is, of course, much more fragile than local rock, and forms dust at every operation it undergoes, even during its transport along the drives of a coal mine. On that account it constitutes one of the greatest risks in colliery operations so that great and constant care has to be exercised so as to keep it within safe limits. The author says that the air enters along the hanging wall and returns along the floor. As a matter of fact in a deep mine with no system observed it will be quite the reverse. With regard to letting the foul air through the whole workings, I think that is never done when the mine is of any extent. Mr. Johnson has pointed out an instance where the ventilation after passing through one mine and becoming polluted is taken through the workings of an adjoining mine. Well, that is not a good system of ventilation, for every mine where possible should be self-contained and have its own upcast shaft. You will agree with me, I think, that whilst miners do not complain about the foul air generated in their own mines that it is scarcely courteous and polite to ask them to put up with other people's stinks. Of course the author thinks that managers ought to sandpaper their drives and make them smooth so as to reduce the friction of the circulating air. It is not the height or cleanliness of a drive that makes it better for ventilation, but it is the area of a drive rather than the height and the velocity at which the air travels that decides the friction. As Mr. Johnson points out, no one is wanted to watch regulators. In fact in a well arranged mine the ventilation should be self-acting. No one will depend entirely on doors or regulators, and they should only be used when there is no other alternative. There are such things as overcasts, and their use would go far to get rid of the artificial element in mine ventilation. As a rule the fewer doors you have in a mine, the more reliable the ventilation should be, and by adopting these overcasts the number of doors is reduced to a minimum. The author says that the recommendation of the Mining Regulations Commission to split the air cannot be carried out. As a matter of fact good ventilation in a large mine depends upon splitting the current; and cannot be obtained without judicious splitting. Mr. Weston also refers to the question of having the disused stopes and drives closed up. The author says they ought to be left open, as they afford a reservoir of cool air. As a matter of fact in a deep mine, these abandoned stopes, drives and other places will rather form reservoirs of hot foul air, and they ought to be closed up so as to obviate this accumulation of foul air and so prevent a sluggish current of air in the mine. Now I come to the question of the medical aspect. One would think it is rather a damaging confession for the author to say that the medical aspect of this question remains about the same as it did ten years ago. We mining engineers are inclined to ask why has he not given more attention to the medical aspect and less to ventilation, if that is the case, because in the medical aspect he ought to be better versed than in the ventilation of mines, no matter how well intentioned and praiseworthy his efforts in this dust question may be.

I think we are indebted to the author for the way in which he has taken up this matter, and can only regret that his paper does not go more fully into his promised methods of dealing with it for the benefit of those who have not seen it We are quite aware he has done it solely for the relief of suffering humanity, and we ought to encourage him all we possibly can. It is the duty of every mining engineer and mine manager to assist him in every possible way. There is one thing we are not surprised at, and that is the conclusion he has arrived at that miners are sadly indifferent in regard to this question. It is very unfortunate, but easily understood why it is so with such a slow and insidious disease. He has used some very strong language with regard to this indifference, so it may have the desired effect. No doubt if the mining regulations, as a whole, were carried out in a thorough manner, this indifference would cease to exist. Let us hope that the new efforts now being made in carrying them out thoroughly will have the desired effect, and that the miners' indifference will cease to exist.

Mr. F. J. Pooler (*Associate*) : I will not presume to criticise, I merely wish to ask one or two questions with regard to the medical aspect of the question. Some time ago I saw an article discussing the relationship between moisture in the air and the working efficiency of those people who are working in that air. There was a statement about the loss of efficiency of the lungs in purifying the blood as regards addition of oxygen and removal of carbon dioxide, and I would like to know :—

1. Is it a fact that increased moisture lessens the power of the lungs to absorb oxygen ? If so, are any tables available showing this effect ?

2. What connection is there between human working efficiency and percentage of moisture in air?

3. Are miners more liable to lung troubles in damper mines than in drier ones?

Perhaps the author can give me some information on these points.

Dr. J. L. Aymard (*Associate*): I may say that Dr. Porter himself gave me some information which is not exactly such as to be favourably received by the mining community. It was that Prof. Haldane holds that there will be no difficulty in mining at very deep levels owing to the humidity of the air, but you will have to reduce the working hours. I think it is well established that human efficiency is largely decreased by humidity in the air. The great effort we should make is to keep the air as dry as possible.

Mr. E. J. Laschinger (*Member of Council*): It seems to me that a certain degree of humidity is necessary for efficiency. The point to be ascertained is what is the exact amount necessary. In certain diseases of the lungs a fixed humidity point is prescribed for the cure of patients. I have recently read that in America in private houses, schools, etc., humidity of the air is regulated and maintained at its efficient point. Everyone who has given any great attention to the matter knows that if the air is saturated it has a bad influence on the muscular system. This fact was brought very plainly to my notice years ago when taking underground temperatures. The moisture thrown off by the skin does not dry away, and you are bathed in your own sweat and heat. Schemes have been brought forward lately for regulating humidity, and I think from an engineering point of view, if the medical authorities can say what ought to be the proper humidity of the air, it is not without the bounds of possibility to do this on a practical scale. It is a problem which will have to be taken in hand before long, especially in very deep mines. If we can get the air cool as well as regulate its humidity, it will be all the better. This is an engineering matter which, of course, will require money, but it is not without the bounds of possibility to artificially maintain an atmosphere that will conduce to the maximum output of labour

The President: I may point out that the consensus of medical opinion is that 70% is about the best humidity for human life. In the Rand mines we have a humidity always over 90%, generally near 100%, and therefore the mines have no chance of reaching the best conditions. Our only hope is to bring it down from 100 to 90.

Mr. E. J. Laschinger: My contention is that it is not at all impossible, but that it is within the bounds of practical engineering to regulate the humidity and temperature of air in mines without excessive cost.

The President: All the better: I am glad to hear it.

Mr. E. M. Weston (*Member*): Is not our trouble this: that as the mines grow deeper and hotter, and no doubt drier, we must use water largely in our stopes and drives to allay dust, and under these conditions the evaporation of this water must tend to increase the humidity to a dangerous degree, even if we dry and cool the air as Mr. Laschinger suggests before it enters the mine.

The President: Yes, but the question of watering dust underground only refers to development work, whereas the air of the whole of the other workings of the mine is saturated to begin with.

Dr. J. L. Aymard (*Associate*): Another interesting point is this The engineers at work in the engine room of the Atlantic boats work under great heat, and engineers have told me that men can work long hours without trouble providing they perspire very very freely. When a new engineer goes down he is watched very carefully to see the amount of water he drinks and the amount he perspires. If he does not perspire freely he is sent up and his hours of work are reduced. Another point overlooked is the question of altitude. There was an interesting paper read recently and very little attention has been paid to it. A man coming to work here may feel a change for a short time, but I am sure compensation takes place. Our blood is supposed to become very thick, and to have a very deleterious effect upon the heart. I think it is purely a question of getting acclimatised. I believe you can live here more healthily than in any other part of the world if you only get acclimatised. Of course living at one altitude and working at another is different, and this is what the deep level miners do.

Mr. Gascoyne (*Member*): My experience is, that working in a temperature of 90 and over, is very detrimental to the efficiency of the workmen. I have found that working in that temperature, men are not able to do a full day's work. In deep underground working, when drives have been exposed for any length of time, they gradually cool down, and air can be delivered to the working face at practically the same temperature as it leaves the bottom of the downcast shaft. It is only a question of frequent splits, having special drives, and delivering the air to the advanced workings without coming in contact with other workings *en route*, that will keep the workings as cool as possible and increase the efficiency of the men. In very deep mines, in the newly-opened working faces, the ground is perceptibly warm, and the return air, after having gone through the working places, is distinctly hot and moist, and carries a good deal of heat and moisture out of the mine, so that ventilation must do a great deal towards helping to keep up the efficiency of labour in the deep mines.

Mr. E M. Weston (*Member*): In the silver-lead mines of Przibram, Bohemia, they are mining below 3,000 feet, and the Government who own the mines are making provision to supply the miners with, not 60 cubic feet per minute, but with 200 cubic feet of air per man per minute, so as to keep the efficiency of their workmen up to the mark. It is only a question of time when the fact must be recognised that the mines working below 4,000 or 5,000 feet must be prepared to supply air in somewhat the same proportion, and this air may have to be dried and cooled to keep down the heat and humidity of the workings to a level that will render human labour possible, let alone properly efficient.

Mr. Tom Johnson (*Member of Council*): We must use water in our working places, and if we increase our quantity of air we can carry much more water than we do when using a small quantity of air.

Mr. R. Gascoyne (*Member*): When Dr. Haldane said that men were not able to work more than six hours, he was probably referring to coal mining at say 3,000ft. and over, but I consider that similar conditions cannot be expected on the Rand until we attain a depth of 10,000ft.

Mr. T. G. Martyn (*Visitor*): There is one point which has not been mentioned this evening, *viz.*, that as the fresh-air circulation is increased in a mine, the evaporation of moisture is greatly increased, and with every unit of moisture taken up by the air, you will have a *corresponding reduction of temperature*; the spontaneous evaporation of water being almost the most effective way of cooling anything. If we increase the current of unsaturated air, the cooling effect will be enormously increased at the same time that the percentage of moisture in the air supply at the working face is reduced to a lower and healthier figure.

Mr. G. A. Robertson (*Member*) (contributed): Water spraying underground is only in a measure effective so long as the faces damped retain moisture. In connection with the present crusade against coal dust in collieries I noticed in an English paper recently that extensive experiments were conducted in a coal mine with a view to finding a suitable fluid for spraying purposes. It was found that water spraying had to be repeated more often than was practicable, and the spraying of soapy water seems to have been the best solution tried due to the fact that after evaporation has played its part a pasty substance is left to which the dust adheres. Where practicable in our mines here the idea is certainly worth a trial.

The meeting then closed.

Visit to the Crown Mines, Limited.

About 150 members of the Society availed themselves of an invitation from the Directors and Management of the Crown Mines, Ltd., to visit and inspect a portion of the surface works of their great property on Saturday afternoon, the 25th February.

The visitors were received at the mine by Mr. R. C. Warriner, the General Manager, Mr. H. T. Brett and Mr. Pitchford (Joint Managers), Mr. Fraser Alexander (Reduction Officer), Mr. S. H. Pearce (Consulting Metallurgist), and other members of the staff. Under this able direction they were shown the new crusher stations, the arrangements under construction for the more economical handling of ore from the shafts to the mills, the Butters' filter plant (which Mr. C. G. Patterson demonstrated and explained), the new staff quarters, the inhabitants of which were envied by many visitors not so advantageously situated, and the quarters for the men who are being recruited in South Africa for the purpose of being trained as miners.

At the conclusion of the tour of inspection, which lasted over two hours, the visitors adjourned to the Manager's beautiful grounds, were they were received by Mr. and Mrs. Warriner, Mrs. Brett and Miss Pitchford, and hospitably entertained. Dr. Moir (the President) in expressing on behalf of the members their thanks to Mr. Warriner, Mr. Brett and Mr. Pitchford and the staff for the instructive and pleasant afternoon, paid a well merited tribute to the management for their efforts to improve the conditions under which the men lived.

He also desired to thank especially Mrs. Warriner and the ladies for their courtesy and kindness in looking after them so well that afternoon. Mr. Warriner briefly responded, and the gathering dispersed.

The following description of the property was circulated to the members for their information:

The property embraces, from the east, the Robinson Central Deep, Crown Deep, Crown Reef and South Rand, — Langlaagte Deep, Langlaagte Royal and Paarl Central on the west, and in addition 1,278 claims of deep level ground. In extent it is about three miles from east to west on the strike of the reef.

There are on the property, nine main shafts and seven crusher stations when including those at the Paarl Central and Langlaagte Royal, and the property is being so laid out, both on the surface and underground, that in the future there will be only two large crusher stations ; one at No. 5 and one at No. 7 shaft. By a system of underground chutes and haulage, the rock from the Langlaagte Royal, Paarl Central, Langlaagte Deep and western areas will be concentrated at No. 7 shaft and hoisted there, while all the rock from the eastern section will gradually be concentrated at No. 5 shaft. These two shafts will have to handle between 9,000 and 10,000 tons daily or the full requirements from the mine. The two shafts will be connected on the 13th level at a vertical depth of approximately 2,200 ft. This level will be a main haulage level, the main drive being 14½ ft. wide and practically straight from one end of the property to the other. This drive will be served by electric haulage so arranged that ore may be delivered to either shaft.

No. 7 Shaft.—This shaft is at present vertical to incline with three hoisting ways and pump way. Alterations are well in progress to convert the hoisting at this shaft into double stage with four hoisting ways in the vertical, hoisting four ton skips, and four hoisting ways in the incline, hoisting five ton skips, all hoisting being done electrically.

No. 7 Shaft Crusher Station.—This is the western station of the Crown Mines, and is being equipped to handle 3,500 to 4,000 tons per day on day shift only. Special features are three 30 in. conveyor belts delivering into nine 30 in. × 12 in. Hadfield & Jack crushers. The return side of these belts are utilised for waste which is delivered to a transverse belt delivering to the waste bin and thence to dump. The rock from the crushers is caught in bins of 1,000 tons capacity, thence it is discharged through pneumatically operated doors into 40 ton hopper trucks. These trucks are hauled in train by 50 ton electric locomotives, and may be delivered to anyone of the mills of the Crown Mines. The trucks are constructed so that their discharge may be pneumatically controlled from the locomotive, thus reducing the labour required to a minimum.

No. 5 Shaft. — Total vertical depth to be 3,400 ft. Three main haulage levels will be established in this shaft at distances of 600 ft. apart, vertical. These main haulage levels will be the 13th, 16th and 19th, and they will control all the ore down to the South Rand dyke. On each of the levels electric haulage will be installed. At each of these levels the ore bins will have a capacity of 2,000 tons, and winding will be done by three 8-ton electrical winders—as large, if not larger, than any electrical winders in the world.

No. 5 Crusher Station.—This station will be identical with that at the No. 7 shaft, excepting that its capacity will be larger, being about 5,000 to 6,000 tons per day of 10 hours.

Metallurgical.—The present equipment is 675 stamps with 20 tube mills, and sand and slime plants made up as follows : —

A. Crown Deep	300	stamps and	10	tube mills
B. Langlaagte Deep	200	,,	6	,,
C. Crown Reef	120	,,	3	,,
D. Bonanza	55	,,	1	,,

With the present plant we have a capacity of 165,000 tons per month.

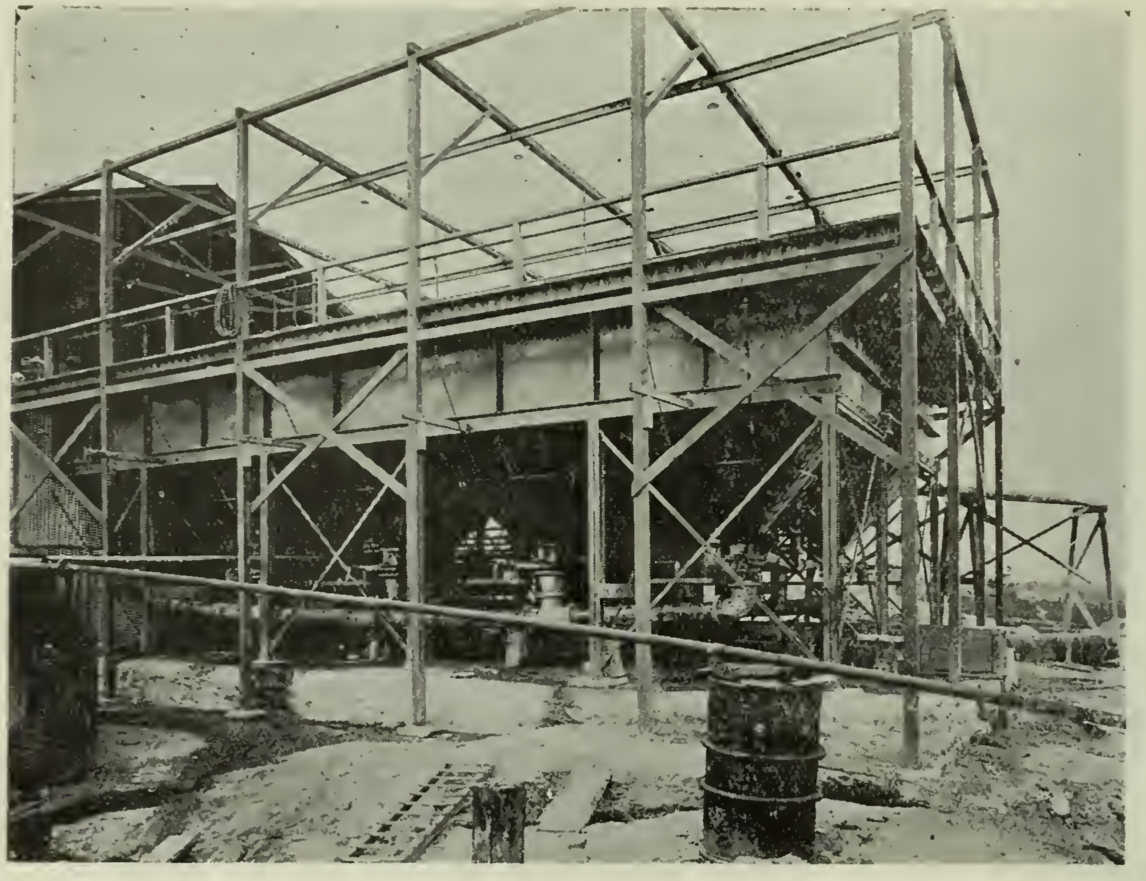
A fifth mill is being erected of more modern type, which, when completed, will increase the total crushing capacity to well over 200,000 tons per month. Special features are the tandem classifications and the tube milling at A plant.

The installation of a 300 leaf Butters' vacuum filter slime plant at C plant, which is unique not only on account of its being the first vacuum filter plant erected on these fields, but that it should occupy the site of the first slime plant designed and erected on the Rand by Mr. J. R. Williams in 1894.

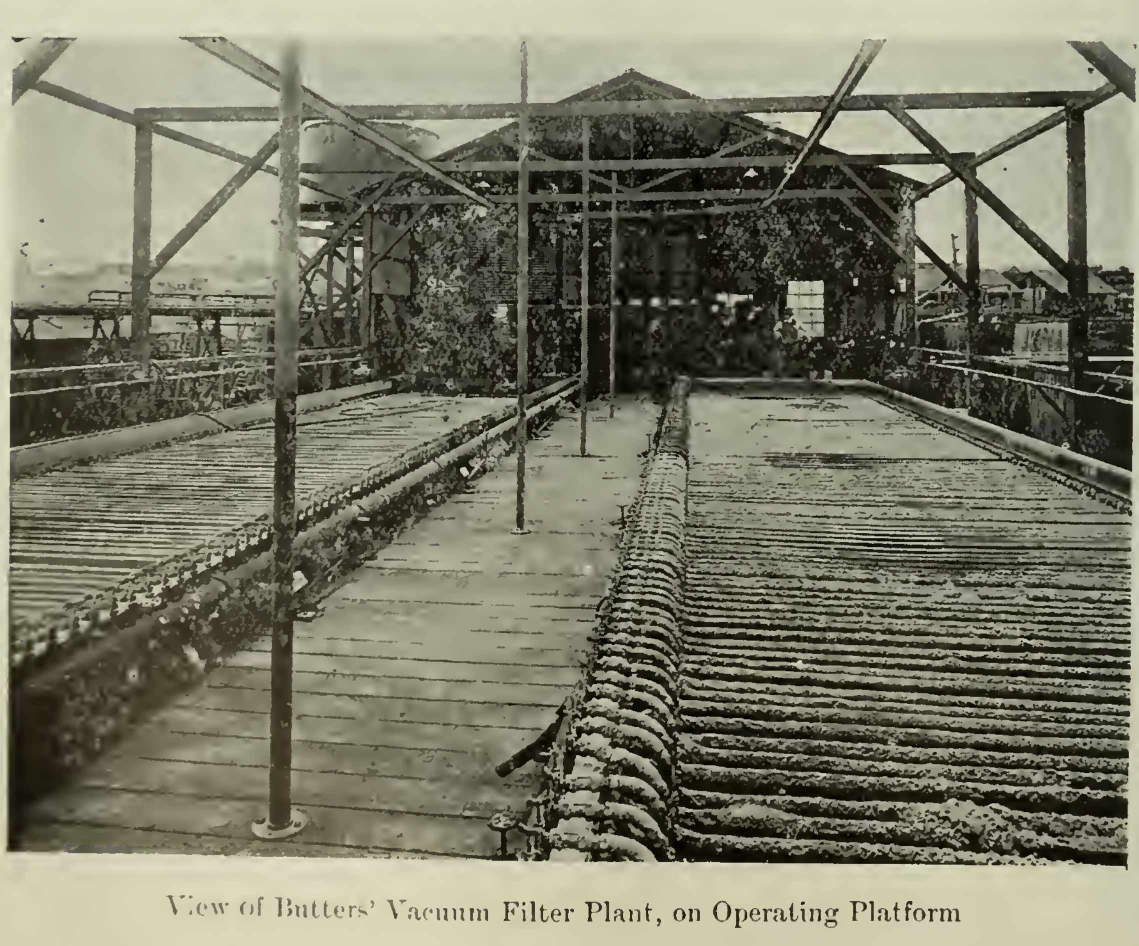
Butters' Vacuum Filter Plant :—We have also to thank Mr. C. G. Patterson for the following description of the Butters' vacuum filter plant as installed and worked on the Crown Reef section of the Crown Mines :—

The plant consists of two stock vats equipped (temporarily) with air agitation to keep the slime in suspension. After the values have been dissolved in Brown agitators, the pulp at a dilution of 2 to 1 is transferred to the stock pulp vats, from where it is fed to the filters as required.

One wash solution vat. All precipitated solution is delivered to this vat to be used subsequently for a wash in the filters. Two Butters' filter boxes each fitted with 150 Butters' patent vacuum filter leaves and one 14 × 14 Gould's vacuum pump. Each filter box is an independent filtering unit. One 10 in. Robeson-Davidson slime pump handles pulp and solution to and from the filter boxes and the stock vats, serving the boxes alternately. The piping is arranged round the pump in a loop, having four valves interposed so as to reverse the direction of flow through the pipes. The pump is set in either the emptying or filling position instantly, by means of one pilot valve introducing hydraulic pressure into the cylinders of the four valves simultaneously. The motor for the 10 in. pump and also valves in the pump systems are operated from the

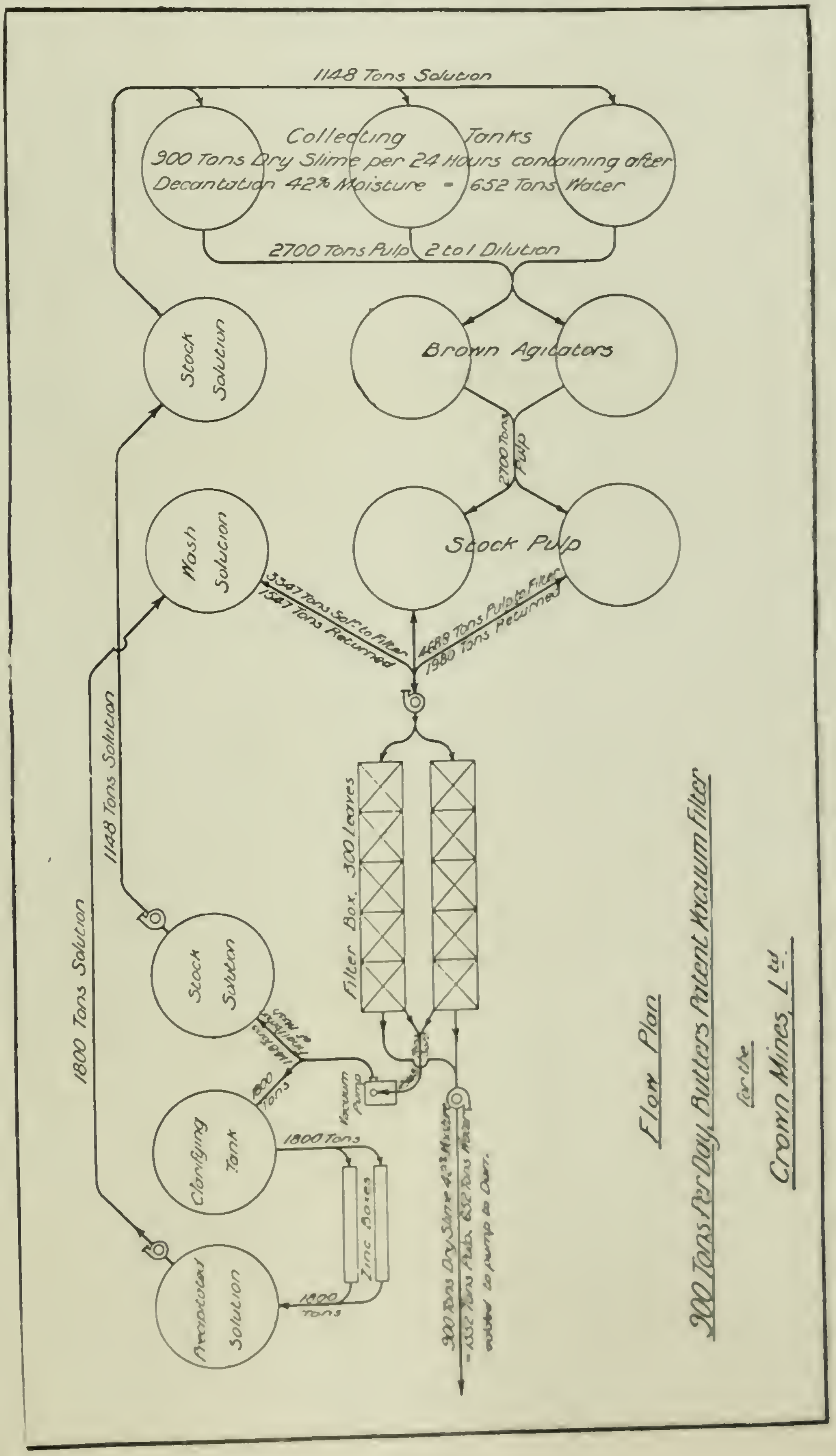


View of Butters' Vacuum Filter Plant, from beneath.



View of Butters' Vacuum Filter Plant, on Operating Platform switch board. The style of plant is called the "pumping system," and the process of filtration is performed as follows: —

Forming the Cake.—The filter box is filled with pulp to a point over the top of the filter leaves, and a valve opened connecting the vacuum pump directly with the filters. Clear cyanide solution is drawn from within the filter leaves and delivered to the clarifying vat for precipitation, the slime remaining as a cake on the outside of the filters. The filters are kept submerged by refilling the box at intervals, and jets



of air are introduced at the points of each hopper to keep the slime in suspension. When a cake 1 in. to 1½ in. thick has been formed, the surplus pulp is pumped back to the stock pulp vat, and the box is then filled with solution to wash the cake. During the time of forming and washing the cake, the vacuum is maintained at the highest possible point, but when the cake is exposed to air during the transfer of pulp and wash solution, the vacuum is reduced to 5 in. to prevent the cake cracking.

Washing.—The cake being formed by atmospheric pressure, the resistance and permeability is equal over the whole surface of the leaf, thus producing an ideal condition for the recovery of the valuable moisture contained therein, which equals 30% to 35%. Sufficient wash solution (about 2 tons per ton of slime) is drawn through the cake by the vacuum pump to effect a complete displacement of the original moisture. A portion of this solution goes to the clarifying vats for precipitation ; the flow is then diverted to a stock vat to be used in making up new charges in the Brown treatment vats.

Discharging.—When the wash is complete, the vacuum is disconnected and a reverse flow of solution (by gravity from the vat on the roof) is introduced to the interior of the filters, causing the cakes to drop. The surplus solution is then pumped from the filter box through decanters and returned to the wash solution vat. The mass of thick sludge remaining in the box is diluted with water and agitated with air for a few minutes to make a homogeneous pulp of 1 to 1, which is then pumped to the residue dam by a second, 10 in. R. & D. pump, the delivery pipe of which is fitted with an automatic sampler.

Cycle of Operations.—

Filling box with pulp and forming cake	45 min.
Transferring and washing	70 „
Discharging	20 „

Total time of cycle 2 hours 15 minutes.

Tons treated per cycle, 1¼ in. cake = 50 tons each box.

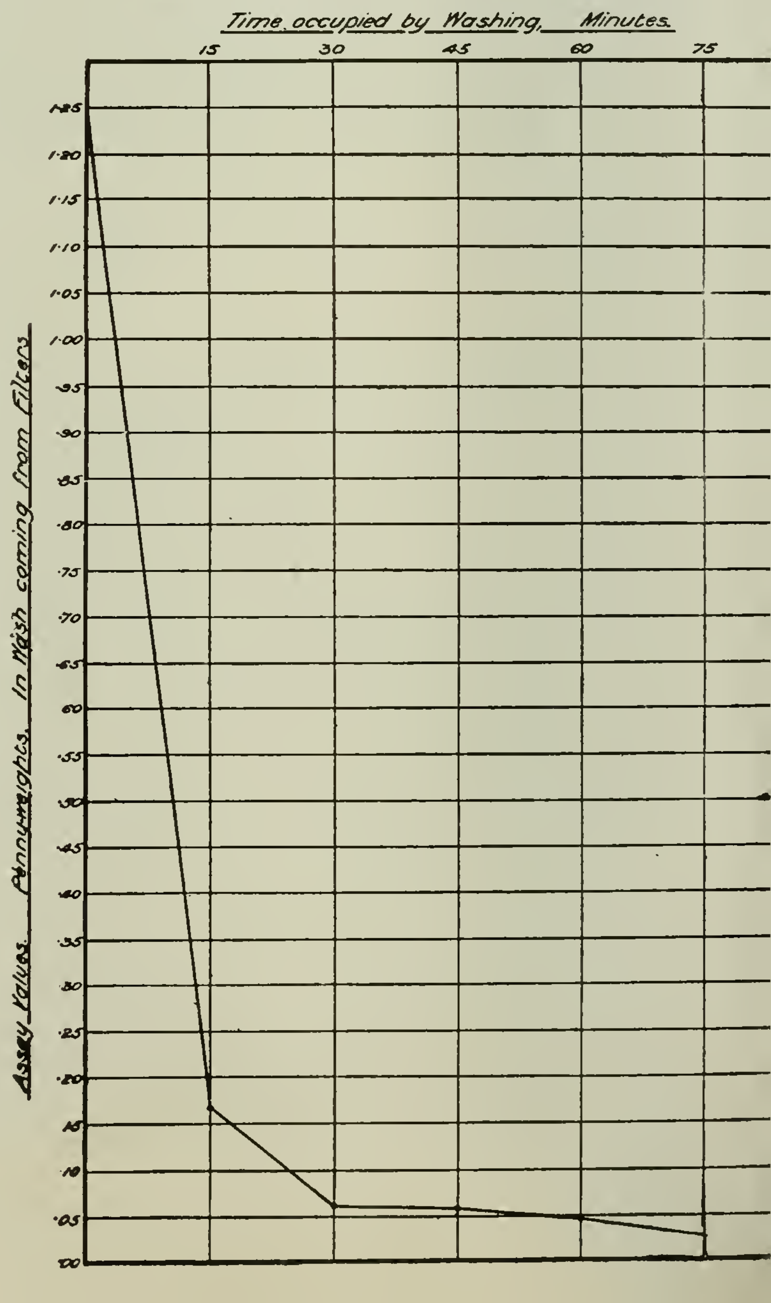
Acid Treatment.— Five leaves are removed from each filter box every day, thoroughly washed with a spray of water and air, and then immersed in a vat of 2% hydrochloric acid solution, for the purpose of dissolving the calcium carbonate and keeping the leaves soft, pliable and in good working condition. The HCl solution is passed through the leaves in reverse directions alternately by means of an automatic arrangement that requires no power or attention. The leaves are removed from the acid bath after four to six hours' treatment, again washed with the spray and returned to the filter box.

Space will not permit of a complete analysis of the six month's test work, as conditions were not normal, owing to insufficient slimes to work the plant to full capacity, and to the treatment of deleterious accumulated slime in combination with the current slime the major portion of the time.

In January a special test was made on current slime only, treated first by the Adair-Usher process, the residues from which were delivered to the filter for the purpose of determining what further recovery of dissolved values was possible by filtration. In this test, as also in the former operations with accumulated slime, the efficiency of the filter was fully demonstrated, assays of the residues showing consistently during the whole period over 99% recovery of dissolved values.

There is some dissolving of gold, incidentally effected, during the process of filtration ; however the Butters' filter is essentially a slime washing machine, purely mechanical in its functions, and might be termed a process of forced decantation.

Washing Curve. Charge No. 24. 18·8·10.



The plant was designed to treat 500 tons per day. Subsequent operations proved, that, owing to the exceptional adaptability of Rand slime to filtration, it has a capacity of 900 tons per day.

The plant was put in commission on August 12th, 1910, and after a most exacting test over a period of six months, has been formally taken over by the Crown Mines, Ltd., as a part of their permanent plant.

Obituary.

The deaths of the following members are recorded with much regret:—

Dr. **Max Marcus Klein**, M.D., Ph.D., who died suddenly on the 16th January. Dr. Klein was well known for the great interest he took in the health and dietary of the natives on the Witwatersrand, and in April, 1905, contributed a paper on "Food, and the Practical Aspect of the same in Compound Work" to the S.A. Association of Engineers. Dr. Klein, who practised at Brakpan and was also medical officer for the Rand Collieries, was elected a member of the Society in March, 1908.

Mr. **Douglas William Greig**, who died on the 10th March from the result of burns received a week earlier while destroying damaged nitro-cotton was one of the technical chemists of the Dynamite Factory at Modderfontein. A lance-corporal in the Imperial Light Horse, he was buried with military honours at the Factory Cemetery on the 11th March. Mr. Greig, who was admitted an associate of the Society in April, 1909, read an interesting contribution to the discussion on Prof. Stanley's paper on the "Smelting of Titaniferous Iron Ores" in January, 1910, and in collaboration with Mr. Wm. Cullen submitted a paper to the Society in September, 1909, on the "Analyses of Gases from Burning Nitro-Glycerine Explosives. His untimely death came as a great shock to his many friends.

Mr. **David Murdoch**, cyanider at the Simmer and Jack Proprietary Mines, Ltd. Mr. Murdoch was elected a member in April, 1909.

Mr. **W. D. Morton**, who was elected a member in July, 1906, was at one time a partner of Mr. Howard Pim, the well-known accountant and auditor of many Rand Mines and De Beers Consolidated Mines. Mr. Morton had been in indifferent health for some years past.

Mr. **C. C. Marshall**, M.I.M.E., one of the oldest associates of the Society, having been admitted in May, 1898. Mr. Marshall, who practised as a mining engineer and surveyor had a varied experience on the Rand, having been employed at various times with the General Mining and Finance Corporation, Ltd., the Van Ryn Estate, West Rand Mines, Geduld Proprietary Mines. On leaving the Rand he spent some years in Rhodesia, where he died.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Chemistry of Coal-dust Explosions.—"The chemical aspect of coal-dust explosions has been carefully studied at the Experimental Station at Liévin, France, and the discussion of the results obtained in this direction forms one of the most important parts of a report on the production of dust explosions recently issued by the Director of the Station. The experiments were conducted in the main test gallery, then 65 mm. long (1 metre being equivalent to ·914 yd.), and in order to collect the gaseous products of combustion a glass flask, exhausted of air, and connected by means of a sealed glass tube to the interior of the gallery, was caused to fill itself immediately the explosion reached that point. This was done by the flame firing a detonator attached to the tube, which was shattered, thus allowing the gases to rush into the flask. To ascertain the composition of the gases entering in this way, the analysis had to be corrected for the amount of air previously existing in the flask owing to incomplete exhaustion. In the accompanying Table I., the actual analyses are set forth in columns 3, 4, and 5, while the corrected figures are given in columns 6, 7, and 8. The figures in the first column refer to the number of the experiment, samples of the gases not being taken in every one of the tests, but only now and again.

An examination of column 7 shows how little free oxygen remained after the passage of the flame; the combustion must, therefore, have been almost complete. In the test No. 43 the residual oxygen attained 8·4%, but it is seen that this experiment was conducted with the lowest dust charge; namely, 112 gm. per cubic metre, a gram being equivalent to ·0353 ounce. Theoretically, this amount of dust is sufficient to absorb the whole of the free oxygen of the air; therefore, the analysis shows that, as 40% of the air remained unchanged, a portion only of the 112 gm. of dust took part in the explosion.

In column 9 is expressed the relation between the volume of carbon monoxide and the total volume of the two oxides of carbon formed in the combustion—that is $\frac{CO}{CO_2 + CO}$. Leaving the results of Nos. 150 and 151 out of account for the present, it is seen that this ratio, though very variable, is low, and that it appears rather larger for the heavier dust charges than for the lighter ones. The fact revealed here could hardly have been anticipated; it is that although an excess of dust may be present at a high temperature, the amount of carbon dioxide formed in the flame greatly predominates over the amount of carbon monoxide. It must, however, be noted that the gases were taken from immediately behind the breast of the flame, and before the red-hot carbon

TABLE I.

Number.	Dust charge in grams per cubic metre of gallery.	Analyses of the gases collected in the flask.			Corrected composition of the gaseous products of combustion.			$\frac{CO}{CO_2+CO}$ (volumetric proportion.)	Relation between the weight of hydrogen burnt and the weight of carbon burnt.
(1)	(2)	CO_2 (3)	O (4)	CO (5)	OO_2 (6)	O (7)	CO (8)	(9)	(10)
		per cent.	per cent.	per cent.	per cent.	per cent.	per cent.	per cent.	
43	112	7·50	11·00	·50	9·00	8·40	·60	6·0	·1043
53	225	10·25	5·00	3·00	12·10	1·20	3·60	23·0	·1196
79	337	10·25	5·75	1·50	10·50	4·60	1·50	13·0	·1391
56	337	9·75	5·75	2·00	11·40	2·10	2·30	17·0	·1463
39	450	9·75	4·25	4·50	11·60	·30	5·40	32·0	·1210
139	450	12·50	3·25	1·50	13·05	1·45	1·60	10·7	·1242
36	450	11·00	3·00	4·00	9·95	1·90	4·10	28·5	·1610
58	450	10·25	5·00	2·25	12·00	1·20	2·60	18·0	·1410
77	450	12·00	2·50	1·50	12·00	1·25	1·50	11·1	·1677
57	450	9·25	5·75	2·00	11·10	1·50	2·40	17·7	·1715
62	450	7·75	7·00	3·25	11·00		4·60	29·5	·1565
63	450	7·50	7·00	3·25	9·90	1·65	4·30	30·2	·1700
108	450	10·00	2·75	3·75	10·60	·48	4·00	27·2	·1772
35	450	8·00	6·50	1·50	9·20	2·90	1·70	15·7	·2394
150	450	5·50	2·00	6·50	5·30	·95	6·30	54·1	·3275
151	450	4·25	2·50	6·50	4·20	1·00	6·40	60·4	·3972
75	610	10·50	5·00	1·25	10·60	3·80	1·30	10·6	·1623
74	610	10·25	2·75	4·50	10·50	1·50	4·60	30·5	·1454
59	900	7·00	6·75	4·25	8·40	3·00	5·10	37·7	·1682

had a chance to re-act on the carbon dioxide and reduce it to the monoxide. That the latter reduction actually takes place, and that the reaction takes a short but appreciable time to complete, was demonstrated by tests Nos. 150 and 151, wherein the gaseous products were obtained from distances, respectively, 10 and 5 meters behind the breast of the flame. A glance at the analyses of the gases taken in these two tests will show that there was an excess of the monoxide over the dioxide of carbon at these distances behind the flame. It is, therefore, apparent that chemical activity continued behind the flame, resulting in the rapid increase in the percentage of carbon monoxide.

How very dangerous is the afterdamp of a dust explosion may be gathered from these last analyses. Not only had the oxygen almost entirely disappeared, but carbon monoxide was produced in such high proportions that if the products had been diluted with a large quantity of fresh air they would still have been dangerous to breathe even for a short time.

In the last column of Table I. is stated the ratio by weight, w, of the hydrogen to the carbon consumed in the combustion. The figures in this column are noteworthy in that they indicate the importance of the part played in the ignition by the volatile constituents of the dust. Before the bearing of these relations can be discussed, it is necessary to explain the manner in which they were calculated.

The quantity of carbon burnt is deducible straightway from the amounts of carbon dioxide and monoxide produced. The hydrogen, however, cannot be computed quite so easily, since it forms watery vapour in burning, which condenses, and hence does not make an appearance in the results of the analyses. The weight of hydrogen has, therefore to be obtained indirectly by calculating the amount of oxygen which has disappeared as water. Now the analyses render account of all the oxygen, free and combined, except that forming water; while the nitrogen formerly existing mixed with oxygen as the air of the gallery, is obtainable by difference. Thus, by using the well-known relation between the volumes of nitrogen and oxygen in normal air, we are able to compute the total original amount of the latter gas, and then, by difference, the amount which, after the combustion, exists in combination with hydrogen as water.

The following gives further particulars of the calculation: Let 100 volumes of the gas in the flask contain a volumes of CO_2, b volumes of oxygen, c volumes of CO, and d volumes of hydrogen and hydrocarbons, the remainder being nitrogen. Among these various gases will be mixed r volumes of air, being the quantity initially present in the flask by reason of imperfect exhaustion. The latter quantity, r, is easily found from the initial pressure in the flask. Making the necessary correction for r, the total volume of oxygen entering the flask, both in the free and combined states, is:

$$O = a + b + \frac{c}{2} - {\cdot}21r;$$

and in the same way the quantity of nitrogen entering is

$$N = 100 - (a + b + c + d) - {\cdot}79r;$$

Of the nitrogen so calculated a small quantity, n, will have been derived from the coal dust; the remainder $(N - n)$ existed originally in the air of the gallery, and accompanying it there must have been $\frac{21}{79}(N - n)$ volumes of oxygen. If k volumes of oxygen were also derived from the coal dust, then, by difference,

$$\tfrac{21}{79}(N - n) + k - O$$

measures the amount of oxygen now in union with hydrogen in the form of water. Therefore, by volume,

$$\frac{\text{Water vapour}}{CO_2+CO}=2\left(\frac{\frac{21}{79}(N-n)+k-O}{a+c}\right)$$

and by weight,

$$\frac{\text{Hydrogen consumed}}{\text{Carbon consumed}}=w=\tfrac{2}{12}\cdot 2\left(\frac{\frac{21}{79}(N-n)+k-O}{a+c}\right)$$

Substituting for N and O their values as obtained above, we may write:

$$w=\frac{8{\cdot}75-{\cdot}421(a+b)-{\cdot}255c}{a+c}-\frac{d}{12(a+c)}+\frac{4k-n}{a+c}$$

This expression includes a principal term—the first one, and two corrections—the last two terms, of which the latter are of little importance. It has been found that d has a value never exceeding unity, and that the smallest value of $(a+c)$ is 12; therefore the term $\frac{d}{12(a+c)}$ is at the most equal to $\frac{1}{144}$; *i.e.*, only 5% of the mean value of w. To evaluate the term $\frac{4k-n}{a+c}$, it is necessary to consider the weight of dust producing the quantities k and n. Let us suppose that 450 gm. per cubic metre of air had been put into effective suspension. This amount will be equivalent to 45 gm. per 100 liters of air, and will very nearly be equivalent to 45 gm. per 100 litres of the gas in the flask. Now 45 gm. of the dust used in these experiments contained 8·5% by weight of oxygen and nitrogen, that is to say, 3·8 gm. of these gases; and these when liberated would occupy 3 litres. Since the residual particles after an explosion still contain a large part of the volatile constituents of the dust, only a portion of the 3 litres of gas will actually be set free. Inasmuch as those quantities of oxygen and nitrogen existing in the proportion in which those gases are found in normal air will be eliminated in the difference $(4k-n)$, the latter term can only represent a fraction of a litre. It is therefore apparent that the last term of the above expression is of even less importance than the middle one. Hence, in calculating the figures in column 10 of Table I. these last two terms were neglected.

The general high value of w is striking and important. In the dust used for the experiments there were only 4 parts by weight of hydrogen to 75 of carbon. If the carbon had completely burned, w would have been about ·05, while as a matter of fact, it always attained at least double that figure.

If the volatile matter were taken to consist entirely of hydrogen—certainly an incorrect assumption—the above results would show that the weight of the volatiles taking part in the combustion would be derived from at least twice the weight of the dust wholly burned; the proportion will, as a fact, exceed this, for besides hydrogen, the volatiles contain carbon in the form of hydrocarbons, and that element goes to swell the denominator of the ratio w.

The mean value of w appears to be increased when heavier dust charges were used, and this might be explained from the fact that the more numerous the particles of dust in a cloud, the greater the total surface from which volatiles may be distilled.

Now, it has already been stated that the gases entering the flask in all the other experiments contained only an insignificant amount of unburnt volatiles; however, on estimating as marsh gas the unburnt volatiles obtained in tests Nos. 150 and 151, they were found to be present in the proportion of 3·8% and 5·2% by volume, respectively. It, therefore, follows that for these tests the last two terms of the formula for w derived above are no longer negligible, and must be taken into account. After the necessary corrections had been applied, it was found that the values of w for the two experiments in question were, respectively, ·301 and ·352. Although these figures are less than those given in column 10, they are nevertheless considerably larger than their equivalents in the other tests where the gases were taken from close behind the flame. The presence of unconsumed volatile matter, and the high value of w obtaining when the gases were taken some distance behind the front of the flame, show that the distillation of the dust continued in that region after the flame had passed. Since there is no more, or nearly no more, free oxygen available to unite with the volatiles so produced, there must remain a surplus of the latter. The increment in w shows, however, that some of the hydrogen of the volatiles liberated after the passage of the flame had united with oxygen, while the proportion of carbon monoxide has also increased; it is, therefore, probable that, in the region behind the flame, reactions such as the following take place:

$$H_2+CO_2=H_2O+CO.$$

Other observations were made on the phenomena of combustion. The first of these was in reference to the amount of smoke produced by an explosion. By raising a shutter, a connection was made after every experiment between the gallery and a fan drift, and the smoke then blown out of the open end of the gallery by the fan, when its colour and other characteristics could be noted. When the dust charge was at least 450 grams per cubic metre of air, the smoke was always thick and black. With lesser charges it was a clear grey, and sometimes almost white.

Dense black smoke consists of an excess of fine dust particles, together with watery vapour which has cooled and condensed, and soot resulting from the combustion of hydrocarbons in an atmosphere almost depleted of oxygen. Grey or white smoke indicates a combustion of the dust which is complete or nearly so, only ash remaining, together with watery vapour in process of condensation. In the last case there existed a sufficiency of oxygen to burn the hydrocarbons without the formation of soot.

In a number of experiments the solid residues in the gallery were systematically collected. They were obtained from the floor and from a dozen transverse shelves each 20 centimetres wide, placed every 2 metres between points in the gallery, respectively, 18 and 40 metres from the origin of the explosion, it being in this portion of the gallery that the bulk of the solid products were deposited. As a rule, little of the residues was found near the source of the explosion, and often none at all in the last few metres of the gallery. This latter fact might be explained through the re-entry of air occurring immediately after an explosion, which would sweep back into the interior any solid matter lying near the mouth of the gallery.

Table II. gives the weight and composition of the residues collected, the weights being stated per cubic metre of the zone from which the products were taken. From what has just been said, it will be apparent that had the products been collected from the full length, and compared with the full volume of the gallery, the results would have been lower than those given below. On the other hand, they would have been higher had the matter from the walls also been included, and also if it had been possible to have added those solid particles ejected from the gallery by the force of the explosion and not drawn in again by the back suction of air.

As would be expected, the weight of residual solid matter increases with the dust charged into the

TABLE II.

Number of Experiment.	Weight of dust per cubic metre of gallery	Weight of solid residues per cubic metre of gallery.	Analyses of solid residues.			Percentage of volatile matter (moisture deducted) in the solid products after the ash has been eliminated.	
			Volatiles including moisture.	Moisture.	Ash.	In detail.	Mean.
	Grams	Grams	per cent	per cent	per cent		
43	112	40	21·15	1·69	22·52	25·0	25·00
53	225	123	21·57	2·42	14·52	22·4	22·40
56	337	210	25·08	1·99	12·45	26·3	26·25
79	337	220	25·09	2·19	12·94	26·2	26·25
36	450	240	25 70	2·20	13·95	27·2	25·30
39	450	227	22·86	2·20	14·54	24·1	25·30
58	450	335	23·37	2·21	13·12	24·6	25·30
75	610	530	26·49	2·19	12·70	27·8	27·80
54	900	670	27·09	1·87	10 52	28·2	27·50
59	900	680	25·98	2·02	10·39	26·8	27·50
Composition of the coal dust used in the experiments			28·00	1·00	9·00	29·6	29·6

TABLE III.

	Volatile Matter Per Cent.	Ash Per Cent.
Coke in grains, or agglomerated	15·3	14·04
	14·4	14·23
	13·4	11·08
	12·3	19·20
	11·4	27·20
Coke in thin flakes from the surface of dust	24·6	11·71
	24 0	11·35
Soot mixed with dust	13·8	19·20
Residual coal-dust	28·1	10·75
	26·3	14·05
	26 2	10·11
	22·4	13·10
	18·9	14·70

gallery ; it is almost proportional to the excess of the charge over about 100 grams per cubic metre, which corresponds to the complete combustion of the carbon of the coal-dust by the oxygen of the air.

The analyses show that the products were poorer in volatiles and richer in ash than the original dust from which they were derived. A reference to the last column of the table also shows that, even after the ash constituent has been eliminated from the products and the original dust, the former is still the poorer in volatiles, thus proving that the reduction in volatiles revealed by the analysis does not result merely from the increase in ash, but that there must have been a partial distillation of hydrogen and hydrocarbons from that part of the dust which was not completely burned. This result is in concordance with the conclusions which have already been drawn from the gas analyses. The solid products were appreciably more humid than the original dust, owing to the condensation of some of the large quantity of watery vapour produced in the explosion. Water was also found condensed on cold portions of the walls of the gallery.

A further examination of Table II. shows that there was generally a more pronounced reduction in the percentage of volatile matter when the dust charge was small. In other words, although the amount of volatile matter per cubic metre is greater with the heavier charges, it does not increase proportionately with the charge, but at a lesser rate. This can be explained partly from the laws of variation of temperature during combustion, and partly from the fact that, with the heavier charges, the dust is not perfectly lifted ; thus there must be a portion of the dust in contact with the floor, which, although included as 'solid residue,' has not been thoroughly subjected to the effects of a high temperature Hence, in reality, Table II. registers the analysis of a mixture of the actual solid products of combustion of the dust lifted, with dust lying immediately against the floor, the composition of which would hardly be altered.

Examination showed that the solid residues usually consisted of three parts; namely, dust whose physical aspect remained unchanged, coke either in a finely divided condition or as grains and soot. These three elements, which were difficult to separate properly, were distributed in unequal proportions along the length of the gallery. Circumstances in that part of the gallery not allowing of the agglomeration of large pieces, the coke forming in grains the size of a pea or of a pin's head, within a few metres of the origin of the explosion, was carried by the rush of air either towards the middle of the gallery or sometimes—especially for the heavier dust charges—nearer the open end. The soot, very fine and greasy to the touch, was deposited on top of the coal-dust and coke. It required some time to settle, and was not seen at all when the fan was started straightway after the explosion.

In certain parts of the gallery a firm thin skin of caked matter was occasionally noticed on the surface of the unconsumed dust, but the conditions under which the skin was produced were not clearly determined.

Table III. gives the analyses of the various solid products of combustion, separated fairly well one from another."—*Mines and Minerals.*—Dec., 1910, p. 264. (A. R.)

METALLURGY.

THE EXAMINATION OF WATER BY ELECTRICAL METHODS.—" The well known physical fact that the electrical conductivity of distilled water is greatly increased by the addition of even minute traces of impurities is the basis of a simple device for testing water for boiler-feed purposes which recently made its appearance in England. One of the inventors, W. Pollard Digby, has described its construction and uses in a communication to the Institution of Electrical Engineers, which is abstracted in *The Electrician* of July 29. Mr. Digby does not suggest that the measurement of conductivity alone will supersede chemical analysis in water testing ; that the electrical method has important supplementary uses, however, is amply evident from an examination of the results quoted in his paper. We are unable to reproduce these results in our review, but we give Mr. Digby's description of the apparatus used in the conductivity test and a few details of its applications.

'That water in its varying degrees of impurity is anything but a medium of constant value comes within the experience of all engineers, and hence frequent resort is made by the engineer to the analytical chemist for advice and assistance. However, this is not always done as often as might be desired, and a variation in the nature of a supply may take place with no visible change in the appearance of the water, but yet with such changes in the constituents as would render the water quite unsuitable for boiler-feed purposes. The author's object is to point out the value of simple tests made *in situ* as safeguarding the owners of boilers from such happenings, and as determining when the analytical chemist should be employed. These tests, however, are not intended in any degree to supersede the analyst, but may serve to determine when detailed analyses are necessary by indicating changes in the impurities in the water. The method resolves itself into a determination of the conductivity of the liquid at any predetermined temperature and a comparison of such determinations with the known values of analysed samples. Any marked variation from the normal should be followed by a further analytical examination.

'An apparatus for conductivity measurements has been devised by Mr. C. W. V. Biggs and the author. The original type consists of a straight glass tube so constructed as to hold the equivalent of a body of liquid 10 cm. in length and 1 square cm. in sectional area, the liquid being contained between platinum electrodes attached to plugs of ebonite fixed in cups which form enlargements of the ends of the tube. The electrodes are suitably connected to terminals in the ebonite ends, and are capable of being adjusted. The liquid to be tested is introduced into the main tube through a small tubulure, rising at an angle from the middle of the upper part; and when tested is run off through a small tube furnished with a tap attached to the middle of the lower part. At each end of the upper surface of the body tube a vent tube is attached, inclining to the middle of the main tube, and the two unite at a height of about 2½ cm. above it, with a short vertical outlet.

'Manifestly such a piece of apparatus is hardly suitable for use outside the laboratory. The form for practical work consists of a glass U tube, to which is attached to the lowest point of the bend the tube brought from the filling funnel, whilst the outlet, controlled by a glass tap, is joined to the inlet at its lowest point. Near the extremities of the limbs of the main tube overflow ways are sealed on, and terminate in india-rubber tubes grouped with lower outlet. The form and arrangement of the electrodes are especially designed to minimise the disturbing effects of polarisation and other difficulties inseparable from the original type of tube. The electrodes are open cylinders of platinum, about 9 mm. in diameter and 3 mm. in height, connected by three equidistant platinum wires to stout brass glass-covered rods passing through the brass covers which are connected to the terminals. The dimensions of tube and electrodes are such that the equivalent of a body of liquid having a length of 10 cm. and a sectional area of 1 sq. cm. is obtained. The apparatus is mounted on a strong wooden support carrying a thermometer, and the whole is contained in a lock-up portable case.

'The most convenient instrument for measuring the conductance is the 'conductance meter' made for use with the tube. It resembles the familiar 'megger,' but is furnished with a scale graduated in reciprocal megohms and ohms, and with a tube of sectional area one-tenth the length it gives readings of one-tenth of the specific conductivity equivalent to the reciprocal of ten times the specific resistance. Correction for variation of temperature for water containing less than 1% of dissolved salts may be made with sufficient accuracy by taking the coefficient as 2·19% per 1° C., as referred to 20° C.; but above 38° C. such correction is not reliable owing to complications arising from disengagement of small quantities of dissolved gases and also from physico-chemical action.

'At the outset the author experienced considerable difficulty in determining standards for comparison. Of samples passing through the author's hands, good distilled water has varied in its specific conductivity from 4·0 to 1·362 reciprocal megohms for laboratory distilled water, and for distilled water from surface condensers the best has been 3·3 reciprocal megohms. As a general standard of comparison, good distilled water might possibly be deemed as possessing a specific conductivity of 3·3 reciprocal megohms, or a specific resistance of 300,000 ohms at about 20° C., this value representing an approximate average value of the output of the Liebig condensers in the laboratory of one of the author's friends, and also representing the best practice from central-station surface condensers that has come under the author's notice.'

The effect of minute traces of impurities in greatly increasing the conductivity of distilled water is a well known physical phenomenon. Mr. Digby gives a number of typical results of his investigations of the effect of minute traces of sodium chloride, sodium carbonate and calcium suphate. The difficulty with the electrical test, of course, lies in the fact that it cannot distinguish between the different impurities. Nevertheless, conductivity tests are of value in detecting variations in the quality of feed water, and in checking the operation of marine evaporators and of evaporators for the supply of distilled water for hydraulic gun mountings. One of their most practical applications is in checking the operation of surface condensers. Given a boiler free from priming, a single conductivity measurement of water passing from the condenser to the hot well will indicate the tightness of the condenser tubes. If the approximate rate of flow is known at any moment, together with the value of the circulating water, a reference to standard curves, examples of which are given by Mr. Digby, will give the percentage of leakage at that moment. Conductivity determinations, while indicating that overhauling is necessary, may save unnecessary examinations. Conductivity tests may also be used to indicate boiler priming. If the conductivity of the hot-well water rises as the load on the engine increases, the only possible deduction is that the boiler is priming. Mr. Digby gives the results of a number of tests of boiler plants, which serve to demonstrate the procedure and the utility of electrical tests for this purpose.

'So far as checking the operation of steam plants by conductivity methods is concerned, their application does not cease with checking the nature of the feed water or with ascertaining condenser tightness, or even with defining boiler priming. A film of oil is more to be dreaded than scale-forming impurities in the management of boilers. An oil eliminator in one form or another is a necessary adjunct in all electricity-supply stations containing reciprocating sets, but their careless operation in a manner other than that designed by the makers may involve the substitution of a less evil for a greater one. The author has good reason to believe that in certain cases an excess of chemicals is added beyond that required for the elimination of the oil. This in cumulative small doses can only result in the increasing impurity of the boiler water through its enrichment with a substance which by common consent is regarded as a source of priming.

'A last point may be mentioned. Where hard waters containing an excessive quantity of calcium and magnesium carbonates are softened by the Clark process, the conductivity tube and conductance metre can be used as a substitute for the standard soap test in defining how far the process of softening has been carried.'"—W. Polhard Digby, *Journal of Institution of Electrical Engineers.—Engineering Magazine*, Sept. 1910, p. 911. [Abstracted from *The Electrician*, July 29, 1910.] (R. A.)

Air-Lift for Raising Sand.—"Most mining men are familiar with the principle of the 'Pohle' air-lift, but there is a common impression that, compared with other methods of raising liquids, it is not economical in respect of power costs. The air-lift under notice was installed under conditions well suited to accurate measurement of the air used and the work done; and it is thought that the following particulars may be useful.

In this case it was necessary to carry the sand residue from the ordinary cyanide leaching vats across a public road, requiring a lift of 23 ft. 3 ins. The submergence was fixed at 63% of the total length of water pipe. To get that with a single lift would have required a pit 39 ft. 6 in. deep, but by adopting a double or two-stage lift, the depth of the pit was reduced to 15 ft. 6 in., saving 24 ft. of sinking, and still maintaining the 63% submergence.

The supply of air is regulated by floats in the receiving head of each lift, so that a nearly constant level is maintained in each head-box and no air is blown through to waste.

The air used was drawn from a large receiver. In testing the air-lift, this receiver was filled to a gauge pressure of 70 lb. per sq. in., and was then shut off from the compressor, and the fall in pressure in a given time noted. During this time no air was drawn from the receiver for any other purpose. Mine water was used for sluicing, and the amount available was only little more than sufficient to carry away (along a V-shaped launder having a fall of ⅞-in. per foot) as much sand as one man could shovel into the discharge door of the vat. The sand is heavy, and rapid motion is necessary throughout to prevent it settling; the proportion of water to sand is about 2½ to 1.

Before the air-lift was put in, it was the practice to shovel the sand through the bottom discharge-doors into trucks; with the air-lift one man empties the vat in considerably less time than two did with the trucks, no time having to be lost in changing and waiting for trucks. The vats hold 30 long tons and, with one man shovelling, they are emptied in three and a-half hours, using 45 cub. to 47 cub. ft. of free air per minute.

Each discharge door has a 4-in. socket screwed into it, and the door is never removed. From the bottom of the socket a pipe leads down vertically, with an elbow just above the launder; inside the vat a pipe is screwed into the top of the socket long enough to reach above the top edge of the vat. To discharge the vat the pipe projecting through the sand is unscrewed and pulled out, leaving a convenient hole through which to start sluicing. To avoid the risk of having more sand sluiced into the launder than the water is able to carry along, from 25% to 30% of the total added water is supplied to the highest end of the launder running under the discharge doors.

Both sections of the lift are of the same dimensions, except in height, and consist of a 6-in. cast-iron pipe (which constitutes the well) with a 3-in. wrought-iron pipe inside, extending from within four inches of the closed bottom end of the 6-in. pipe, up to the required height of lift.

The air pipes are ¾-in., though ½-in. would be ample; they are carried down the pit outside the 6-in. pipes, and are turned up at the bottom through the centre of the dead-end at the bottom of the 6-in. pipes. They extend upwards for 10-in. in the 6-in. and for 6-in in the 3-in. pipes. The nozzle ends of the air pipes are closed and twelve hack-saw slits are cut across each of the ends, the cuts extending round about one quarter of the circumference of the ¾-in. pipe and cut at an angle of 45 deg., thus giving the air an upward direction.

The lift has worked well from the start and (except on occasions when the supply of water has suddenly failed) it has caused no trouble or delay. When the water supply fails a ½-in. water pipe (which is kept connected by flexible hose to a supply having a pressure of about 50 lb. per sq. in.) is passed down the 6-in pipe which is cleared almost as fast as the ½-in. pipe can be lowered. Two or three minutes is sufficient to clear it at any time.

The only wearing parts are the 3-in. pipes, and these, having no bends, are likely to last a long time. The consumption of air may be taken as about 50% more than the best pump would use when in perfect condition. Unlike sand pumps the air-lift remains in perfect condition.

The added water is heated to a temperature 95% F. by passing a part of it over exhaust steam pipes on its way to the lift. This heat, no doubt, adds to the efficiency of the lift.

Dimensions.

	No. 1 Lift.	No. 2 Lift.
Well pipe	6 in. C.I.	6 in. C.I.
Well pipe length	13 ft. 6 in.	22 ft. 6 in.
Water column	3 in. W.I.	3 in. W.I.
Water column length ...	22 ft. 2 in.	38 ft. 0 in.
Air pipe	¾-in.	¾-in.
Depth of water	15 ft. 6 in.	24 ft. 6 in.
Lift	9 ft. 0 in.	14 ft. 3 in.

Work Done by the Two Lifts Combined.

Cub. ft. free air per min.	Lb. sand per min.	Lb. water added per min.	Lb. sand per cub. foot free air.	Total lb. sand and water per cub. feet free air.	Cub. feet free air per gallon (10 lb.)	Cub. feet free air per ft. gallon.
47	320	800	6·8	23·83	0·4	0·0182

—A. R. Mill.—*Monthly Journal of Chamber of Mines of W. Australia*,—Nov., 1910, p. 367. (H. A. W.)

Resinita.—"This new substance, which appears to offer some analogies to Celluloid and to present a wide field for industrial application, is obtained from phenol and formaldehyde by adding catalytic reagents. The result, a yellow liquid, which the inventor has named 'Resol,' can be used for the impregnation of woods, paper, or other porous material, which it renders impermeable. From the distillation of the resol is produced 'Resit,' whose properties depend on the rapidity and temperature of the distillation; normally it is hard, but acquires plasticity on elevating the temperature; if it is submitted for a certain time to 80^0, and the temperature is afterwards raised to 200°, the Resit proper is obtained. This is solid, infusible, insoluble, and resists both acids and alkalis. Its colour is red, ruby, or purple, unless ammoniacal salts have been used, when the colour is yellow. It is transparent or translucent, very brilliant, with a conchoidal fracture. By sacrificing transparency its elasticity can be augmented by the addition of talc, fecula, &c. In its diverse states Resinita has characteristics of value, on account of the similarity to glass, celluloid, ebonite, and vegetable ivory. This substance is analogous in preparation, properties and applications to Bakelite, an American invention, and a staple product of the United States. The empirical formula is $C_{43}H_{38}O_7$."—(Reprinted from "Revista Minera."—*The Mining Journal*,—Jan. 14, 1911, p. 41. (R. A.)

Properties of Tungsten and Molybdenum.—"Tungsten is known as a grey brilliant metal, hard and brittle. It is used chiefly in the manufacture of special steels, and for the filaments of lamps on account of its high point of fusion. Commercial tungsten is a grey powder obtained from the reduction of yellow oxide, and is usually impure; when employed for the fabrication of filaments it is refined. The pure metal is still brittle and lacks ductility, though in the laboratory of the General Electric Company ductile tungsen has been recently produced, with attributes which differ from those of ordinary tungsten. Ductile tungsten is brilliant and possesses the colour of steel, and can be drawn into very fine wires—can be even attenuated to 1-50th of a millimetre. The power of resistance augments with the stretching; thus a wire of 0·125 mm. in diameter opposes a force of 322 to 343 kilogrammes per square mm., whilst one of 0·03 gives 406 to 427. Ductile molybdenum wires, in the first case, resist with a power of 150 to 182 kilogrammes per square mm., and with a diameter of 0·038 gives a resistance of 189 to 217. The density of both metals increases with the slenderness of the thread Before drawing, the density of tungsten is 18·81, but after, it grows to 19·20, and even to 20·19, according to the diameter. Molybdenum originally 10·02, acquires, after drawing, a density of 10·04 to 10·32. The hardness of tungsten and molybdenum varies with the mechanical treatment to which either has been subjected; in its hardest form either scratches glass, whilst the softer kinds can be easily cut."—(Reprinted from "Revista Minera").—*The Mining Journal*,—Jan. 14, 1911, p. 42. (R A.)

MINING.

The Final Report of the South Africa Mining Regulation Commission.—"Good ventilation is not merely a legally recognized necessity for the preservation of health, but has an important economic aspect in its effect on the industrial efficiency of the workman and the cost of mineral production.

The necessity for ventilation in metalliferous mines arises mainly from the injury to health caused by the presence in the mine air of fine dust, of the poisonous gases produced in blasting, and of human emanations, respiratory and other.

The ventilation of mines may be (*a*) natural, and (*b*) artificial or mechanical; natural ventilation depends upon (1) the difference in temperature and humidity, and consequently in weight, between the air in and out of the mine; (2) the absolute humidity of the outside air; and (3) the difference of level between the mouths of the connected shafts; natural ventilation has the advantage of being inexpensive, but is subject to the following drawbacks, viz., (1) the amount of air supplied depends upon conditions beyond control, and independent of the varying mine requirements and hygienic necessities; (2) as the temperature and humidity of the outside air approximate to that of the mine air, the ventilating current diminishes and may completely disappear; (3) the direction of the air-current is variable.

Natural ventilation is supplemented on the Witwatersrand by the exhaust air from machine drills; this supply is of great local importance, as it is derived at the working place, but the amount only constitutes a fraction of what is required for the proper ventilation of a mine, and its beneficial effects do not extend for more than 100 ft. from the face; some miners are prejudiced against compressor air, though there is no evidence that it is injurious to health; it stirs up from the sides and floor of the working place the dust which is the principal cause of miners' phthisis, and occasionally contains some of the poisonous gas CO derived from the accidental combustion of the oil used for lubrication.

Judged by the relative frequency of gassing accidents, natural ventilation is, on the whole, not so effective on the Witwatersrand as on some other fields, this inefficiency being probably due to (1) the very slow rise of the rock temperature as depth increases; (2) the high temperature and humidity of the outside air at certain seasons; (3) the great extent of the underground workings and large number of men at work; and (4) the large consumption of explosives in blasting, and consequent production of great quantities of poisonous gases

Artificial or mechanical ventilation is effected by means of either (*a*) the extractive force of a furnace at the bottom of the upcast shaft; or (*b*) fans—usually exhaust fans—at the top of the upcast shaft; at the East Rand Proprietary and Cinderella Deep Mines the total running cost of an extremely effective installation is under 1d. per ton milled.

The degree of ventilation in a mine may be measured either by (*a*) a quantity standard, that is, the quantity of pure air entering the mine per minute; or by (*b*) a quality standard, this being a determination of the amount of impurity present; the existing Transvaal law provides for a quantity standard of 70 cubic ft. per man per minute, and also for the splitting of the current and distribution of the air over the working faces; the application of this standard is open to very serious practical difficulties; the provision as to distribution is not enforced, probably because attempts to direct natural air-currents usually merely retard them, and the quality standard is better adapted as a general indication to mine inspectors than as a basis for legislation.

The object of a quality standard is to fix the permissible amount of air vitiation, and for this purpose the quantity of carbon dioxide present is accepted as bearing a roughly constant proportion to the amount of impurity present.

Our knowledge of the exact nature of such impurity is incomplete; there is no reliable evidence to support the former belief that during respiration the lungs exhale a volatile organic poison; the immediate ill-effects, headache, general discomfort, etc., of bad ventilation are probably the results of the oppressive smell which arises from the breath, bodies, and clothes of those present, and is caused by very minute quantities of volatile substances present in the air.

According to Pettenkofler (1858), Dr. J. S. Haldane, F R.S., and other modern observers, this smell becomes distinctly perceptible when the carbon dioxide CO_2 in the air exceeds 10 volumes per 10,000; but in 1875, De Chaumont recognized it in barrack-rooms when the CO_2 reached 6 volumes per 10,000.

Continued subjection to bad ventilation means increased liability to disease, and especially to infection produced by organisms present in the mouth and air passages, and conveyed directly through the air from person to person.

Dr. Haldane, F.R.S., who has closely investigated the subject and is the foremost British authority thereon, considers that the worst consequences of a defective air supply are 'the evil effects produced by inhaling poisonous or infective dust'; he doubts that 'constant exposure to volatile respiratory impurities has by itself a very great influence on health,' and agrees that carbon dioxide is 'the best objective criterion of the sufficiency of ventilation.'

In mines, the highly poisonous gases, carbon monoxide CO and nitric oxide NO_2 are also present, the result of the detonation or of the burning of explosives; in dead ends, and immediately after blasting, the ratio of CO_2 to CO in the gases produced by explosives in local use averages 1 to 12, and therefore CO_2 serves in some degree as an index of this danger (presence of CO).

Carbon dioxide, unaccompanied by any injurious substances, is in itself innocuous in quantities under 1 per cent. Of this nature are the normal atmospheric CO_2, amounting to 4 volumes in 10,000 of air, the CO_2 produced by open lights and by the action of acids on carbonates.

The difficulty in fixing a quality standard lies in the impossibitity of distinguishing between the noxious and innocuous CO_2 when both are present together.

On the Witwatersrand the only innocuous CO_2 known to exist in appreciable quantities is the normal atmospheric CO_2, and that produced by open lights.

Exhaustive inquiries made by the Mines Department show that a considerable body of carbonates which could give rise to CO_2 by the action of acid water, is present in the workings of one mine only, the remainder having either no carbonates at all, or only inappreciable quantities in the shape of flakes in fault planes, or as an occasional stringer adjoining a dike; in view of the minute percentage of acid in mine water, there would generally be in these places a sufficient excess of water to hold in solution the CO_2 generated, and as mine air is nearly saturated with moisture, it is improbable that any dissolved CO_2 would subsequently be released by evaporation.

In the Lydenburg and other districts, and in base metal mines, masses of carbonates occasionally exist in proximity to very pyritic reefs, and a considerable amount of innocuous CO_2 may be locally produced.

It was suggested to us that additional quantities of CO_2 might be produced from the following sources, viz , (1) 'ground' gas contained in rock cavities, or occluded in quartz; (2) from the oxidation of food and timber, evaporation of stagnant water, and decomposition of animal waste.

We find that cavities containing CO_2 are practically unknown in the Witwatersrand metalliferous mines; that the occluded gases would not be liberated under the conditions of underground work, and are in any case noxious, containing a large percentage of the poisonous CO; that the oxidation of timber and food would produce quantities of CO_2 too small to affect any standard of vitiation, and that the gases from stagnant water and animal waste are inappreciable and offensive or noxious in charcter.

Careful experiment was made at the Langlaagte Deep mine, under the supervision of the Mines Department, to ascertain whether any production of "ground" CO_2 could be inferred from the difference between the estimated amounts of CO_2 entering and leaving the mine; the possibilities of error in the assumptions upon which the estimates were based appear so considerable, and the results calculated on different assumptions so widely discordant, that we regard the various Langlaagte Deep results as inconclusive, and in this view we are supported by the Government Mining Engineer.

In the absence of any theoretical source of any appreciable quantity of 'ground' CO_2 in the Witwatersrand metalliferous mines, we consider that practically the whole of the noxious CO_2 is due to respiration and explosives, including fuse, etc., and that the innocuous CO_2 is derived from the atmosphere and open lights only.

The limits for noxious CO_2 previously recommended or embodied in legislation are as follows:

(*a*) Five volumes per 10,000, by the Roscoe Committee (1896) on the Ventilation of Humidified Factories. This was legalized under the Cotton Cloth Factory Act, and also in the Factory and Workshops Act, 1901.

(*b*) Eight volumes per 10,000, by Haldane's Committee (1902) on Ventilation of Factories and Workshops. This standard now regulates all factories, including humidified, in the United Kingdom.

(*c*) Eight volumes per 10,000, by West Australian Mines Regulations Act.

(*d*) Six volumes per 10,000, by New South Wales Mines Act.

(*e*) Eight volumes per 10,000, by Victoria Mines Act, 1907. Because we cannot definitely state that there is no material quantity of 'ground' CO_2 in the mines of the Rand, though we believe that the actual amount, if any, is small, we propose that a working allowance of 5 parts in 10,000 be made for it, and for CO_2 from other uncertain possible sources, with the sole object of fixing a standard which is practicable from the administrative point of view, and which will enlist the voluntary co-operation of the mines in its enforcement; and, in view of our recommendations (see below) as to a CO limit, and as to sectional and local ventilation, the prevention of dust and fumes, the total limit of 20 parts of CO_2 per 10,000 is well within the limits of safety, is reasonable and easily obtainable, and should be enforced.

We, therefore recommend as follows (see Draft Regulations 56—63):

GENERAL VENTILATION.

(*a*) That the legal maximum for noxious CO_2 permissible in mines in the Transvaal be fixed at 8 parts by volume in 10,000 of air.

(*b*) That an amount of 4 parts of CO_2 by volume in 10,000 of air shall be allowed in addition to the aforesaid maximum as representing innocuous CO_2 normally present in the atmosphere.

(*c*) That where candles or similar illuminants are used, a further addition of 3 parts of CO_2 by volume in 10,000 of air shall be allowed as representing innocuous CO_2 resulting from the combustion of such illuminants.

(*d*) That in order to meet, from the point of view of practical administration, difficulties in regard to possible innocuous CO_2 from 'country rock' and other uncertain sources in the mines of the Rand, a further allowance of 5 parts per 10,000 be made, making a total limit of 20 parts of CO_2 per 10,000 of air.

(*e*) That in the Lydenburg and other districts where there is geologically strong presumptive evidence of a production of ground CO_2, early investigation be undertaken by the Government, and that a proper and reasonable allowance be made therefor, the total amount in the mine air not to exceed 1% by volume.

(*f*) That all samples for testing purposes under these provisions be taken not less than one hour after blasting.

(*g*) That each mine be informed of the results of any official analysis of the air therefrom, and notified that the ventilation is defective when the above proportions have been exceeded, and at the same time supplied, so far as practicable, with information as to the nature of any defect noticed: and that legal proceedings be not taken against a mine unless, after a reasonable interval following such notice, the stated proportion is found on examination of one or more samples to be again exceeded, and the mine is unable to show that steps have been taken reasonably calculated, in the opinion of the Government Mining Engineer, to secure the requisite ventilation.

(*h*) That any analysis on which a prosecution immediately depends shall be made by a specially qualified person.

(*i*) That arrangements be made for inspectors of mines to have the use, when desired, of a properly tested portable apparatus for estimating on the spot the proportion of CO_2 in the air.

With regard to the very interesting question as to the necessity for correcting the proposed CO_2 limit for the altitude of the Rand, the evidence is somewhat conflicting, and many samples will be taken at considerable depths and, therefore, we do not recommend any addition on this account to the proposed standard.

In view of the extremely poisonous effects of CO and NO_2 on the human system, and the frequency of gassing fatalities on the Rand, we also recommend that the maximum permissible amount of CO in any part of a mine shall not exceed ·01%, and no practically determinable amount of NO_2 shall be permitted in any part of a mine." – *Mines and Minerals*, Jan., 1911, p. 337. (A. R.)

Explosions in Coal Mines. – The recent disaster in the Lancashire coalfield has called forth numerous comments from correspondents, and attention may be directed to some remarks which seem most nearly to indicate the probable cause of the explosion, and to suggest a definite line of action for their prevention. Mr Jacob Atherton objects to the present method of working a coal mine by a single down cast and a single up-cast shaft, and he shows that this leads to accumulations of gas in the old workings, and that a spark from a pick striking flint, damage to a lamp, or the firing of a shot may ignite this gas in such circumstances that the men at the working face and about the levels are hopelessly trapped. As a remedy, he urges that mine-owners should work upon the principle of driving levels—down and up brows—from the pit's mouth to the boundaries of their properties, thus bringing the face of the coal towards the pit shaft, instead of the present system of driving the face away from the shaft. Levels, down-brows and up-brows, all with airways, should "be driven through the solid coal, and thus assure free exit from the working face of the mine from all directions along these solid main roads." In support of his contention he cites the case of a Lancashire colliery worked on this principle, "with the result that miners worked with naked lights, such as candles and torches. No explosions took place, the ventilation was perfect, the lighting was good, and the miners were perfectly happy under such conditions." As compensation for initial cost, he suggests the "saving of timber, day wagemen, and constant repairing of levels and airways." Mr. J. Marriner arrives at the same general conclusion that the number of exits is insufficient. He advocates that the coal-dust trouble should be investigated by the Government, and he expresses wonder that the Government "refused to contribute towards the cost of experiments." He further holds, that it would be quite reasonable to insist that miners should be allowed to work only where two exits are available, and he advises that these exits should enter the workings at opposite ends of the mine: moreover, they should pass through solid ground or be secured so that they will not fall in at the time they are most required. – *Times Engineering Supplement*, Jan. 4, 1911. (J. M.)

Concrete for Shaft Tracks. – "The extended use of reinforced concrete on the Rand is a noteworthy feature at the present time, and especially is this true of new equipments. Concrete now seems to be utilized wherever practicable. Its extensive employment in the metallurgical works of the City Deep plant has already been dealt with, and we are now able to give particulars of the use it is being put to at the Bantjes Consolidated Mines at Florida.

At this property it is being utilized in a new direction: new as far as the Rand is concerned, at any rate, with the exception of some similar work done at the Angelo Deep two or three years ago. But as this Angelo Deep work does not appear to have attracted much attention, the employment of concrete in two shafts of the Bantjes mine is a most interesting matter. The central incline has been stripped and enlarged, and the shaft is now 32 ft. 4 in. wide all over, instead of 17 ft., as formerly. Five compartments are now prepared, and there will be four hauling ways instead of two only in the old shaft. Concrete blocks are now being carried down the shaft, which dips at an angle of 35°. The skip rails run on these, and the gauge has been increased from 3 ft. to 4 ft. 2 in. The central concrete blocks, which carry two rails, are 2 ft. 9 in. in width at the top, whilst the outside blocks, on each of which only one rail rests, are 12 in. wide at the top. Only one divider is being put in between the two pairs of skip roads. Sixty-pound rails are used, and strong steel supports are employed in the shaft, which will contain very little timber: in fact, no timber at all. The shaft is indeed an excellent piece of work, and it should be possible to haul in it at the rate of a mile a minute with a minimum risk.

The advantages claimed for this method of supporting skip rails are: (1) Higher rate of hoisting: (2) greater safety: (3) a cleaner shaft: (4) much more permanent work: (5) less wear and tear on the rope. The cost is estimated at less than that of a wooden road, and the work of laying the concrete proceeds very rapidly. At a little depth in the shaft, where solid rock exists, the concrete is mixed on the floor of the shaft and put in the skip road direct." – *Mining Science*, Sept. 18, 1910. (K. L. G.)

The Care of Miners' Eyes. – "It is really astonishing that the eye pulled this way and that by the muscles every day for years and years, and sub jected to all kinds of blows and rude shocks, should keep its beautifully delicate organisation throughout. Any other instrument such as an opera glass or microscope if submitted to the thousandth part of the rough usage that the eye has to put up with would be ruined past all repair, and yet the eye remains useful all the time. It is at once at autochrome camera, a cinematograph producing life-sized coloured pictures in motion; a photometer, a stereoscope, a range finder, a microscope, and an opera glass; while the whole apparatus is provided with a compound non-distorting rectilinear lens working at F/4, or in some cases at F/3, and provided with a self-adjusted iris diaphragm. Helmholtz could hardly have borne these facts in mind when he stated that, had an optician made him an instrument as imperfect as the eye, he would have returned it to him. From a large number of experiments made by the writer, he has found that when shooting at a target in dull light, or when the target was painted khaki colour, plain spectacles, made of a light shade of spectrum blue glass, enables him to see the mark much better. This material has the peculiar property of cutting off both the irritating violet rays as well as the red ones On the other hand, if the target be a black and white one, the marksman can see it better through a pale amber in a pale pinkish orange yellow glass. The glass used by Lumiere for taking photographs in colour answers admirably, but the so-called Euphos glass of Schanz does fairly well.

People talk about their sight being strong or weak, and they often boast that they can see to read without spectacles up to the age of 60, or even 70 years. Now, this is a silly thing to boast about. It is well-known to occulists that the normal healthy eye which can see distant objects quite well, always requires spectacles on attaining the 45th year. The lens in the eye is controlled by a little circular muscle which surrounds the eye inside it. The lens is made to alter its shape by means of this little muscle, so that one can see both distant and near objects at will. The nearer the object the more the muscle has to act, so that when the object, such as a newspaper, is held about 14 inches from the eyes, quite a considerable effort is required on the part of the muscle to bring the words into sharp focus. Now, during childhood, the lens is very elastic, and the muscle can adjust its form quite easily, but as the child becomes an adult, the lens becomes much more rigid, so that it gets year by year more difficult to see the newspaper at comfortable reading distance. At length, a pair of spectacles has to be procured. But after a few years the person finds that he must either hold his paper at arm's length or move his glasses to the tip of his nose, by which he makes them of stronger power. This constant strain causes the eyes to water, and in this way acts injuriously. It is wise, therefore, for anyone when he gets to 40 or 45, to be properly fitted with glasses. People who are naturally a little short-sighted can read quite well when they get to adult age without glasses, but those people who are naturally long-sighted need them even for seeing distant objects. Such people should always wear them for reading whatever age they are. By so doing they greatly rest the eyes, and prevent their becoming tired and misty.

The eye is a spherical ball about the size of a plum (*i.e*, an inch in diameter), having behind a round, thick stalk—the optic nerve—which conveys the impressions of the images seen to the brain. Many people imagine that dust or dirt can work its way to the back of eye and set up terrible inflammation, but this is quite a mistake. The eye is covered with a kind of skin known as the conjunctiva, which is quite transparent where it covers the clear part of the eye; if traced backwards it forms a kind of cul-de-sac all around the eye about half-way back, and is then deflected (*i.e.*, folded back) on itself to line the under surface of each lid. Thus the eye is quite shut in and no dirt can get behind, When dust or particles of grit or steel fly into the eye they are arrested by the conjunctiva, and, if sharp, they may stick into it. The most painful cases are those where they are lodged either on the cornea (*i.e.*, the clear part of the eye in front), or underneath the upper lid. These are the commonest places for dust particles to lodge, and everyone should know how to remove them, as they are exceedingly painful, and often a doctor is not at hand to remove them.

If the particle is lodged on the front of the eye the best way to remove it is to let fall a drop or two of sweet oil right on to the front of the eye with a feather, and then to brush the foreign body off either with the soft end of the feather, or a small camel's hair paint brush. If there is a chemist near, get him to put a drop or two of cocaine solution in—it cannot possibly do any harm—and the sufferer will not feel the brush, and consequently will keep quite still while it is being removed. If you can see nothing, and the eye is very painful, look very carefully to see whether the 'cornea' is scratched by the particle. If it is, a drop of cocaine, followed by a drop of sweet oil—or castor oil—will give the necessary relief. If, however, nothing can be seen, and the eye is very irritable, the probability is that the piece of grit or steel has lodged under the upper lid. First, get the sufferer to catch hold of a few of the lashes of the upper lid, and tell him to pull down the lid, looking down at the same time. Two or three attempts will probably dislodge the particle, and the tears will wash it away. If that is no good, let him sit down on a chair, wash your fingers, and procure a wooden match. Now stand behind him and tell him to look down. Catch hold of a few of the eyelashes with the left hand finger and thumb, and press the match, holding it horizontally, downwards and backwards with the right hand against the outside of the lid, and at the same time pull the lid up. This will turn the lid inside out, and in all probability you will see the piece of grit resting on the red shiny surface of the inside of the lid. Then remove it with a brush or the wood end of the match, which latter you can take away while firmly holding the lid back. A drop of cocaine or oil on the surface will at once put matters right, and the patient will suffer no longer. It is a wise thing for miners to keep a 1% solution of cocaine in a bottle in a handy spot—as a drop in the eye can never do any harm, and it helps one to remove both the grit and the pain in a marvellously short time. It is very important to remove the foreign body as soon as possible, as it invariably provokes inflammation, and every hour makes the poor sufferer worse, and if neglected, may bring about a troublesome disease which may confine him to bed for days.

Miners are very careless about dust. When working in the mine, the dust should always be kept under by water, both by squirting it into the drill holes, and sprinkling the galleries, adits, and other passages. Dust not only affects the eyes and brings about all sorts of troubles there, but it gets into the lungs, and causes a form of consumption known as fibroid phthisis, which sometimes ends fatally, and always causes severe coughing fits, and even pleurisy in bad cases.

Managers of mines are often greatly to blame for giving the miners insufficient illumination. The galleries cannot be too well lighted, as the wretched candles and lamps usually provided tend to tax the eyesight severely. The miner often has to work in a very confined position, lying on his side or back, or crouching down with his head and eyes turned obliquely round. This produces an undue strain on certain muscles and, together with the bad light, causes a tremor of the eyes, known as nystagmus. This tremor, if continued for a long time, becomes at last chronic and automatic, so that the miner habitually moves his eyes rapidly from side to side, even when he leaves the mine. The bad light is chiefly responsible for this, because the eye trembles in its endeavours to see badly lit objects. Were the light better in quality and better distributed, by having plenty of lamps scattered about, this nystagmus would not occur.—GEORGE LINDSAY JOHNSON, M.D., F.R.C.S.—*The Australian Mining Standard*, Jan. 25, p. 84—5, 1911. (W. A. C.)

GEOLOGY OF KOLAR GOLD FIELD.—"Practically the whole of the province of Mysore consists of a plateau about 3,000 ft. above sea level. It is quite undulating on the whole, but with the exception of the granite or gneiss peaks scattered about, the country is not hilly. With the exception of the region mineralized with gold-bearing quartz, the whole country is of a foliated granitoid rock which the geologists insist upon calling gneiss. It is uniformly gray in colour, varying in crystallization from coarse to fine, and from light to dark in tint. The most striking feature of the landscape is the gneiss hills, which are scattered about without any regularity whatever in their arrangement. In size these vary from a few boulders in elevated places to single haystack-like cones more than a thousand feet high, some forming miniature mountain ranges 10 miles long and 1,500 ft. above the surrounding country at the highest points. Much of this gneiss disintegrates rather rapidly, leaving a lot of sandy, barren soil. This rock furnishes the building stone of this whole region, being readily obtained in pieces from 4 in. to 1 foot thick and of any length and width that can be handled, making an ideal stone for steps, copings, and the like.

The laterite drift is the most puzzling phenomenon presented in all this region, its very obscurity adding interest to it.

The whole surface of the country, with the exception of the gneiss hills and 'Goldfield Ridge,' is covered with laterite. Indeed, so nearly universal is it that the theory of its covering the entire surface up to certain altitudes at the time it was laid down seems quite correct. How far to the north the laterite extends beyond the sea north of Madras the writer does not know, but between Kolar and Madras it covers many thousand square miles. Near Madras the laterite is used for buildings, railway stations, and bridge piers, but that about Kolar is quite unfit for building purposes. Where found in large masses it is quite amygdaloidal in structure, the cells in it being from the size of beans to that of small marbles. The laterite occurs massive on the top of certain hills, which are uniformly flat, and on one hill 15 miles from Kolar, it is about 20 feet thick: whereas, all over the country—scattered everywhere—is the fragmentary laterite, nearly decomposed, giving the peculiar iron colour, dark red or maroon, to the soil to a depth from 2 ft. to 4 ft., the portions remaining being little shot-like balls, either round or kidney shaped.

The state geologists have not discovered the origin of laterite, nor a place where it is found to great depth: so the theory is that it was spread over the country under water at some very remote period, coming from 'parts unknown.'

The hornblende schist formation extends roughly for 30 miles north and south, and varies in width from 1 to 4 miles. The Kolar gold field is approximately at the south end of the formation. The rock has a greenish colour and its ability to withstand the weather varies everywhere, with the result that ridges of the rock are seen protruding above the surface in some places, while in others it has been completely decomposed into a clayey soil running through all gradations and combinations of colour from light green and light yellow to nearly white—and this to a depth of from 6 ft. to 12 ft. It is common in digging to pass through the iron stain from the laterite at the surface for a depth of 2 ft. or 3 ft., and then come to the unstained clay of the decomposed country rock.

In this schist, running nearly north and south, are many quartz veins of various widths from a few inches to 10 ft. and even more. These quartz veins carry free and almost pure gold: for example, at the Balaghat mines gold is about 97% pure as it comes from the mill. Many of the quartz veins are barren, although the quartz to all appearances is the same in one vein as another. The quartz is either white or gray, both carrying some or no gold as the case may be.

The quartz veins which showed gold at the surface were worked out by the natives a long time ago. There are no records extant of these old workings, but tradition indicates that the work ceased with the invasion of the Mohammedans under Arungzeeb in the latter part of the seventeenth century. Be that as it may, in several places, both at Kolar and in other fields of South India, the natives did work out all the surface quartz down to depths varying from 60 to 300 feet. The present gold mines are working on the same veins at much greater depths.

The natives seem to have prospected this field thoroughly in the old days, for, although there are many quartz outcrops, none of them carry any gold, from which it is quite clear that whatever gold was at the surface was found and removed. So great is the confidence of the present-day mining engineers in the thoroughnesss of the work of the ancients, that there has been no systematic prospecting of these fields, except at or near old workings. No doubt some day in places where there are barren quartz outcrops with placer gold in the ground all about, there will be serious effort made to find out what exists at depth. There are many such places.

The Goldfield Ridge is one long and nearly straight upheaval bounding this last formation on the west. It is the most interesting feature to the student of geology of anything in this region. The combinations are mostly iron and quartzite, but, for variety in lamination, crumpling, and folding, as well as variety in colours and texture, it would be difficult to duplicate. It seems certain that the 'ridge' is a comparatively recent upheaval. There is no laterite on it, although but half a mile from it is a hill 50 feet higher whose top is covered with laterite, from which it may be concluded that if the ridge had been in existence at the time of the laterite drift it would also have been covered with laterite.

Where the rocks of the ridge are in place the stratification is practically vertical, but there has been so much twisting and squeezing that all the forms of lamination read about in geologies are exemplified in countless places. In height the ridge is 50 ft. to 300 ft. above the surrounding country.

It should be added that there are masses and streaks of quartz (white) in the ridge, which in some places look suggestive of gold. In some places these bodies of quartz have been prospected but no gold of importance found, and the quartz itself suddenly terminated or gradually pinched out.

The Mines.—There are six producing mines in the Kolar field, all of which have been working with various vicissitudes for nearly 20 years. Another half dozen or so which were worked in the past are now idle. In some cases the idle mines had at one time good ore, which was worked out, while in other cases they used up their capital in prospecting.

All these mines are on the two principal veins, which have various short branches extending here and there from them. The principal veins are not entirely continuous either in quartz or in gold, the latter being mostly in 'ore shoots' of varying vertical depths and lengths, all leading to the north along the veins.

The dip of the veins is about 30 degrees from the vertical, west, so that most of the mines have both incline and vertical shafts.

Only one mine has sufficient water for its own needs. One does no pumping whatever—at all events it is called a perfectly dry mine, although down 300 ft.

The ore is free milling and each mine does its own crushing, amalgamating, and cyaniding The process of ore treatment is identical at all the mines—stamping, amalgamating, and treating the tailing with cyanide solution. Details in treatment vary with the ideas of the superintendents rather than because of any essential difference in the ore. Some, for example, practice both battery and table amalgamation, and others only table. The slime is rather abundant and gives considerable trouble.

There is no opportunity to take advantage of gravity in arranging the milling plants, owing to the contours of the ground which furnishes no side hills, consequently the tailing has to be elevated to considerable height, the larger mines having tailing dumps as much as 50 ft. high—often higher, and covering several acres. Mine labour is cheap; men, 9 cents a day; women and boys, 6 or 7 cents, above ground. So the tailing especially is handled by manual labour.

Any description of this field would be incomplete without mention of the great Mysore Mine, which began milling operations in 1884, with total amount of ore crushed of 454 tons, from which 454 ozs. of gold was recovered. Two years later the mine paid $150,000 in dividends. The next year the dividends fell to $75,000, but since that have gone on increasing until the annual dividends have reached as high as $2,000,000.

Addenda.—At the thirtieth annual meeting of the Mysore Gold Mining Co., held in London, England, in March, 1910, the following information was given out. The Mysore government re-leased the property to the company for 30 years at an additional royalty of 2½% of the output; however, the government reduced the price of electrical power from £24 to £10 per horsepower per annum, which probably offsets the increased royalty. The working costs, including royalty and income tax, absorb 33% of the total value of the ore. 'The mine employs 10,000 men, women, and children, 3,000 of them miners, and the medical staff looks after 90,000 souls.' 'Whenever cholera breaks out, the healthiest place in the whole of India is Mysore camp.' The ore reserve exceeds 1,000,000 tons, and the ore treated amounted to 234,500 tons, from which was recovered 228,249 ozs. of gold, or 18½ dwt. of fine gold per ton. During the year, 1,053 ft. of shafts and 754 ft. of winzes were put down. The deepest point in the mine is 4,175 ft. on the dip of the deposit. The mine has distributed since its commencement in 1880, £5,935,094 in dividends to shareholders."—C. S. DURAND.—*Mines and Minerals*, Jan., 1911, p. 350. (A. R.)

HYDRAULIC FILLING IN GERMANY.—"At the International Congress of Mining, Metallurgy, Applied Mechanics, and Practical Geology, which began at Dusseldorf on the 19th ult., and to which invitations were accepted by some 1,762 persons in all parts of the world, a large number of papers were read. Among them was one by Otto Putz, mining engineer, of Tarnowitz, Upper Silesia, on 'The Present Position of the Hydraulic Packing Process in Germany.' Owing to its interest locally the following summary is given of it:—

"The hydraulic system of goaf packing, first employed at the Myslowitz Colliery, Upper Silesia, in the middle of 1901, is now used in 145 sets of plant at about 110 collieries. This relatively small extension of the process is due, not to any inferiority in the process itself, but more especially to the heavy wear on the pipes, the difficulty of clarifying the water (or of pumping the unclarified water), and the lack of a sufficient supply of packing material. The annual wear and tear on the pipes in the hydraulic packing plant in Germany is estimated at about £50,000. This item can be reduced, on the one hand by using sand and loam, or by crushing to small size other forms of packing material, and on the other hand by providing the pipes with efficient lining. The Stephan type of oval pipe, with iron lining, and also the porcelain-lined pipes appear to have the longest working life. In the case of the oval pipes, the maximum wear encountered hitherto amounts to 1 mm. ($\frac{1}{25}$th in.) for every 10,000 cub. metres (13,000 cub. yards) of packing material conveyed, as compared with 1 to 4 mm. in the case of porcelain-lined pipes. However, it is not yet possible to form a final opinion on this point. The flushing water is clarified either in the goaf itself or outside; and in the latter case the water is either in motion or perfectly stagnant. Up to the present considerable difficulty and heavy expense have attended the removal of the deposited sediment from the settling tanks, where this form of clarification has been adopted; but the position is expected to be improved by the pneumatic system of transporting sludge introduced by Schubert, of Beuthen (Upper Silesia). In this process the sediment is transferred, intermittently, from the settling tanks into a receptacle, either by natural differences in pressure or by suction. When this receptacle is full, the contents are ejected by compressed air and forced through a pipe to their desired destination. In this manner 700 ft. to 1,800 cub. feet of sediment can be transported per hour, at a cost of about 3d. per cubic yard (as compared with 3s. for hand labour) for a vertical height of 160 feet, and a longitudinal distance of 1,300 yards. In most cases, in order to secure a permanent supply of packing material in abundance, it is necessary to resort to long-distance carriage; and as the State Railways are out of the question for this purpose, owing to their high rates and congested traffic, it is advisable for collieries to associate for the construction and operation of railway tracks, a venture that is more profitable in proportion as the volume of traffic is increased, and therefore as the number of participators is larger. An association of this kind has already been formed in Upper Silesia, with offices at Gleiwitz, and has purchased agricultural land outside the actual industrial district and intends to construct a line about 16 miles in length. In many cases the construction and operation of ropeways will be cheaper. From the foregoing it will be evident that the difficulties still militating against the further rapid spread of the hydraulic packing process will soon be removed, a circumstance that is highly desirable for reasons of national economy, since it is estimated that, by means of this process, about £7,500,000 worth of mineral treasures that would otherwise be inaccessible, can be recovered from the earth in Germany.'"—*S.A. Mining Journal*, July 23, 1910. (C. B. S.)

RAND MINING IMPROVEMENTS.—"In a speech delivered at the annual meeting of the Glen Deep, Ltd., Mr. R. W. Schumacher, the chairman, gave an interesting summary of the improvements being effected in underground methods upon the Rand. Reference to his remarks will provide us with an opportunity for reviewing the principal tendencies of change that have characterized recent progress. In the first place he dealt with the bolder methods of laying out underground workings and the saving of money and labour resulting from the adoption of longer stope backs, permanent tramming levels and mechanical haulage. We have not yet heard of the mechanical haulage system being installed upon the Rand, but underground workings are certainly being planned with a view to its introduction. Preparations therefore must necessarily include longer backs. 'Our methods of breaking rock can in many cases,' said Mr. Schumacher, 'be improved by paying more attention to the natural lines of weakness.' This remark no doubt has reference to the value of 'fracture planes' discussed by Musson Thomas and others before the Chemical, Metallurgical & Mining Society of S. A. In some mines, or some stopes, miners may admittedly be at fault by failing to take full advantage of the rock's breaking qualities, but, judging by the best opinion current on the Rand, the 'fracture plane' fetish is unlikely to lead to any valuable reforms. The benefits of narrow stoping—that is to say excluding as much of the external waste rock as possible—were emphasized in the speech. These, of course, have always been held in view by Rand managements though commonly lost under the baneful influence of the indiscriminating call for low costs, by careless mining.

Another point raised was the possibility of introducing more extensively labor-saving appliances for the transportation of ore from the stope faces to the main tramming levels. When the flatter 'reefs' in the eastern portion of the Rand become more widely exploited and the scarcity of labour is more acute, the 'advantage' of methods replacing the native shoveller will be little more apparent than the 'necessity.' How this may best be done remains to be determined under different working conditions. A good deal has been written of the air winch operating in the Jumpers Deep, serving a track down the middle of the stope with side tracks, on to which the trucks are switched. The claims made for it so far seem to be little more than optimistic estimates, and there are other schemes which may yet achieve the desired end more economically.

Mr. Schumacher foresaw a greater concentration of work than at present in vogue. In many mines to-day an excessive number of levels are contributing ore. This certainly leads to waste, especially in supervision and standing charges. At the same time, it appears to us that concentration must almost automatically follow the previously mentioned mechanical haulage and increased backs, dealing with big tonnages. 'Much has been done during the last few years to improve the general condition of the native' and, indeed, much remains to be done. The information we have gathered from Rand engineers, while it does not turn us to the belief that the Kafirs are ill-treated or subjected unwisely to extreme discipline, certainly has convinced us that the popularity of the field with the natives can be appreciably increased by devoting greater attention to their comfort and feeding. The pay is ample for their requirements and the work not too onerous.

Finally, Mr. Schumacher touched upon ventilation, assuring the shareholders that the importance of the subject is fully recognized. Artificial ventilation has been introduced upon two mines of the Rand and it is to be employed in other properties in the immediate future."—*The Mining World*, November 27, 1909. (C. B. S.)

THE USE OF FERRO-CONCRETE IN MINES.—"The rock pressure liberated by mining operations cannot be estimated in advance with sufficient accuracy, and is especially subject to frequent changes, so that the miner has to deal with strains of unknown extent and direction acting on the timbering of the mine. In addition to compression, a bending strain also comes primarily into action, so that the interior of the timbering material is subjected to tension strains as well as those of compression. Now brickwork and ordinary rammed concrete are mainly capable of standing pressure only and fail when powerful tension strains come into operation. Ferro-concrete, on the other hand, is able also to withstand a certain amount of tension without destruction of coherence. The advantageous conditions of strength exhibited by ferro-concrete rest upon the systematic combination of two building materials, each supplementing the other in their static properties—namely, concrete, which has a high compression strength, and iron, which has a high tensile strength. Owing to its high power of resisting rock pressure, ferro-concrete seems, therefore, specially adapted to replace brickwork and rammed concrete as a lining for mine galleries. Further advantages of ferro-concrete for this purpose are—resistance to the influence of pit air and moisture, impermeability to water, proof against fire, low frictional resistance to the passage of the ventilating current and excellent adaptability to all requisite shapes, combined with compactness of construction. Its defects are—difficulty of working on the finished lining and difficulty of preparation, this latter being the chief, the nature of mining operations being adverse to the planning and execution of constructions in ferro-concrete. On the one hand, the rock pressure cannot be calculated, and therefore the character and strength of the ferro-concrete structures must be based on estimates alone. Then, owing to the narrowness of the spaces, the imperfect lighting and the great difficulties in the way of supervision, a great deal has to be left to the reliability of the men employed for carrying out the work. Failures in ferro-concrete work in mines are almost always due to defects in execution. The chief reason why ferro-concrete has not been very widely employed in lining pit galleries is probably owing to the high requirements exacted with regard to the expert knowledge and reliability of the technical superintendent and workmen. In any event, the existing ferro-concrete linings in shafts, cross-cuts, engine chambers and so forth justify the conclusion that, given proper execution of the work, ferro-concrete lining in general entirely fulfils the expectations formed with regard to its capabilities."—B. VIEBIG. —*The Colliery Guardian*, July 8, 1910, p. 67. (C. B. S.)

SOURCES OF RAND GOLD.—"In a discussion of vulcanism and differential pressure in ore deposition, the author says in Economic Geology: 'Regarding the origin of the gold in the Rand banket, I believe that the stream of occluded moisture in the magma which formed the dikes was the solvent which deposited the gold and the iron sulphide in the reefs. After the extrusive stage had ceased and the mobility of the magma due to occluded water vapour was insufficient to permit further movement of the mass of the dike, it solidified from the top downward and acted as a plug which prevented the solutions from going through to the surface. The solutions could then seek an outlet through the porous banket and deposit their dissolved minerals by reduction of heat and pressure. Associated hydrocarbons could then deposit graphite in the reef and also precipitate metallic gold from its solutions. The silica in the solutions would be deposited among the pebbles and cement the whole into one compact and impervious mass. By selective action different parts of the reef would thus serve as the channels at different times, and owing to variations in the gold contents of the solutions the reef would be made of different degrees of richness.

The cementation due to the silica contents of the solutions would block up the passage in different parts and force the passage of the solutions in a new channel. Naturally, the bottom reef would serve as the channel of circulation for the greatest length of time because of the dike solidifying from the surface downward, and that would be the last one to be shut off.

The reason, I believe, why the mineralization is not interrupted by the dikes is that it was contemporaneous and emanated from them, and I think that it will be found that in regions where the reef is not intersected by dikes, there is no gold. The same is true of the Calumet conglomerate, which is known to occur for a distance of 30 miles or more and only contains copper within two or three miles of that distance, where it is associated with eruptives from which the solutions were derived.'"—HIRAM W. HIXON.—*Mining World*, Dec. 31, 1910, p. 1244. (W. A. C.)

MISCELLANEOUS.

RADIUM IN PITCHBLENDE.—"Since radium has been found to be always a transformation product from uranium, it is natural to look for it only in those ores which contain uranium. The proportion of radium to the amount of uranium in one ore is so definite that Rutherford and Boltwood have determined the numerical relation. They found that there is ·0000038 gm. of radium to 1 gm. of uranium. Thus 1 ton (2,000 lb.) would contain ·0034 gm. of radium for every percentage of uranium present. Or 1 ton of 60% uranium ore will contain two-tenths of a gram of radium, which is equivalent to ·35 gm. of pure radium bromide. Because of this extremely small portion of radium, it would require an ore with a considerable percentage of uranium to be worth treating for the extraction of the radium. This is why pitchblende has been used in preference to other minerals. Although there are a number of other minerals in which uranium is found, pitchblende has the largest percentage."—L. F. MILLER. —*Mining Science*, July 14, 1910. (K. L. G.)

AIR LIFT PUMPING.—"It is still a popular fancy that air in lifting fluids from depths acts in a great measure on an ejector principle and all sorts of nozzles and cones are designed to take advantage of this supposed action of compressed air, but it is all much simpler than that, and the basis of the lift action of air lies in the fact that the discharge pipe contains a mixture of air and water which weighs less than the continuous water discharge pipe; consequently the heavier surrounding water pushes the enclosed lighter mixture upward causing the phenomena known as 'air lift pumping.'

Most pumping experiments by this method lie within the limits of 125 ft., consequently the commercial tables, curves and data in general have been calculated for such conditions, and these will be helpful in consideration of deep well pumping.

In discussing air lifts certain general terms are used and must be understood. By lift is meant distance from the surface of the liquid being pumped to the point of its discharge. By submergence is meant the depth of the discharge pipe under the surface of the liquid being pumped. By percentage of submergence is meant the ratio of the length of the submerged portion of the pipe to the total length of the discharge pipe. The total length of the discharge pipe will of course be the lift plus the submergence. For example, if the surface of the water be 100 ft. below the point of discharge this would be called a lift of 100 ft.; if the discharge pipe extends below the surface of the water 150 ft. it would be called submergence of 150 ft., the total length of the discharge pipe would then be 250 ft. and the submergence would be called 60%. If there be a given ascertained percentage of submergence the actual submergence may be ascertained by multiplying the lift by the percentage of submergence and dividing this product by 100 minus the percentage of submergence, expressed as follows:

$$\text{Submergence} = \frac{\text{Lift} \times \text{per cent. of submergence}}{100\% - \text{per cent. of submergence}}$$

Thus in the above example,

$$\frac{100 \times 60}{100 - 60} \text{ or } \frac{6{,}000}{40} = 150 \text{ submergence}$$

From the beginning of air lift experience it was assumed that the most economical condition for air lift pumping was when the submergence was 60%, but recent developments have somewhat shaken this idea and it is doubtful whether there has been enough of accumulated data collected on this subject to make any definite statements.

The first formula used for determining the amount of free air required to do pumping, assuming 60% submergence, was as follows:

$$\text{Quantity of free air required} = \frac{\text{Gallons and lift}}{125}$$

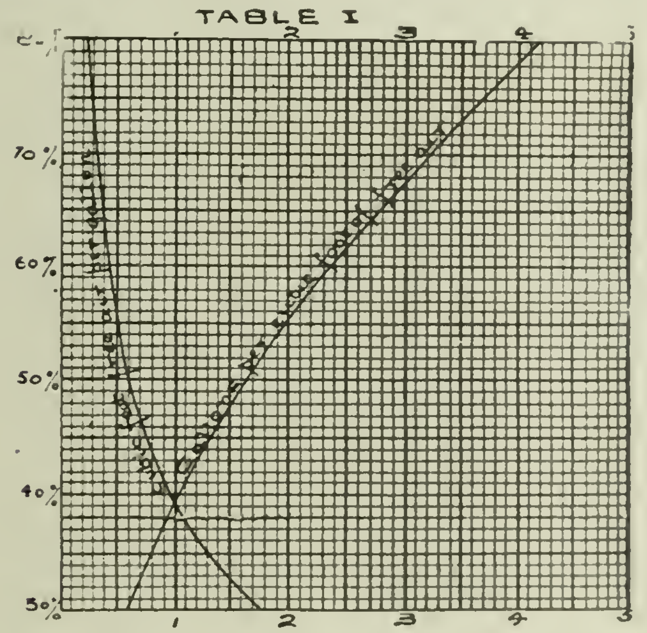
This is a rough rule which still holds good for small lifts, up to 100 ft., but it is too generous for deeper pumping. For example, if it be desired to know the amount of free air required to pump 100 gallons per minute 125 ft. high the result would be:

$$\text{Quantity} = \frac{125}{100 \times 125} = 100 \text{ cub. ft.}$$

of free air, and the pressure required would always be measured by the submergence; thus in the above problem the lift being 125 ft. and the submergence 60%, the submergence would be 1½ times the lift or 187 ft., and the working pressure would be that due to 187 ft. Strictly speaking this would be about 80 lb., but inasmuch as there is pipe friction to be considered it is safe to take this pressure in pounds equal to one-half the submergence, thus one-half of 187 is 93½ lb., which would be a safe working pressure for such conditions.

When compressed air is introduced in the well in a finely divided state so that the bubbles are small and evenly distributed throughout most of the water the best results are produced. It is evident that if the air pipe merely discharges the air into the water with the full opening of the pipe the result will be large bubbles instead of the finely divided condition which is desired. This has led to the construction of many different patterns of what are called 'pump heads,' which is another name for the extremity of the compressed air pipe fashioned in such a manner as to distribute the air to the best advantage to the water being pumped.

The Indiana Air Lift Co. issues an interesting diagram, which I have marked Table No. I.,



and the table of dimensions which I have marked Table No. II. These may be considered fairly accurate at the lifts from 10 ft. to 125 ft

TABLE II.—Capabilities of Air Lift Pumps.

Air Pipe.	Size Pump.	70 per cent.	65 per cent.	60 per cent.	50 per cent.	40 per cent.	33 per cent.	Size of Well.
⅜	1	15	10	10	9	8	6	2½
½	1¼	30	15	15	13	10	8	3
¾	1½	40	25	25	22	15	10	3¼
¾	2	75	55	45	40	30	20	4
1	2½	100	80	75	65	50	30	4½
1	3	200	120	110	95	70	50	5⅝
1	3½	225	150	150	125	90	70	6
1	4	350	200	200	170	125	90	6
1¼	4½	450	275	250	210	170	120	7
1¼	5	550	350	320	260	200	150	8
1½	6	800	500	450	380	300	200	9
1½	7	1200	750	650	550	425	300	10
2	8	1700	1000	800	675	525	400	11
2	9	2000	1250	1050	900	700	500	13
2	10	2500	1600	1300	1100	900	600	14

With these two tables it is easy to determine all the elements of an ordinary air lift installation for pumping water. Having determined the percentage of submergence by dividing the submergence by the submergence and lift—let us say it was 50%, look along the left hand edge of Table No. I. and find 50% and follow along the right until it intersects the second curve and you will have the value 1·65 which means that at that submergence 1 cub. ft. of free air will lift 1·65 gallon of water.

Table No. II. gives the proper sizes of the Indiana pump heads and the proper sizes of water and air pipes for any given condition. Following up the previous problem of 50% submergence, if it be wished to deliver 125 gallons of water per minute the 50% column should be followed down to 125 gallons, then on the second column will be found the size of the Indiana pump head, namely, 3½ in., which is also the size of the discharge pipe. On the first column will be found the size of the air pipe, 1 in., and on the extreme right hand column will be found the smallest size well that will contain the outfit; namely, 6 in.

It will be noted that no mention is made of the lift, because the table being intended for ordinary conditions of 125 ft. or less, it has been assumed that it takes the same number of cubic feet of free air to lift 10 gallons with 60 ft. submergence as 10 gallons lifted 80 ft. with 120 ft. submergence, the working pressure only changing, in the former case being 20 lb. and the latter 40 lb. Now this assumption is not exactly true, but within the practical limits of these lifts it is near enough to be a good convenient rule. When more accuracy is required for greater depths Table No. III., calculated by George H. Reichard, is valuable, as it takes into consideration the expansion of the air bubbles on their way from the lower depths to the surface.

The reason these expansions must be taken into consideration is evident from the very nature of the action of the air lift. Inasmuch as the action of the air lift depends upon an emulsion of air and water, which mixture is lighter than water, it is evident that a perfect condition would be where the bubbles, when introduced at the bottom of the well would maintain the same size in their passage to the discharge. It will readily be seen, however, that inasmuch as the pressure is relieved from the air bubbles as they rise toward the surface, the bubbles get larger and larger, the proportion of air to water

TABLE IV.—Brake Horse Power to Compress 10 cub. ft. Free Air per Minute.

Gauge Pressure.	Brake h.p.	Gauge Pressure	Brake h.p.	
5	·235	130	2·14	
10	·435	140	2·23	Two Stage.
15	·606	150	2·31	
20	·756	200	2·60	
25	·9	250	2·85	
30	1·02	300	3·07	
40	1·25	350	3·26	
50	1·45	400	3·40	
60	1·60	450	3·54	
70	1·77	500	3·68	
80	1·92	600	4·00	
90	2·05	700	3·85	Three Stage.
100	2·18	800	4·00	
110	1·98	900	4·16	
120	2·07	1,000	4·32	

TABLE III.—Approximate Cubic Feet of Free Air and Working Pressure Required to Raise One Gallon of Water by Air Lift.

Formula = $\text{Log.}\frac{H+34}{34} \times 234$

H = Submergence in feet
L = Lift in feet

Ratio of Submergence to Lift.

Lift in feet.	25% $\frac{1}{3}$ to 1		30% $\frac{1}{2}$ to 1		43% $\frac{3}{4}$ to 1		50% 1 to 1		55% $1\frac{1}{4}$ to 1		60% $1\frac{1}{2}$ to 1		66% 2 to 1		70% $2\frac{1}{2}$ to 1		75% 3 to 1	
	Free air cub.ft.	Working press.	Free air cubic feet.	Working press.	Free air cubic feet.	Working press.	Free air cubic feet.	Working press.	Free air cubic feet.	Working press.	Free air cubic feet.	Working press.	Free air cubic feet.	Working press.	Free air cubic feet.	Working press.	Free air cub.ft.	Working press.
20							·428	9	·365	11	·310	$13\frac{1}{2}$	·252	18	·217	$22\frac{1}{2}$	·195	27
30							·470	$13\frac{1}{2}$	·400	17	·350	20	·290	27	·255	34	·230	$40\frac{1}{2}$
40							·508	18	·435	$22\frac{1}{2}$	·387	27	·325	36	·287	45	·263	54
50							·546	$22\frac{1}{2}$	·470	28	·422	34	·360	45	·320	56	·285	$67\frac{1}{2}$
60							·582	27	·510	34	·457	$40\frac{1}{2}$	·392	54	·350	$67\frac{1}{2}$	·323	81
80							·653	36	-575	45	·522	54	·455	72	·410	90	·380	108
100					·850	34	·720	45	·640	56	·585	$67\frac{1}{2}$	·512	90	·465	$112\frac{1}{2}$	·433	135
120					·915	$40\frac{1}{2}$	·785	54	·703	$67\frac{1}{2}$	·642	81	·567	108	·520	135	·483	162
140					·982	47	·847	63	·763	79	·697	$94\frac{1}{2}$	·622	126	·572	$157\frac{1}{2}$	·533	189
160					1·047	54	.907	72	·820	90	·755	108	·675	144	·624	180	·583	216
180					1·107	61	·965	81	·875	101	·810	$121\frac{1}{2}$	·725	162	672	202	·630	243
200	1·8	30	1·436	45	1·167	$67\frac{1}{2}$	1·022	90	·930	$112\frac{1}{2}$	·862	135	·775	180	·720	225	·675	270
250	1·96	38	1·592	56	1·312	84	1·156	$112\frac{1}{2}$	1·069	141	·988	168	·895	225	·830	282		
300	2·12	45	1·750	$67\frac{1}{2}$	1·507	100	1·292	135	1·206	169	1·110	200	1·010	270	942	338		
350	2·28	53	1·897	79	1·635	118	1·449	$157\frac{1}{2}$	1·317	197	1·232	236	1·110	315	1·050	394		
400	2·45	60	2·045	90	1·725	135	1·542	180	1·429	225	1·345	270	1·232	360	1·155	450		
450	2·60	68	2·182	100	1·850	152	1·669	205	1·542	253	1·457	304	1·337	410	1·280	506		
500	2·74	75	2·328	$112\frac{1}{2}$	1·952	169	1·790	225	1·657	281	1·570	338	1·442	450	1·356	562		
550	2·88	83	2·455	128	2·105	$185\frac{1}{2}$	1·907	$247\frac{1}{2}$	1·772	309	1·678	271	1·564	495				
600	3·02	90	2·564	135	2·225	205	2·018	270	1·884	$337\frac{1}{2}$	1·784	410	1·642	540				
650	3·16	98	2·730	146	2·345	219	2·175	$292\frac{1}{2}$	2·012	$365\frac{1}{2}$	1·908	438	1·742	585				
700	3·31	105	2·845	$157\frac{1}{2}$	2·460	236	2·258	315	2·100	394	2·000	472	1·842	630				
750	3·45	113	2·970	169	2·576	256	2·353	$337\frac{1}{2}$	2·225	424	2·080	512	1·935	675				
800	3 60	120	3·095	180	2·690	270	2·465	360	2·320	450	2·190	540	2·063	720				
850	3·72	128	3·215	191	2·800	287	2·570	$382\frac{1}{2}$	2·410	478	2 29	574	2·125	765				
900	3·85	135	3·338	$202\frac{1}{2}$	2·915	304	2·675	405	2·50	506	2·325	608	2·220	810				
950	3·98	143	3·455	214	2·0	322	2·780	$427\frac{1}{2}$	2·610	535	2·485	644	2·310	854				
1000	4·11	150	3·575	225	3·465	$337\frac{1}{2}$	2·885	450	2·710	$562\frac{1}{2}$	2 580	675	2·407	900				

increases exactly in proportion to the expansion and this decreases the efficiency of the lift.

The quantity of air given in this table, No. III., is $2\frac{1}{2}$ times the theoretical quantity required to do the work. Two and one-half has been selected as a co-efficient in this matter as a result of experience. Some engineers have advocated the use of 3 and even $3\frac{1}{2}$ as a co-efficient, but I believe that the table as given to be approximately correct. In the first line of the table the percentage of submergence is given; also the ratio of the submergence to the lift. After having determined the amount of free air necessary to do the pumping and the pressure required, then by reference to Table No. IV. the brake horse-power to compress the air may be determined. This table shows the actual horse-power necessary to compress 10 cub. ft. of free air per minute to the pressure mentioned. An allowance is made in this table for friction and other losses of power, and is generous enough to allow an ample amount of power to do the work."—E. A. RIX.—*Mining and Scientific Press*, Oct. 15, 1910, p. 505. (W. R. D.)

CEMENT CONCRETE VATS AND TANKS.—"Impervious, odourless, tasteless, and sanitary vats and tanks can be constructed of reinforced concrete, the reinforcing to be designed by a competent engineer, provided the interior surfaces are treated as follows:

After the forms are removed, grind off with a carborundum stone any projections due to the concrete seeping through the joints between the boards. Keep the surface damp for two weeks from the placing of the concrete. Wash the surface thoroughly and allow to dry. Mix up a solution of one part water-glass (sodium silicate) 40° B., with four to six parts water, total five to seven parts, according to the density of the concrete surface treated. The denser the surface the weaker should be the solution.

Apply the water-glass solution with a brush. After four hours and within 24 hours, wash off the surface with clear water. Again allow the surface to dry. When dry apply another coat of the water-glass solution. After four hours and within 24 hours, again wash off the surface with clear water and allow to dry. Repeat this process for three or four coats, which should be sufficient to close up all the pores.

The water-glass (sodium silicate) which has penetrated the pores has come in contact with the alkalies in the cement and concrete and formed into an insoluble hard material, causing the surface to become very hard to a depth of $\frac{1}{8}$ to $\frac{1}{2}$ in., according to the density of the concrete. The excess sodium silicate which has remained on the surface, not having come in contact with the alkalies, is soluble, therefore, easily washed off with water. The reason for washing off the surface between each coat and allowing the surface to dry, is to obtain a more thorough penetration of the sodium silicate."—A. MOYER.—*Mines and Minerals*, Jan., 1911, p. 364. (W. A. C.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

CHEMISTRY OF COAL. By MYLES BROWN, M.E. (Wigan: *The Science and Art of Mining.*)

"This little hand-book, comprising about 80 pp., contains a lot of useful information for students, colliery managers, and purchasers of coal. The use of coal analysis and fuel tests has come prominently to the front of late, and is a clear indication that purchasers of coal are becoming dissatisfied with the old form of contract note and are adopting specifications which give the contract note a reliable scientific basis. The information which the author of this book presents should be of considerable help to those who are endeavouring to keep pace with the more advanced requirements of purchasers."—*Iron and Coal Trades Review*, Dec. 16, 1910. (A. R.)

PRACTICAL DATA FOR THE CYANIDE PLANT. By HERBERT E. MEGRAW. pp. 93. Illustrated. Soft cover. Price $2 net. (New York, 1910: Megraw-Hill Book Company, 239 West 39th St.)

"To help the 'Man on Shift' this volume was written. Although it is confessedly a compilation, it is not without original matter. The author, Mr. Herbert A. Megraw, while claiming little originality, hopes that the book will accomplish its object in solving the everyday difficulties of the worker in cyanide. We are sure that, in many cases, it will.

The ground surveyed is as follows: Crushing and grinding, the cyanide plant, slimes, precipitation, formulæ in mensuration, tables of general weights and measures, general reference tables. The section 'precipitation' is thus divided: Solutions, stoichiometry, preliminary experiments on ores, trouble, data. Under these heads, the sequence of steps is well presented. Necessary chemical equations are included and explained. Under the heading 'trouble' a lot of very helpful and practical hints are given." *Canadian Mining Journal*, Dec. 1, 1910. (A. R.)

PRACTICAL SHAFT SINKING. By FRANCIS DONALDSON, Chief Engineer of the Dravo Contracting Co. 8vo, pp. 139. 63 Illustrations. Price $2. (New York City: McGraw-Hill Book Co.)

"This book is written by a practical man engaged in the business of shaft sinking, and covers the ground of modern shaft sinking from A to Z. Chapter I. includes contract agreement for excavation; the extra prices to be paid for excess of water pumped above 100 gallons per minute; and timbering specifications where shafts require to be lined. Chapter II. covers the power plant required for sinking shafts to various depths; the disposal of excavated material; and an itemized cost of an entire sinking plant for a shaft 500 ft. deep. Chapter III. In this chapter the author shows how to sink and support earth from the surface to rock; the various kinds of piling used, including steel; the use and construction of circular and oblong concrete caissons, with the construction of the shoes. Chapter IV. covers the pneumatic process of sinking through soft ground; also the shield method practiced in Illinois. Chapter V. covers rock excavations; the tools for drilling; placing holes for blasting; progress and probable cost per foot. Chapter VI. The processes followed abroad and the Poetsch-Sooysmith freezing process are described. Chapter VII. The Kind-Chaudron process, and the cementation of water-bearing fissures. Chapter VIII. is devoted entirely to lifting water during the progress of shaft sinking. Chapter IX. deals with various kinds of shaft linings used abroad and in this country. Chapter X. gives the cost per linear foot for rectangular, angular, elliptical, and quadrilateral cement-lined shafts."—*Mines and Minerals*, Dec., 1910. (A. R.)

MODERN ASSAYING. By J. REGINALD SMITH. (J. B. Lippincott.)

A text book on "Modern Assaying," which only boasts 150 pages or so, can scarcely be expected to be an exhaustive treatise on so wide a subject. In so far as sampling and crushing go, the matter of the book is excellent, but beyond a description of the various processes employed in assaying gold and silver ores, together with brief notes on lead and copper, and mercury assays—all of which can be obtained in greater detail in any of the ordinary text books on assaying—we see little object in the publication of this book, unless it be as an advertisement for a certain make of ore-crusher. (M. T. M.)

TESTING FOR METALLURGICAL PROCESSES. By JAMES A. BARR San Francisco: *Mining and Scientific Press*), p.p 216. $2.

As a text book for advanced students, this publication is one of the most successful that we have seen. Dealing as it does with chlorination tests, with amalgamation and cyanidation of gold ores—with furnace work and calculations of furnace charges, pyrometry, calorimetry, &c,—it covers the work which a mining or metallurgical student would be required to perform in his final year of study.

Such a book has long been required, and Mr. Barr has certainly made the metallurgical profession indebted to him for putting into a convenient form his laboratory notes. More than that, a large number of the experiments can be performed in ordinary mine assay offices—whether attached to gold, silver, or base-metal mines, so that the book is useful to the practical assayer, as well as to the student. For assayers on the Reef (and cyanide managers, too, for that matter) we can cordially recommend the chapters on "Amalgamating," "Cyanidation," and "Retorting."

Altogether, we have little or no fault to find with the book, which will be of great use in the laboratory. We hope that the author will, at some future date, expand and amplify his book into a comprehensive volume, which will cover all laboratory work for mining and metallurgical students. (M. T. M.)

PRACTICAL STAMP MILLING AND AMALGAMATION. By H. W. MACFARREN (San Francisco: *Mining and Scientific Press*), p.p. 166. $2

This little book should be of invaluable assistance to the battery manager, the amalgamator, and, more especially to the tribute miner and proprietor of the small proposition, who has had little or no experience of metallurgy. Every detail of the routine of stamp-battery practice is dealt with—concisely, clearly, yet fully.

The book is written from the American standpoint, of course, and one notices a few inaccuracies in the references to work on foreign fields, while here and there the author might have gone into the reasons for certain difficulties which crop up, and for the methods employed in overcoming them, a little more fully. However, the book is ostensibly a *practical* one, and the introduction of theoretical detail might have defeated the end which the author had in view, *viz.*, to give in a concise, brief manner, information by which millmen may be guided, and the methods found to be most satisfactory in treating various types of ores.

Mr. MacFarren is evidently a practical man himself, so that his ideas and suggestions are based on his own experience and observations.

One could only wish that some battery-manager on the Witwatersrand would come forward with a similar treatise on the procedure followed on the Rand. Such a book would be of inestimable value to our millmen, amalgamators and students. (M. T. M.)

ROCK DRILLS—DESIGN, CONSTRUCTION AND USE. By EUSTACE M. WESTON, Reef Lecturer on Mining, Transvaal University College. pp. 367. Profusely Illustrated. Price $4 net. (New York, 1910: McGraw-Hill Book Company, 239 West 39th Street.)

"The gaps in professional handbooks are being gradually and well filled. This latest addition to the list of practical books will be welcomed by many.

Until now, there has been no attempt to cover the whole subject of rock drills. We have needed, particularly, a comparison of English, Australian and American drills. Such a comparison is necessary to aid the engineer in his choice of machine for his special needs.

The successive subjects treated are: standard piston drills; hammer drills; electric drills; operating on surface and underground; piston drills using air exclusively; philosophy of drilling rock; repair and maintenance of rock drills; drill steel and drill bits; explosives and their use; theory of blasting with high explosives; examples of drill practice in Africa, Australia and America; drill tests and contests; dust and its prevention; notes on the use of compressed air. It will be seen that the scope of the book is large and that the sequence is logical.

As probably the evolution of the hammer drill is the most important of modern developments, it is worth while digressing here to notice Mr. Weston's estimate of the comparative merits of the piston and the hammer types. Mr Weston points out that, unless one keeps in mind the fact that the rock drill is a commercial machine, one might be tempted to believe the modern hammer drill to be by far the superior machine. The weight of the hammer of the largest type of hammer drill is 15 lbs. The weight of piston, steel, etc, of a piston drill ranges from 60 lbs. to 125 lbs. Thus, whilst the velocity of the hammer drill should be 16 times that of the piston drill to get equivalent effect, in practice it is only four, times as great In other words, a very high air pressure must be used to permit the hammer drill to compete at all with the piston drill. Mr. Weston expounds the subject capably and well.

Most instructive are two chapters entitled, 'Examples of Rock Drill Practice,' and one entitled, 'Rock Drill Tests and Contests.' The former takes up 90 pages. They cover practice in South Africa and in America respectively. Many cost items are given. The chapter on 'Rock Drill Tests and Contests' takes up South African tests largely.

Whilst Mr. Weston's book will call for additions and revisions perhaps more rapidly than books on other subjects, yet is is distinctly a book that will help the mining engineer. In its field it is unique.—*Canadian Mining Journal*, Dec. 1, 1910. (A. R.)

Allen's Commercial Organic Analysis. Vol. IV., 4th Edition, entirely Re-written. Roy 8vo, pp. 474. *Churchill.* Net 21s.

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Abstracts of Patent Applications.

(C.) 21/10. E. J. Way (1), A. J. Arbuckle (2). Improvements in vats or vessels for the treatment of crushed ore products. 22.1.10.

This claim is for a vat, fitted with a number of small cones, and containing both in these cones, and in convenient positions between them, circulating columns in which pulp is made to circulate by the application of air jets, after the manner of the Pohle lift. The claim is for a plurality of such devices contained in one vat.

(C.) 596/10. Otto Stebang Tonnesen. Improved apparatus for sampling slimes and other liquids. 5.12.10.

This application is for a design of apparatus for sampling slime and the like. It consists of a tube with end shaped as a cutting edge, provided with an airtight plunger to retain the sample therein by atmospheric pressure.

(C.) 603/10. Frederick Stubbs. Improvements in safety amalgam receptacles. 8.12.10.

This application is to cover a specified receptacle or vessel for holding the amalgam or black sand that is taken from the amalgamating plates, and the novelty claimed is in connection with a valve at the bottom of the tube or inlet of the vessel, which will prevent any material that has once been put in from being taken out again, excepting by unlocking the lid.

(C.) 628/10. Max Weidtman. Method of separating diamonds and other stones having somewhat the same specific gravity from the matrix. 23.12.10.

This is a device for separating diamonds and other stones, having somewhat the same specific gravity, from the matrix in which they are embedded by treating with a liquid, the specific gravity of which is greater than that of the matter, but less than that of the diamonds.

(C) 638/10. Alfred Arthur Lockwood. Improvements in the treatment of auriferous and argentiferous ores. 29.12.10.

This application refers to the use of an alkaline silicate in treating gold or silver ores with cyanide of potassium, with the object of reducing the consumption of cyanide by such base metal compounds as copper pyrite. It is believed that the base metal compound becomes covered with a thin coating of silicate.

Selected Transvaal Patent Applications.

Relating to Chemistry, Metallurgy and Mining.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 615/10. Peter Nerman Nissen. Improvements in stamp mills. 13.12.10.

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DUNNACHIE, W. J. N., *l/o* Langlaagte; Geldenhuis Deep, Ltd., West Section, P. O. Cleveland.

DODD, M., *to* P. O. Box 2486, Johannesburg.

FILMER, junr., H. J., *to* "Altona," Marshall Street, Belgravia.

HAHN, Prof. P. D., *to* York House, The Gardens, Capetown.

KRAUSE, H. L., *l/o* Klerksdorp; New Kleinfontein Co., Ltd., P. O. Box 2, Benoni.

LEACH, G. W., *l/o* Cleveland, Johannesburg; Bree Street, Parys, O.F.S.

McGUIRE, J. E., *l/o* Johannesburg; P. O. Box 539, Bulawayo.

McLEAN, C. S., *l/o* Germiston; Lonely Reef G.M. private bag, Inyati, Rhodesia.

MEYER, C. E., *l/o* Barberton; Rayton, Transvaal.

MORISON, S., *to* P. O. Box 1447, Johannesburg.

MATTHEWS, E. L., *l/o* Klerksdorp; c/o R. Saner, P. O. Box 3394, Johannesburg.

PINKVOS, E., *l/o* Hamburg; Kirchberg i. Sachsen, Germany.

PACKARD, G. A., *l/o* Wakefield; Raven Copper Company, Butte, Montana, U.S.A.

STACPOOLE, A. R., *l/o* Louws Creek; Cumberland House, Johannesburg.

STOUT, W. H., *to* P. O. Box 1392, Johannesburg.

SOLLY, B. C. T., *l/o* Germiston; P. O. Box 114, Roodepoort.

SOUTHWELL, B., *l/o* Germiston: P. O. Box 50, Johannesburg.

TAYLOR, M., *to* P. O. Box 46, Roodepoort.

TAYLOR, WILFRID, *l/o* Ventersdorp; P. O. Box 68, Gwelo, Rhodesia.

TOM, I., *l/o* Australia; c/o Lewis & Marks, P. O. Box 1030, Johannesburg.

TOWNSEND, C. T., *l/o* Brilliant Extended Cyanide Works; Cyanide Works, One Mile, Gympie, Queensland, Australia.

WAYNE, T. H. B., *l/o* Bulawayo; c/o Standard Bank of South Africa, Limited, 10 St. Clements Lane, London, E.C.

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Vol. XI. MARCH, 1911. No. 9.

Proceedings
AT
Ordinary General Meeting,
March 18, 1911.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the South African School of Mines, on Saturday evening, March 18th, Dr. James Moir (President) in the chair. There were also present:—

62 Members: Messrs. W. R. Dowling, R. Allen, K. L. Graham, Tom Johnson, E. J. Laschinger, J. E. Thomas, A. Whitby, H. A. White, J. A. Wilkinson, James Littlejohn, A. F. Crosse, E. H. Johnson, A. McA. Johnston (Members of Council), W. J. Abel. W. Beaver, B. V. Blundun, J. Chilton, F. W. Cindel, W. M. Coulter, Michael Dodd, E. A. Furner, A. Gullachsen, R. Gascoyne, James Gray, A. J. Herald, J. l'Ons, A. J. Johnson, J. H. Johnson, R. A. Lawrie, G. A Lawson, Henry Lea, J. Lea, L. Marks, T. G. Martyn, G. Melvill, H. H. Morrell, P. T. Morrisby, W. C. Mossop, S. Newton, W. J. North, E. A. Osterloh, J. C. Phillips, H. W. Pidsley, J. B. Polglase, J. F. Pyles, W. A. Quince, D. Robertson, G. A. Robertson, W. H. Roe, F. W. Rush, A. Salkinson, A. Schwarz, S. Shlom, S. H. Steels, Ralph Stokes, J. A. Taylor, A. Thomas, J. T. Triggs, C. F. Webb, E. M. Weston, J. Whitehouse, and A. Wilkinson.

26 Associates and Students: Messrs. C. F. Bayly, C. J. Crocker, J. Cronin, C. A. Damant, D. C. Edington, J. Gibson, J. S. Grace, W. J. R. Hunter, A. King, G. F. Mathews, F. J. Pooler, H. B. Powter, A. G. Rusden, H. Rusden, R Sawyer, P. Scatterty, H. Stadler, E. H. Tamplin, A. M. Thomas, I Tom, P. A. Tucker, H. Ward, F. Wartenweiler, J. C. Webster, P. Wilson, and L. A. Womble.

18 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as printed in the February *Journal*, were confirmed.

NEW MEMBERS.

Messrs. J. Littlejohn and J. A. Wilkinson were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows:—

Castle, Lewis Percy de, c/o Don Proprietary Mines, Ltd., Gwelo, Rhodesia. Mining Engineer.

Gullachsen, Berent Conrad, Geldenhuis Deep, Ltd., P. O. Box 5, Cleveland. Mining Engineer.

Harrison, Mark John, Knight Central G. M. Co., P. O. Box 91, Germiston. Amalgamator.

Martyn, Thomas Graham, New Primrose G. M. Co., P. O. Box 193, Germiston. Metallurgical Chemist.

Newbery, J. W., The Tarquah Mining and Exploration Co., Tarquah, Gold Coast, West Africa. Mine Manager.

Polglase, J. B., Durban Roodepoort Deep, Ltd., P. O. Box 110, Roodepoort. Mine Overseer.

Rouillard, Anthony Philippe., Durban Roodepoort Deep, Ltd., P. O. Box 110, Roodepoort. Mine Manager.

Thompson, George Robert, B.Sc., A.R.S.M., S. A. School of Mines, P. O. Box 1176, Johannesburg. Professor of Mining.

Wilson, James Careh, Weynek Tin Co., Ltd., P. O. Rooiberg, *via* Warmbaths. Mine Manager.

Winterton, Arthur Henry, East Rand Proprietary Mines, Ltd., P. O. Box 91, East Rand. Mill Foreman.

The Secretary: Since the last meeting of the Society the following have been admitted by the Council:—

As Associates.—

Bell, Herbert Charles Fitzwilliam, P. O. Box 21, Randfontein. Surveyor. (*Transfer from Student Roll*).

Cowlin, Hugh Houghton, A.R.S.M., Simmer Deep, Ltd., P. O. Box 178, Germiston. Metallurgical Engineer.

Edington, David Cameron, B.Sc.(Edin.), Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Mining Engineer.

Porter, Robert Allan, Simmer Deep, Ltd., P. O. Box 178, Germiston. Reduction Works Sampler.

As Students.—

Abbey, Thomas Shaw Hillier, Geldenhuis Deep, Ltd., North Section, P. O. Box 5, Cleveland. Cyanide Learner.

Davis, Walter Herbert, Geldenhuis Deep, North Section, P. O. Box 5, Cleveland. Learner.

Spence, Eric Hudson, City Deep, Ltd., P. O. Box 1411, Johannesburg. Learner.

General Business.

Mr. Andrew F. Crosse (*Past-President*): I have a suggestion which I should like to bring up before you this evening. It would be a very good scheme for our Society to arrange for either a temporary or perhaps even a permanent exhibition of the most modern and up-to-date appliances—chemical, physical and mechanical—which are required for assaying, sampling, ore crushing, testing various products, sieving, etc. The list is so long that time does not allow of my mentioning any more this evening—there are so many recent improvements and such varied new apparatus which could be exhibited, in some central position in this town, which would be very interesting to our members and valuable to the mining industry. For instance, the other day, during our visit to the Crown Mines, Mr. Stadler, who has done so much work in connection with grading analysis, showed me a very ingenious invention of his own, for passing a sample of crushed ore through a series of sieves. All such improvements as this should be collected under one roof and exhibited for the benefit and instruction of those interested.

I hope that our Council will take up this matter, and see if my suggestion cannot be made to assume a practical form.

The President: I think this is a very interesting suggestion, and I should like to ask Mr. Crosse whether he means this to be a permanent exhibition?

Mr. A. F. Crosse: My idea is that it should be a temporary one, though I believe a temporary exhibition would probably become a permanent one. I think that all the different people who sell these things would only be too keen to have an exhibition for housing their exhibits. Most of us have but little time to go about, though if one had time to go along the reef we would see many things in use that we are not familiar with, whereas if there was a central building where all these things could be assembled, a man could come in on a Saturday afternoon and see the whole outfit.

Mr. E. M. Weston (*Member*): I am speaking without authority, but I feel sure that the Council of the School of Mines would only be too happy to allow their museum to be utilised for such an exhibition.

The President: There is no doubt this idea would be very beneficial to the industry and especially to those who cannot get to Europe every second year to see what progress has been made. Of course there is also the other museum which belongs to the town in the old building of the college. It is a pity we cannot have one decent museum instead of a lot of separate little affairs. However, we will submit the idea to the Council and see if some practical scheme cannot be arranged.

THE AMALGAMATION OF GOLD IN BANKET ORE.

By W. R. Dowling, M.I.M.M. (Vice-President).

Amalgamation being the most important source of gold recovery on the Witwatersrand goldfield, recent developments and present varied opinions upon this subject make the time opportune for a general discussion and review of the situation.

The author does not propose to discuss the manipulative details of the method of amalgamation; such knowledge is assumed, and the general features and present tendencies of the process with the influencing factors only are dealt with. The decreasing grade of the ore due to the fact that the reduction in working costs has brought into the paying zone ore formerly not payable, and the large scale of operations to-day, necessitates the simplification and reduction of plant and operations to the minimum. The enormous tonnages of ore milled make each item of equipment and its operation cost a great deal in the aggregate, and the fact that the interest on some particular non-essential item of expenditure does not amount to much per ton crushed is beside the mark, as in many cases the capital may not be available, or, if available, could possibly be employed more profitably otherwise.

As will be shown, the relative importance of the amalgamation recovery and the methods by which it was carried out have been considerably modified by various factors from time to time in the metallurgical history of the Witwatersrand. In the early days the main source of gold recovery was by amalgamation with a small amount of concentration and subsequent chlorination. The ore was crushed relatively fine by stamps, the screens mostly in use being 800 to 1,000 holes per square inch, and the discharge was set high to aid fine crushing and inside amalgamation. To obtain the maximum recovery it was considered necessary to catch the gold at the earliest possible moment after its release from the ore by crushing. To this end amalgamated copper plates were set inside the mortar boxes, which were made large and roomy and recessed at the back to protect the copper plates. Mercury was fed into the box periodically, and lip and splash plates received the outflow of pulp. Following these plates was the large apron plate, the latter being the only plate in use in most mills to-day. It was usual to cut the apron plate into two or more lengths, and arrange these in steps, mercury wells being placed at each step for the purpose of mixing the pulp and recovering possible floating particles. A large proportion of gold remained in the mortar-box as amalgam on the inside plates and around the dies, and was periodically cleaned up. Some of the gold discharged through the battery screen had already been wetted by mercury, and was also in the form of amalgam, and was caught on the outside plates. At the present time amalgamation tends more and more to be performed by the passage of pulp over a single plain inclined stationary plate, and the above devices of the early millman are likely to be followed into oblivion by the mercury trap, in view of the most efficient mercury trap which the tube-mill circuit affords. Cyanide was the favourite chemical employed for dressing plates, and was liberally used. It might be mentioned in passing that in some mills, many years after the introduction of the cyanide process, cyanide was still in use (1). The large mortar boxes necessarily reduced the crushing capacity of the stamps, and it was due to the introduction of narrow straight-backed boxes for the purpose of increasing crushing efficiency that inside plates became impracticable. The mercury feed to the mortar boxes, however, continued much later (2), it being argued by the exponents of this practice that since a particle of amalgam was larger and heavier than its enclosed gold particle, it was more readily caught. Inside amalgamation and mercury feed to the mortar boxes increased the mercury consumption, and made it impossible to obtain a reliable sample of the ore crushed.

Concentration by means of blanket strakes, buddles, Frue vanners, and other means was practised, and some amalgamable gold was recovered, as well as no doubt some rusty gold from the oxidised ore. Evidence of the amalgamable gold reaching the concentration plants is the amalgam recovery made on the small plates attached to the Frue vanners then running, and also on some shaking secondary plates installed at the Ferreira Mine (3).

Information is very meagre and unreliable as to the percentage of recovery by the means outlined above. The assayer in the early days was not considered a necessary official on the mine, the pan in the hands of the mine captain or mill man being thought the more reliable means of determining the value of the ore. In December, 1894, however, J. S. Curtis sets the recovery at over 60 per cent. where a 900 mesh screen was used (4). It appears, then, that the percentage recovery of the early days on an oxidised and rich ore, by the various amalgamation methods then in use, was not much less than is obtained by the present fine crushing from low-grade pyritic ore by the ordinary straight mill plate and shaking tube-mill plates, as used in most reduction works to-day.

A most important change in the methods of gold recovery was brought about by the introduction of the cyanide process early in 1890. The cyanide process in the first instance was only applied to leachable sand. This had its influence on the crushing of the ore and amalgamation recovery, since, with a view to the production of as little slime as possible, the tendency was to crush coarser, thus decreasing the amalgam recovery, although the total recovery of the gold contents of the ore was considerably improved.

When the treatment of slime was successfully introduced by our Past-Presidents, J. R. Williams and Charles Butters, in 1894, an entirely new view of ore treatment was rendered possible. The production of slime being no longer detrimental to the subsequent treatment by cyanide but the reverse, the ore was now crushed finer, thus again increasing the recovery by amalgamation. The recovery by the latter process was further improved by the use of lime in the mill service water, used to accelerate the settlement of the slime, as it was found that the agglomeration of the slime particles into larger aggregates enabled these to come into contact with the amalgamated surfaces and yield an appreciable proportion of their gold contents (5).

The increased recovery of gold obtainable by finer crushing and the removal of the main objections to this procedure by improvements in sand and slime treatment were generally recognised before the war (1899 to 1902). During this period, when local advance was debarred, progress in fine crushing was made in Australia by the introduction of tube-mills. The introduction of tube-mills for finer crushing in the Transvaal by J. R. Williams in 1904 (6) raised the recovery of gold by amalgamation to 65-70 per cent. of the contents of the ore, of which some 10 to 15 per cent. was by means of the tube-mill plates. The amalgamation recovery varied somewhat on different mines according to the re-crushing plant erected, screening used, tonnage crushed, and value and nature of the ore.

Coming nearer to the present time, it is found that the increased use of tube-mills for re-crushing enables so coarse a screen to be used in the stamp-mill that amalgamation of battery pulp becomes an impossibility owing to the scouring action of the coarse pulp upon the mill plates (7). This necessarily leads to the removal of the plates from the stamp-mill in such cases, and a consideration of the following possible alternatives represented graphically in the corresponding diagrams :—

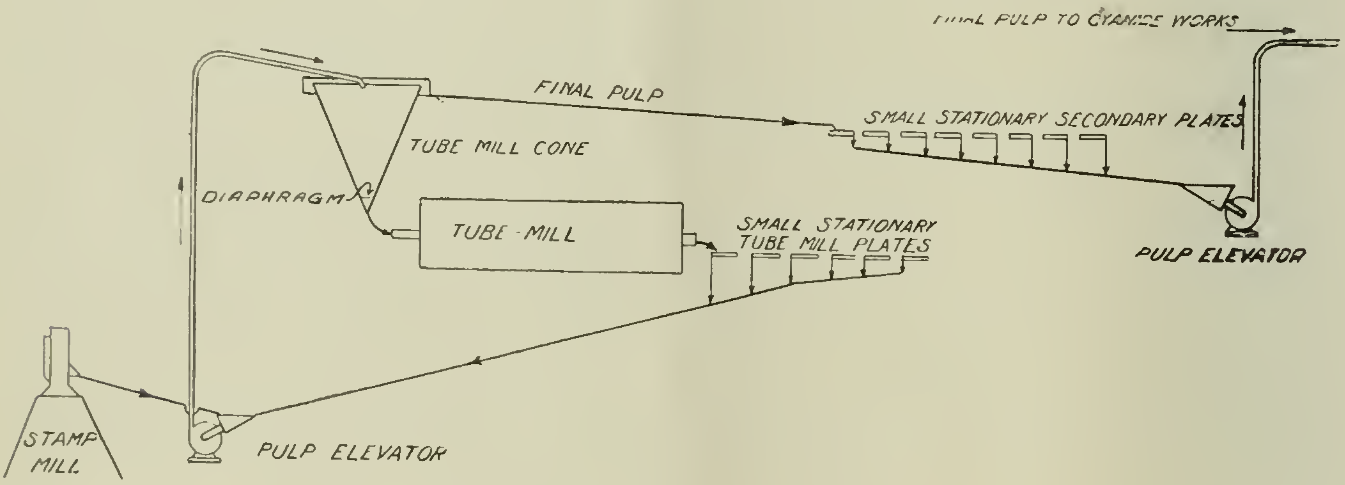
(A). Amalgamation of the tube mill pulp by one set of plates, and the overflow final pulp by another set, as at the Randfontein Central and Knight Central.

(B). Amalgamation of the tube-mill pulp only, as at the Simmer and Jack, Simmer Deep-Jupiter, and the various South Randfontein mills.

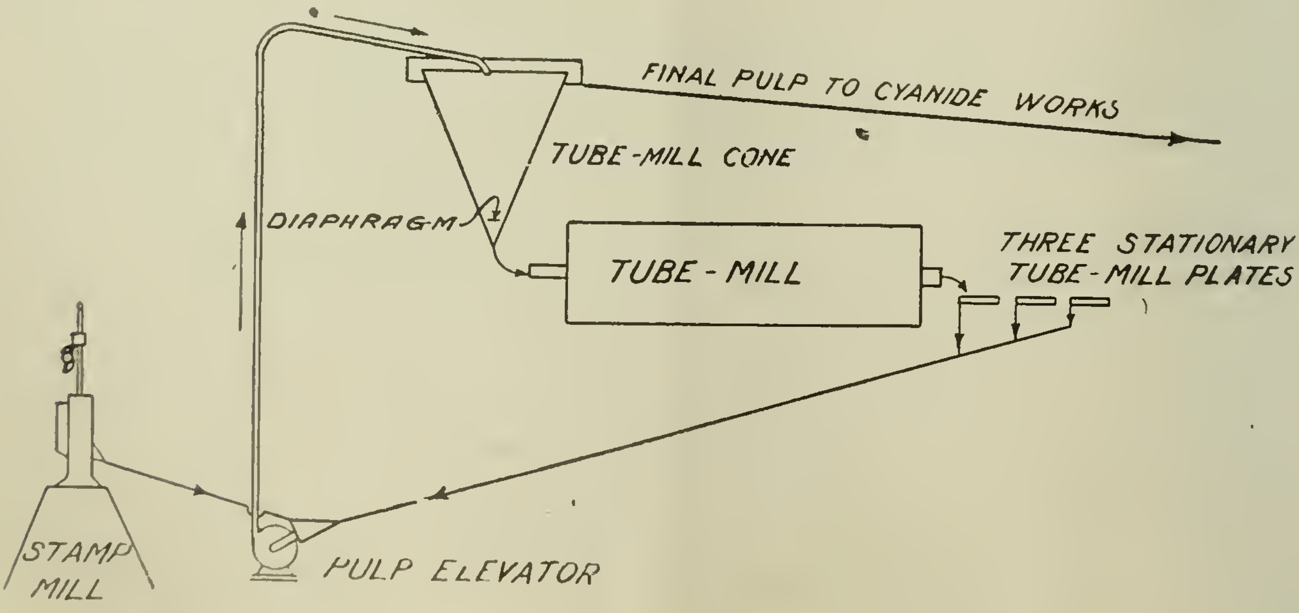
(C). Amalgamation of the mixed pulp after re-grinding of the coarse portion by tube-mills, as at recently erected plants.

After the failure of various efforts to eliminate tube-mill plates, it may now be accepted that there is general agreement amongst metallurgists as to the advisability of amalgamating the tube-mill pulp, since otherwise considerable concentration of coarse gold by the classification of both stamp and tube-mill pulp takes place in the tube-mill circuit. Tube-mill pulp has only to be examined for metallic iron from the wear of shoes and dies and from other sources to illustrate how heavy particles will persist in the circuit till crushed or abraded fine enough to overflow the classifiers.

With efficient tube-mill classification, all but the finest gold released by the stamps is brought into the tube-mill circuit and recovered on the plates, together with that released by crushing in the tube-mills. This being the case, the problem narrows itself down to determining whether it is necessary or advisable to amalgamate the final pulp, leaving the crushing plant as an overflow of the tube-mill classifiers. The considerations which enter into this problem are the extra capital outlay and buildings, the capital locked up as amalgam as setting to the plates, the increased area of amalgamated surface to guard, the consumption of mercury, the labour of dress-


FINAL PULP TO CYANIDE WORKS
FINAL PULP
SMALL STATIONARY SECONDARY PLATES
TUBE MILL CONE
DIAPHRAGM
TUBE -MILL
SMALL STATIONARY TUBE MILL PLATES
PULP ELEVATOR
STAMP MILL
PULP ELEVATOR

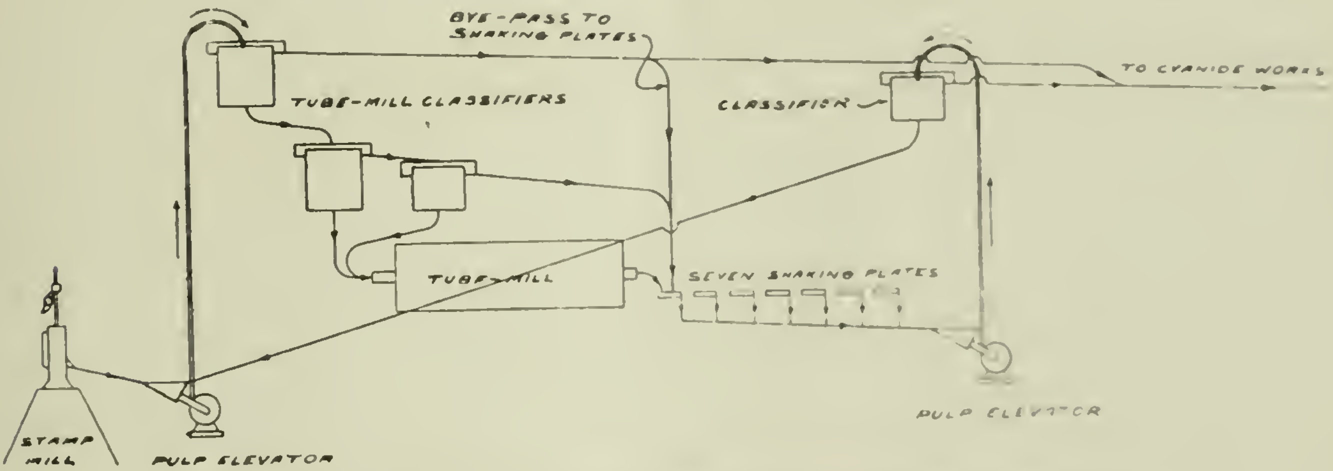
A.—Randfontein Central Arrangement
Six small stationary tube mill plates and eight small stationary secondary plates for final pulp.


FINAL PULP TO CYANIDE WORKS
TUBE-MILL CONE
DIAPHRAGM
TUBE - MILL
THREE STATIONARY TUBE-MILL PLATES
STAMP MILL
PULP ELEVATOR

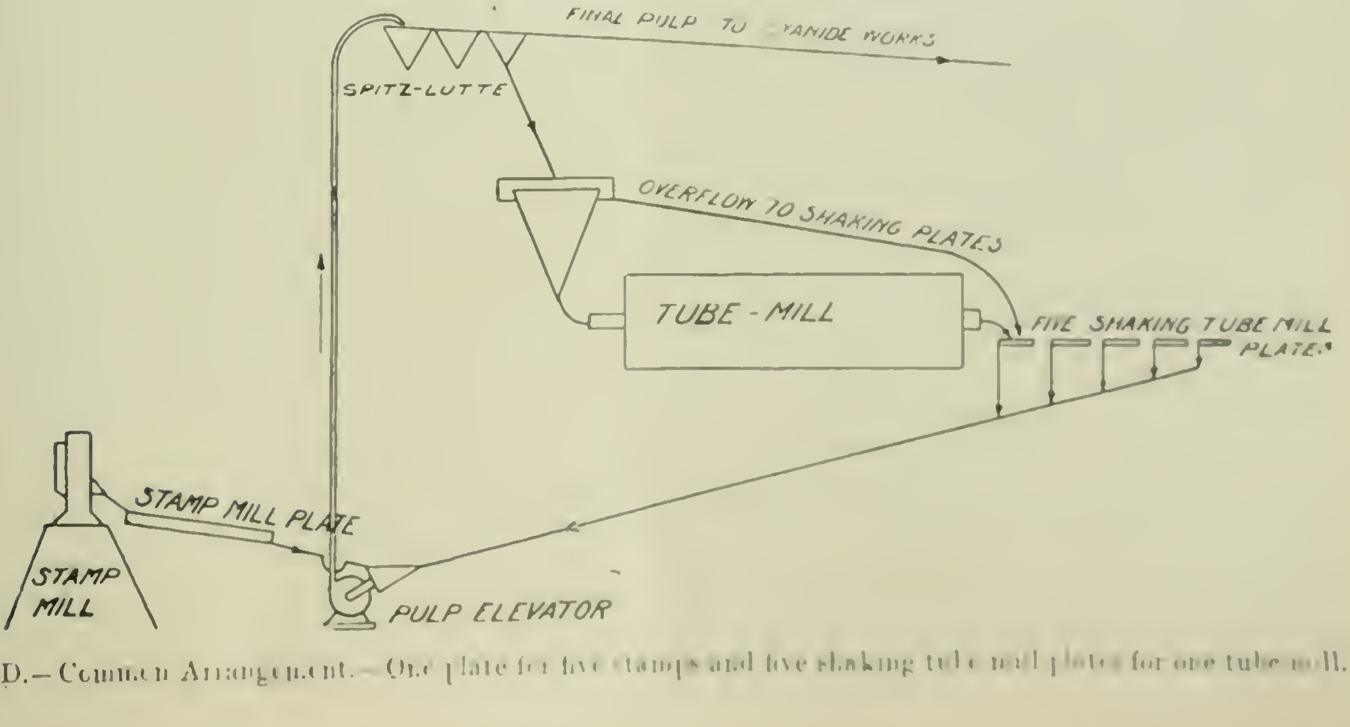
B.—Simmer and Jack Arrangement.—Three stationary plates in tube mill circuit.

ing, and the additional elevation of pulp necessitated by its fall during this amalgamation process. Even if mercury traps are used at the foot of amalgamated plates outside the tube mill circuit for the pulp overflowing the tube-mill classifiers, it is possible for fairly coarse mercury and amalgam to escape to the cyanide plant.

On reference to the sketches, it will be noted that alternatives (A) and (C) have two elevations of the total pulp, plus the tube-mill pulp, whereas (B) has only one elevation of the total pulp, plus the tube-mill pulp. Alternative (B) therefore does not involve any extra charge for re-elevation, and remains the same as in the usual present-day practice, as represented in sketch (D). Unless increased recovery counterbalances these disadvantages, it is preferable to omit this additional operation. The use of very coarse battery screening naturally reduces the amount of metallic gold set free in the stamp-mill, and where the bulk of the crushing is done by the tube mills, the quantity of very fine gold which might reach and overflow the tube mill cones is correspondingly reduced, as such gold is retained on the tube mill plates before the elevation of the tube-mill pulp. Even where the coarseness of crushing in the stamp mill does not preclude the use of plates there, their elimination, when conditions permit, is followed by the various advantages above mentioned, and the rendering available as immediate profit the gold on such plates, which would otherwise not be realised until the end of the life of the mine. Plates thus taken out of the stamp mill may carry from £50 to £300 worth of gold per plate, according to their condition, and in the aggregate this may render some £20,000 worth of gold immediately available in the case of a large mill. The elimination of the final pulp plates, however, can only be considered safe where good tube mill classification obtains, and an equally efficient cyanide treatment follows. In the cyanide plant



C.—Recent Arrangement.- Seven plates amalgamating total mixed pulp.


FINAL PULP TO YANIDE WORKS
SPITZ-LUTTE
OVERFLOW TO SHAKING PLATES
TUBE-MILL
FIVE SHAKING TUBE MILL PLATES
STAMP MILL PLATE
STAMP MILL
PULP ELEVATOR

D.—Common Arrangement.—One plate for five stamps and five shaking tube mill plates for one tube mill.

clean separation of sand from slime is essential, such as is possible with large diaphragm cones and continuous sand collection, producing on the one hand freely leachable sand, and on the other a large percentage of slime free from coarse sand. With tube-mill classifiers liable to choking of small underflow outlets and consequent overflow of coarse sand and gold to the cyanide plant, and with slimy semi-permeable sand charges, the wisdom of allowing any fine amalgamable gold to escape from the crushing plant is doubtful, in spite of the economies above mentioned. In the case of the Simmer and Jack mill, where there are plates in the tube-mill circuit only, though the cyanide residues remain normal, the amalgam recovery dropped from 64% to 55% on a 6¾ dwt. ore, owing to the fact that the fine sand and slime overflowing the classifiers are not amalgamated at all. If necessary, and as shown later in the case of the Simmer Deep plant, finer crushing and larger relative tube-mill cone area could be employed to increase the percentage of gold recovery by amalgamation. Whilst the collected sand is slightly enriched by some very fine free gold, there is no difficulty in dissolving this and obtaining a complete recovery. The popular theory that only 85% of this gold is recovered by cyaniding, instead of 100% by amalgamation, has not so far been borne out either in the laboratory by residue assays or regular panning of residues. Any fine metallic gold overflowing the tube-mill cones is necessarily smaller and more easily dissolved than the average of partly-encased particles in sand which are dissolved by cyanide. The amount of fine metallic gold overflowing the tube-mill cones is much affected by the number and area of the latter. Where a large ratio of tube-mills to stamps exists, such gold, more often than not, cannot be detected in the final pulp by panning or in the sand before treatment. This is the case at the Simmer Deep plant, where, before the removal of the stamp-mill plates, the total plates in operation numbered 82, equivalent to 5,276 sq. ft., of amalgamating area, and the amalgamation recovery was 56·8% of the ore value. After the removal of all the stamp-mill plates and the reduction of those in the tube-mill circuit to 30 stationary plates, presenting an area of 1,700 sq. ft., the amalgamation recovery was 57·7%, and the value of sand and slime before and after treatment has not increased. The total extraction by amalgamation and cyaniding was 93·4% before and 93·5% after removal of the stamp-mill plates. There was slightly finer crushing of the ore in the latter case, there being 77·6% of − 90 mesh (0·006 in.) product in the final pulp before removal of the mill plates, and 81·0% of − 90 mesh (0·006 in.) product after the removal. Where the ratio is lower, as at the Simmer and Jack, a trace of very fine gold can be detected by panning the final pulp and collected sand, though the assay value of the sand residue remains normal. The slime is also enriched, though this may be off-set by better washing, due to the lower percentage of moisture on settlement of well-classified slime, containing a higher percentage of fine (− 200 mesh) sand. For instance, assuming that 1·5 dwt. and 1·8 dwt. slimes are treated, settling to 42·5 and 35% moisture respectively, by two 4 : 1 washes in both cases, of which the first wash only is precipitated down to 0·01 dwt. per ton of solution. With 0·08 dwt. undissolved gold in the first residue and 0·10 dwt. in the second, the total extraction will be 90·4% (0·144 dwt. total residue) from the 1·5 dwt. slime, and 92·2% (0·14 dwt. total residue) from the 1·8 dwt. slime. Where Butters' or other vacuum filters are used, the value of the original slime should be immaterial, as, providing the dissolving of the gold is satisfactory, the residue should contain little more than the trivial amount encased. It is to be hoped that detailed results and working costs of the recent work in this direction at the Crown Mines will before long be laid before our Society.

The author is indebted to F. A. G. Maxwell for much information on regular work at the Randfontein Mines, where plates were eliminated from the stamp-mill, and hopes that he will bring forward his results as a discussion to this paper. In the meantime, it may be said that during 1908, in the four 100-stamp sections of the South Randfontein, when there were amalgamated plates both in the stamp-mill and the tube-mill circuit, the average amalgamation recovery was 54·46%, and the total recovery 92·06%, whereas in 1910, when there were plates in the tube-mill circuit only, the amalgamation recovery was 48·60% and the total recovery 92 94%.

As mentioned in a note lately presented by the author to this Society (8), it has generally been considered necessary for the satisfactory amalgamation of the large volumes of thick tube-mill pulp that the plates should shake. Tracing back the history of this belief to the pioneer work done by J. R. Williams on the Glen Deep in 1904, it seems that in order to prevent the banking of sand on the stationary plates installed with 10% grade, it was found necessary to mount the plates on shaking vanner frames. Not only has the shake been subsequently adopted by all the mines without question, but the number of plates has gradually increased from the two originally installed to five and even six later. When the Knights Deep had five tube-mills, equivalent in cubic capacity of shell to 3·91 standard tube-mills of 22 ft. × 5½ ft., sixteen shaking plates 10½ ft. × 4¾ ft. were in operation. As there was no convenient space for more plates when another tube-mill was erected, it was decided to make the sixteen plates serve. The number has now been reduced to fourteen. The ratios are:—

For 3·91 tube mills 16 plates = 4·09 plates per tube-mill.

For 4·91 tube-mills 16 plates = 3·26 plates per tube-mill.

For 4·91 tube mills 14 plates = 2·55 plates per tube-mill.

The amalgamation recovery was just as efficient with the smaller ratio as with the larger.

In the course of the experiment mentioned in my note, and after having thoroughly tested the stationary five plates system, initiated originally by F. A. G. Maxwell at Randfontein (9), it was decided on W. A. Caldecott's suggestion to perform the whole amalgamation of the tube-mill pulp on two stationary plates per tube-mill. The Simmer East plant has three 22 ft. × 5½ ft. tube-mills. The pulp of each mill has been regularly run over two plates only since December 19th, 1910. The dressings have been at six hour intervals and the scrape taken six inches lower than formerly. The daily scrapes have been normal, yielding the amalgam called for. The two plates of one of the mills were steamed at the beginning of the run and again at the end of a month's run to determine whether the yield from this source was affected in any way. The previous practice in this plant was to steam one-third of the total plates each month so that each plate ran for three months before steaming. It was found that one month's run of the two plates yielded more amalgam when multiplied by three, than the average of the three months of five shaking plates previous to starting the non-shaking experiments. However, to be on the safe side, it was assumed that the plates would not accumulate steam amalgam at the same rate during the second and third months, and it was decided to multiply by only two and a half. This calculation gave a figure for steam amalgam just about equal to the average of the three months of five shaking plates. Adding this to the daily scrape and other sources the recovery is fairly arrived at, and works out at 21·54% of the screen value and 34·17% of the tailing value, compared with 21·80% and 34·01% of the previous five-shaking plate period.

To ensure successful and economical work the ratio of water to solid should be reduced to the minimum, as not only does this decrease the velocity of the stream over the plate, but the cost for re-elevation is also less. The volume of pulp of 400 tons of sand per tube-mill plus 480 tons of water, allows about 10,000 cubic feet of pulp per 24 hours for each of the two plates, or about the same as passes over the plate of a five-stamp battery with a 9-ton duty per stamp, and a 6·5 to 1 ratio of water to ore in the screen pulp. The tube-mill circuit is the only part of the crushing plant where a thick amalgamable pulp is available without the installation and operation of special classifiers for the purpose. Distribution of the stream across the full width of the plate is also of considerable importance, and may be obtained by using the usual box at the head of the plate perforated with about ¾ in. holes along the bottom of the side facing down the plate. These holes should be of such capacity that the pulp attains some head in the box. In the event of any of the holes becoming temporarily choked, a further row of holes an inch or two higher up the front side of the box will serve to carry over the stream, and still maintain a fairly good distribution. The fall given to the stationary plates is 18%. This is the fall arrived at by Mr. Maxwell and since adopted here. With the fall of 18% it is found that the minimum percentage of water in the sand is 55%, and it appears that should it be desired to reduce the fall, the moisture would have to be increased, fall and water ratio being convertible terms within limits. The point to be reached in water ratio and fall is that just short of banking of the sand on the plate. Even with a low water ratio the rush of pulp over two plates appears to the eye to be too great, but this is not really so. As long as the velocity of the pulp stream is such that particles in suspension may come into contact with the plate and not be carried on mechanically without touching the plate, the gold will be caught. It might be considered that pulp carrying only 55% of moisture is too thick to allow the gold to sink through and reach the plate. As, however, amalgamation has been proved efficient with this ratio, this does not appear to be the case. In this connection, it might be mentioned that in pan amalgamation the pulp is kept very thick, although in plate amalgamation a fluid pulp was heretofore considered necessary for good recovery. On a stationary plate there appears a tendency for the gold to amalgamate further down the plate than when shaking; there is hence a liability for the scraping to yield less amalgam and the steaming more with stationary plates as compared with shaking plates. In connection with the satisfactory results obtained in the operation of the two stationary plate method, much credit is due to the practical skill and progressive attitude of Mr. A. J. Herald, the Knights Deep mill foreman, and to his assistant, Mr. Peter Wilson.

The removal of plates from the stamp-mill has had the good result of introducing on these fields the separate plate-house. The Homestake mine and the Waihi (10) set the example many years ago, but the Witwatersrand has been very slow in following the lead and accepting the suggestion made before this Society in 1903 by H. R. S. Wilkes (11). The arguments advanced by some mill-men that a plate attached to each battery makes for good recovery due to better distribution of the pulp by the stamps over the width of the plate, and that even vibration assists lacks confirmation. It appears to the writer that this method is just one of the many practices taken over from other countries without investigation. Tube-mill plates have now been running long enough to prove that most efficient amalgamation can be performed with a pulp sub-divided from a launder and distributed over the width of the plate from a box pierced with holes. Tube-mill pulp carries heavier particles and contains less water than mill pulp, and is therefore a more difficult pulp to distribute. The advantages of a separate plate-house are increased running time, since stamps need not be stopped for plate dressing, and greater attention to the crushing machinery by the attendants, thus increasing the crushing capacity. Again, better amalgamation should be obtained by the greater attention to the work by specially chosen and trained men. The whole operation is confined to a smaller area and lends itself to closer supervision so that the risk of loss by pilfering may be reduced to a minimum. In connection with the matter of conveying pulp the author considers that sufficient importance is not attached to launders. All launders should be as smooth as possible and free from obstruction at the joints. Of all material used for the lining of launders there does not seem to be anything better than cement. This provides an ideal surface presenting least resistance to the flow of the pulp and affording very little lodgment for amalgam and still less for gold in unamalgamated pulp. On those mines here where separate plate-houses or combined plate and cyanide extractor houses have been built, the arrangements have not gone far enough or else too far. The erection of zinc-lathes, elevator pumps and tube mills in the same building as the plates and the extractor boxes means the introduction of gangs of men and natives for repairs, other than those engaged on the gold recovery work. If this machinery is placed in the same building because the launders may carry gold, then the whole tube-mill plant including circuit launders and elevators should be included. The best plan is to include in the building only such machinery as is actually connected with recovery work, so as to limit the men entering the building to those directly engaged in the work. The accumulation of amalgam in the tube-mill launders should be reduced to a minimum and the launders should be closed to protect what does collect. Prior to the introduction of tube-milling amalgam was to be found in the launders to the cyanide plant, and some found its way as coarse particles to the concentrate vats, entailing loss in the residue. Where efficient tube-mill classifiers are installed no amalgam is found to-day reaching the cyanide plant except in an impalpably fine condition, since heavy particles of amalgam pass into and are retained in the tube-mill circuit. There is always a loss of mercury in amalgamation mainly by abrasion, and where this enters the tube-mill amalgamation of the gold liberated by crushing takes place. Where this amalgam does not reach the plates it is found in the launders between the discharge of the tube-mills and the plates. Such impalpably fine mercury or amalgam as enters the cyanide plant dissolves in the same way as gold, and like dissolved lead salts is either wholly or partially precipitated by soluble sulphides in the working solution, or later in the metallic form by zinc shavings in the boxes.

Where plates are eliminated from the stamp-mill the water ratio may be considerably reduced. The clearing of a long plate set at a fall of 9% or 10% by avoidance of any banking of sand from the pulp has been the factor determining the water ratio in the past, and this requires six to seven parts by weight of water to one of solid using average battery screening. It has been found that in the absence of mill plates the ratio may be reduced to 4·5 without serious ill effects provided the mill launders have sufficient grade to carry pulp of this consistency. A coarse screen with less water will give the same result in duty and grading as a finer screen and more water, water and screening being convertible terms within limits. The reduction of water has considerable influence on the cost of pulp elevation and of running water pumps, the reduction of weight from 6·5 to 4·5 of water to 1 of solid being equivalent to about 27%. The saving in power in a large plant is material when working on the reduced ratio, and there is likewise a smaller load on the classifiers and slime collectors. There would, however, be no saving in the consumption of water as much the same volumes are exposed to loss by evaporation, seepage in dams, and moisture in residues.

Although there is no rule for the area of amalgamating plates there is no doubt that the number and area of plates in common use is unnecessarily large and when these are placed in a separate plate-house the total area may be materially reduced. The author is of opinion that much of the amalgam found at the lower end of long battery plates is largely worked down by amalgamators wishing to present a uniform bright surface all over the plate. The old practice was to instal plates about 15 ft. long × 5 ft. wide per battery of 5 stamps, which equals 15 sq. ft. per stamp. Where say one standard large tube-mill for 30 stamps is erected, and assuming that the tube mill is equivalent to 30 stamps, five plates of 11 ft. × 5 ft. would probably be erected. The combined plate area is then :—

30 stamps = 6 plates 15 ft. × 5 ft. = 450 sq. ft. = 15 sq. ft. per stamp.

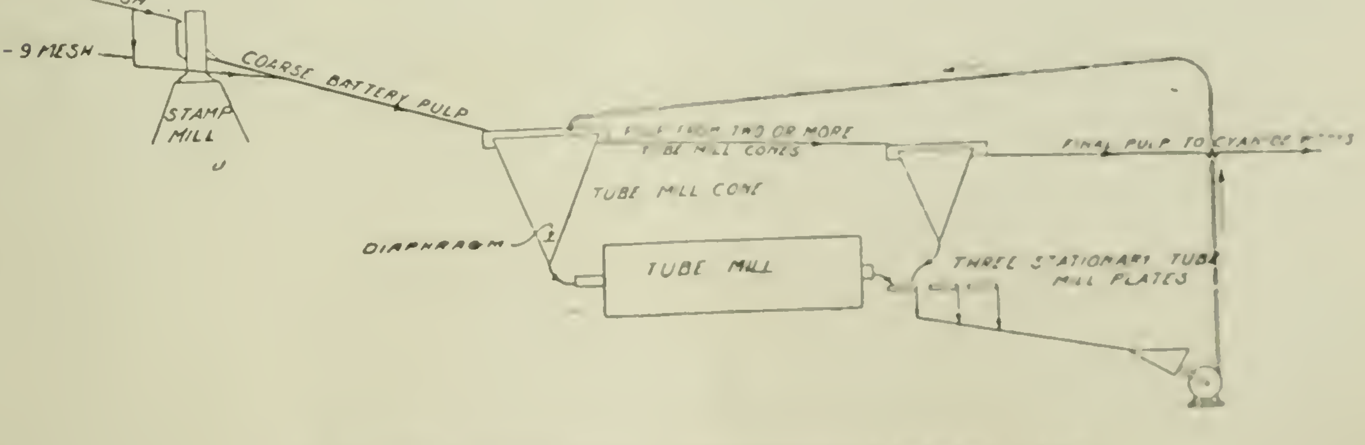
1 tube-mill = 30 stamps − 5 plates 11 ft. × 5 ft. = 275 sq. ft. = 9 sq. ft. per stamp.

Total for 60 stamps units = 725 sq. ft. = 12 sq. ft. per stamp unit.

At the Randfontein Central the total plates erected amount to 9·6 sq. ft. per stamp unit but it is now proposed to use only 4·8 sq. ft. per stamp unit, and there is little doubt that the latter area will be quite sufficient. In the Simmer and Jack re-arrangement the plate area is about 2 sq. ft. per stamp unit. With the high stamp duties of recent years the same plate area is doing nearly double the work it used to do in the past, and is capable of doing a great deal more. Conversely, in regard to the usual 10% fall, where this is increased more crushed material with smaller water ratio may be amalgamated with equally good results. H. W. McFarren in his recent interesting little book (12) says the fall of plates varies from 1½ in. to 3 in. per ft., and should not be less than 2 in. or 2½ in. A fall of 2 in. per foot is equivalent to 16·7%, and 2½ in. to 20·8%, whilst the average is 18.75%. Among the advantages due to reduced area of plates are reduced cost of installation and operation, and especially a reduced cost of mercury the consumption being in proportion to the area of plate. In the Simmer and Jack plant the consumption of mercury has been reduced to one-sixth of the former figure. Where a large area is exposed in amalgamation a proportionate amount of gold is taken up to set the plates, and is not available for realization until the end of the life of the mine. The gold held by well-set plates may be taken at 1 oz. per sq. ft. of plate. A 200 stamp-mill having 40 plates 15 ft. × 5 ft. will thus absorb 3,000 oz. of gold worth £12,000. The future points to the reduction of plate area to the point where only the gold too coarse for cyaniding will be caught by amalgamation. This point has probably been reached in the Simmer and Jack plant which has now only eighteen stationary plates in the tube-mill circuit in place of the former 64 battery plates and 30 shaking tube-mill plates. It will be noted that the Simmer East with plates in the stamp-mill have retained two stationary plates per tube-mill, whereas the Simmer and Jack and the Simmer Deep-Jupiter joint plants without plates in the stamp-mill have each retained three plates per tube-mill. The additional plate in the last two cases is a measure of safety to deal with the richer pulp and to have not less than two plates per tube-mill in action during dressing operations. Diagram (E) shows a proposed arrangement of crushing and amalgamating with three stationary plates only per tube-mill and no plates in the stamp-mill, and with safety cones for the common streams of pulp overflowing the tube-mill classifiers (13).

The author trusts that the foregoing account of some considerations affecting the methods of gold recovery responsible for some £20,000,000 per annum, two-thirds of the Rand's total output, may be of interest to members, including those who have visited the plants where the changes described have been effected, and he trusts during the course of his reply to the discussion to furnish future comparative details of future results.

In conclusion the author wishes to thank Mr. G. A. Chalkley, the Manager of the Knights Deep, and Mr. C. D. Leslie, the Superintending Engineer of the Consolidated Goldfields of South


+9 MESH
-9 MESH
COARSE BATTERY PULP
STAMP MILL
FROM TWO OR MORE TUBE MILL CONES
FINAL PULP TO CYANIDE WORKS
TUBE MILL CONE
DIAPHRAGM
1
TUBE MILL
THREE STATIONARY TUBE MILL PLATES

E.—Proposed Arrangement.—Three stationary tube mill plates.

Africa, for access to information of the various mines of the group, and for permission to read this paper before the Society.

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The President: It gives me very great pleasure to propose a hearty vote of thanks to Mr. Dowling for his excellent paper. Not the least interesting part of this paper is, I think, the excellent historical review he gives at the beginning, which is very useful to those of us who were not here before the war. I think it is a precedent which others who had the privilege of being here before the war might follow.

Mr A. F. Crosse (*Past-President*): As Mr. Dowling was my assistant for some years, I should like to take this opportunity of mentioning my appreciation of his paper this evening. I am sure it will lead to a very interesting discussion, and it is a good thing we have a milling paper to discuss as they have been rather scarce in the past.

Mr. K. L. Graham (*Member of Council*): I have very great pleasure in seconding the vote of thanks to Mr. Dowling for his paper. His contributions to the Society are always interesting, and this one is no exception to the rule. It opens up an enormous field for discussion and will probably occupy a lot of time at our next few meetings.

AIR LIFT AGITATION OF SLIME PULP.

By Robert Allen, M.A., B.Sc., M.I.M.M.
(Member of Council).

One of the principal causes of the fall in the price of silver in recent years has been the increased production of the metal, due to the impetus given to the industry by the great improvement that has taken place in the methods of cyaniding ores. Formerly these ores, when treated by the cyanide process, were first milled, then generally concentrated, the separated sand being leached, whilst the slime was mechanically agitated, the treated pulp being decanted as on the Rand. The leaching process was very protracted, and required a plant of very large capacity, and it was found that to obtain satisfactory results it was necessary to grind the ore very fine—the finer the better—owing chiefly to the fineness of dissemination of the principal silver-bearing mineral, argentite, throughout the gangue. The result of this was that the percentage of slime to be cyanided was gradually increased, whilst that of the sand was at the same time lessened, until in some plants the whole of the ore was "slimed."

The treatment of silver-bearing slime pulp in mechanical agitators has been always an expensive and unsatisfactory method. With the larger sizes of vats the stirring gear is generally of the clumsiest construction, due to the conditions imposed upon the apparatus, and a great waste of power occurs. Also in most cases after a few hours' agitation—less than six—unless fresh air is introduced into the pulp, the cyanide action on the silver-bearing pulp (which requires more oxygen than a gold-bearing pulp) ceases.

Early in 1902 F. C. Brown, engaged in the treatment of silver-gold ores on the Komata Reefs Mine, in New Zealand, in conjunction with S. D. McMiken and others, began experimenting with a cone bottom vat, 7 ft. 6 in. diameter × 37 ft. deep, for agitating slime by means of compressed air, the cone having an angle of slope of 55°. Later on in the same year he agitated charges in conical bottom vats, 10 ft. × 39 ft., the cones having an angle of slope of 60°. In 1903 he attempted to agitate finely-ground sand, of which 93% passed a 200-mesh, in a 10 ft. × 39 ft. vat with a jet of air only, as before, but found that the air moved up through a channel in the centre of the vat, and the bulk of the sand received very little agitation. Early in 1904 the central air lift tube was successfully used in these tall vats, and the arrangement became a standard one for agitating finely-ground sand, and concentrates as well as slime.

As the result of much experimenting, it was found that :—

1. The higher the vat compared with the diameter the less was the power required.
2. The consumption of cyanide was less in tall than in shallower vats, owing to the fact that the necessary quantity of compressed air (and therefore the amount of carbonic acid passed into the pulp) is less per ton of ore in the former than in the latter.
3. That although good agitation could be effected with slimy material in shallow vats, yet, as the coarseness or the density of the material, or the specific gravity of the pulp to be agitated, increased, the ratio of height to diameter should be increased likewise.
4. The diameter of the central air lift pipe should be 1¼ in. for each foot of diameter of the vat.

Incidentally it might be noticed that the cyanide consumption with these agitators was found to be slightly less when potassium cyanide was used than when sodium cyanide was used.

The following table gives the average results of experiments made by F. C. Brown in New Zealand, and averaged over several months each, by agitating different classes of material in different sizes of vats :—

Size of vat.	Material agitated.	Tons (dry) of material.	Cub. ft. of free air required per minute.	Pressure of air required lb. per sq. in.	H.P. required.
10′ × 40′	Sandy slime	30—40	10—13	25	¾—1
7′ 6″ × 37′	Slime	15	4—6	22	⅓—½
7′ 6″ × 37′	Finely ground concentrate	15	15—20	26	1½—2
13′ × 55′	Slime	75	32	32	1¼

These tall vats, the use of which has spread to other parts of the world, are known in New Zealand and Australia as Brown and McMiken vats, in Mexico and U.S.A. as Pachuca tanks, and in other places generally as Brown vats or Air Lift vats. They were introduced into Mexico early in 1907 by Albert Grothe at the Hacienda San Francisco, at Pachuca, for agitating an all-slimed silver ore, and with such successful results that their use has since become general in that country, very few silver-treatment plants being without them, the mechanical agitator having been practically superseded. Now in Pachuca alone there are nearly 80 large vats of this kind. A large number is also in use in U.S.A.

Fig. I. is a plan of Grothe's design of the vat he first installed. It will be noticed that it consists of a cylindrical shell stiffened by horizontal angle iron rings and terminating at the bottom in a double angle iron ring, which rests on a substantial concrete foundation. The cone, which

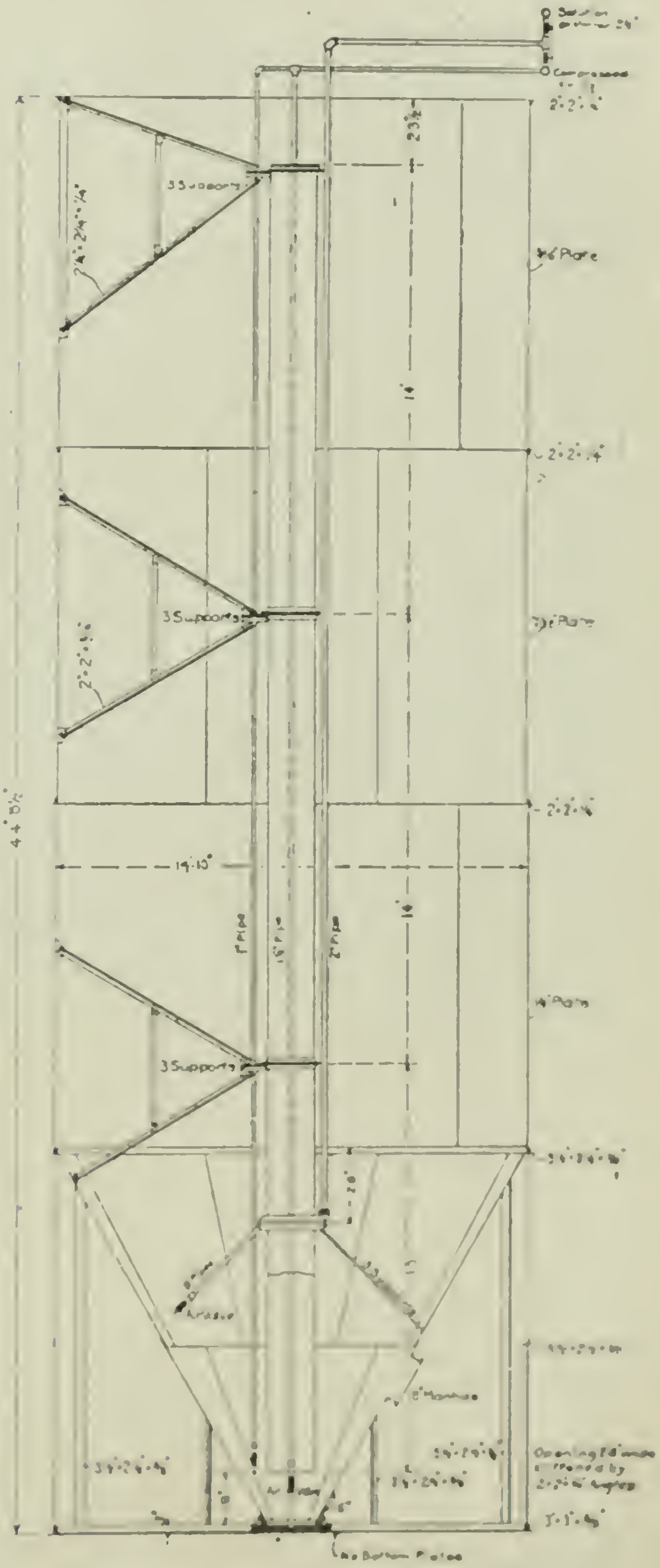

Fig I.

has an angle of 60°, sits upon a heavy cast-iron bottom, which, it will be noticed, supports one-third of the weight. A door-way in the lower part of the shell, having an opening 3 ft. 6 in. high × 2 ft. 6 in. wide, gives access to the space inside the shell below the cone to a manhole and a discharge cock. Openings in the shell give light to the space below the cone. Fig. II. is a photo by Edmund Girault of two of these vats at the San Rafael mill, Pachuca, which shows these openings. The vertical joints of the shell below the cone are stiffened by angle irons. The vertical air lift pipe, in the centre of the vat, has its ends usually 18 in. below the top and 18 in. above the bottom of the vat. The central air pipe is generally the only one used in agitation; but the vertical air pipe external to the air lift pipe is useful in keeping a clear channel through the pulp during the process of filling a vat. When a vat is full it usually only takes a few minutes after the central air supply is turned on to get the whole contents into agitation. The air required at first is, however, much above the normal requirement, generally double. As soon as the contents of the vat are in circulation and the cone is free from banked-up material the air supply should be reduced, to avoid a waste of air, until the cone just keeps clear. This is ascertained by feeling the surface of the cone with a lead weight attached to a string.

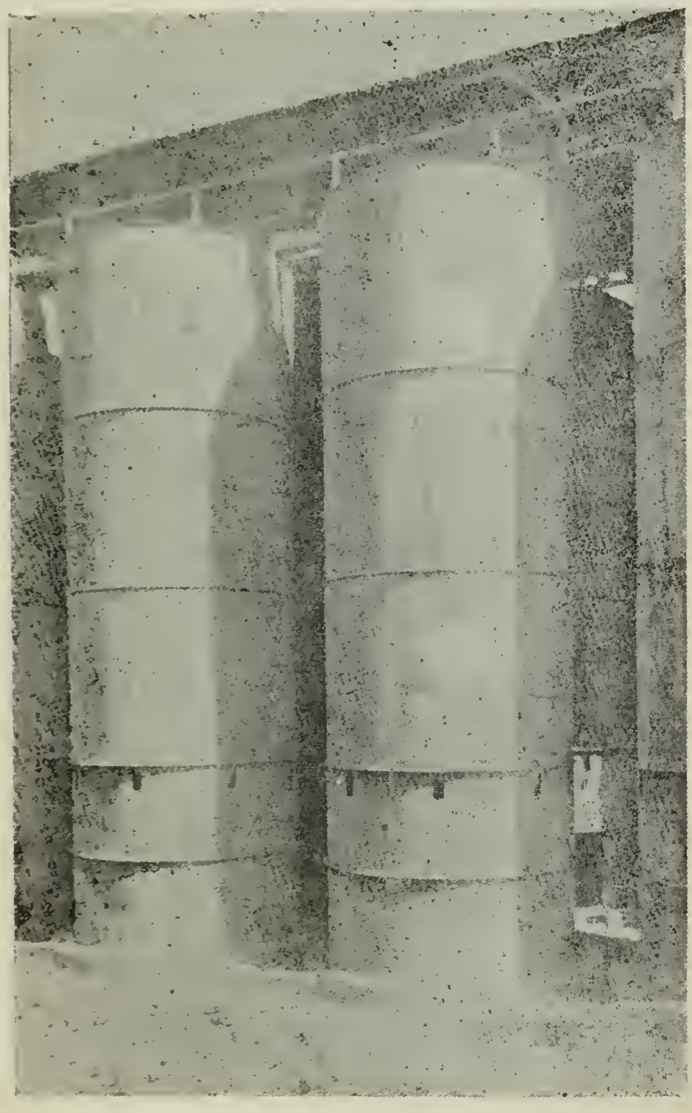

Fig II.—Brown Vats at San Ra Fael Mill, Pachuca.

The eight radially arranged air pipes are useful to start the agitation of a charge which has stood for some time. After several weeks' stoppage it is usually possible to get a charge into perfect agitation within an hour. (It might be noted here that, in the case of the stoppage of a large diameter mechanical agitator for a short time only, it is necessary to dig out its contents to free the paddles).

In the case of ores which have a tendency to scum when pulped, it is best to fill the vats not so full as usual, so that the falling pulp can be made use of to break up the scum.

For the air valve the delivery pipe is usually perforated in several places all around for a length of about 4 in., and a short length of rubber tubing of good quality, strengthened at each rim by an extra ring of rubber, is slipped over the pipe, the valve behaving similarly to that of a bicycle. This rubber sleeve is practically the only thing that ever needs renewal, and the repairs on this type of agitator are almost nil.

The air from this valve in the air lift escapes into the pulp in a thin spray around the air pipe, and mixes intimately with the pulp. Large bubbles at the discharge of the air lift occur when the agitation of a charge is commencing, but in normal working these generally indicate a waste of air. When the air is properly delivered to the pulp, the latter is saturated with it, and it is this air in a fine state of division, and not that of the large bubbles, that assists the chemical action.

It has been urged against the air lift agitator that an unnecessarily large consumption of cyanide occurs with its use, due to the carbonic acid in the air, and also that atomized oil from the compressor, introduced into the pulp, interferes with the chemical action. With regard to the former, the air certainly could be deprived of its carbonic acid before compression by means of lime, but nobody so far seems to have thought it worth while to do this. In the case of the latter, with ordinary care, the amount of oil atomized is so small that it is negligible. Compressed air of sufficient pressure and free from oil might be produced by means of turbine blowers worked in series, and the economy of this method of production of compressed air, for agitation purposes, might be looked into.

The consumption of cyanide in a Brown vat is not usually great in practice; in fact it is usually less than with other agitators when treating the same pulp. This is mainly due to the fact that a charge of slime is much more rapidly treated in a Brown vat.

At Pachuca, almost side by side, are two plants, the Loreto mill and the Hacienda San Francisco, both treating "slimed" and concentrated ore, of almost identical character and value. In the former are 30 ft. mechanically agitated vats, each taking a charge of 57 short tons in a thin pulp, whilst in the latter, as mentioned above, are Brown tanks 15 ft. × 45 ft., operating upon charges of 112 short tons in a thick pulp. The cyanide consumption per ton in the former mill is greater than in the latter, whilst its power consumption is at least ten times as much.

Some years ago at the Waihi Grand Junction Mine, N.Z., F. C. Brown made some comparative experiments with different types of agitators. The agitators were:

(A) Shallow vats with ordinary stirring gear.
(B) Brown vats, 13 ft. × 55 ft.
(C) Brown vats, 7 ft. 6 in. × 37 ft.
(D) Shallow vats each with a central tube and a high speed screw propellor. (These last are known in Australasia as the A.Z. agitators: the Hendryx is of the same type.)

In (A) and (B) for the same period of two months the same class of slime pulp was agitated, the average consumptions of sodium cyanide being respectively 2·6 and 2·4 lb. per short ton. In (C) and (D), also for a period of two months, the same class of ground concentrate pulp was agitated, the average sodium cyanide consumptions being 2·4 and 3·1 lb. per short ton.

In August, 1908, J. Leslie Mennell, of Mexico City, first proposed the series or continuous system of treatment in Brown vats as an improvement upon the ordinary intermittent treatment system. In the latter system, which until recently has been that commonly in vogue, the agitators are each worked on the charge system, each vat being filled separately, its contents being agitated and then discharged. In the former system, now largely superseding the intermittent, he proposed to feed the pulp into one vat only, the head of a series of vats, and connect the successive vats so that the pulp flowed proportionately and at the same rate through the series. In conjunction with A. Grothe, in a series of small vats 18 in. × 54 in. exactly on the Brown model he made a series of tests to see if the system would work in practice. The vats were placed on the same level. Some difficulty was encountered at first in finding a suitable form of connection between the successive vats which would prevent segregation of coarser material in the vats at the head of the series. After some trials the connections between the vats were arranged as shown in Fig. 111., and a uniform quality of flow from vat to vat secured. Then a silver ore somewhat complicated by base metals and requiring a long time of agitation for commercial treatment was selected for experiment and finely ground. Two experiments were made upon this ore, one with a single vat, which was charged with 154 lb. of ore and 220 lb. of solution, and the other with four vats in series as shown in Fig. 111. In the latter case ore and solution were fed into the head vat every fifteen minutes for eight days, at the rate of 154 lb. of ore and 220 lb. of solution per 24 hours. After five days, when the plant seemed to be working regularly, samples were taken from all the vats. Sampling was continued to the end of the eight days and values were found to be uniform. The

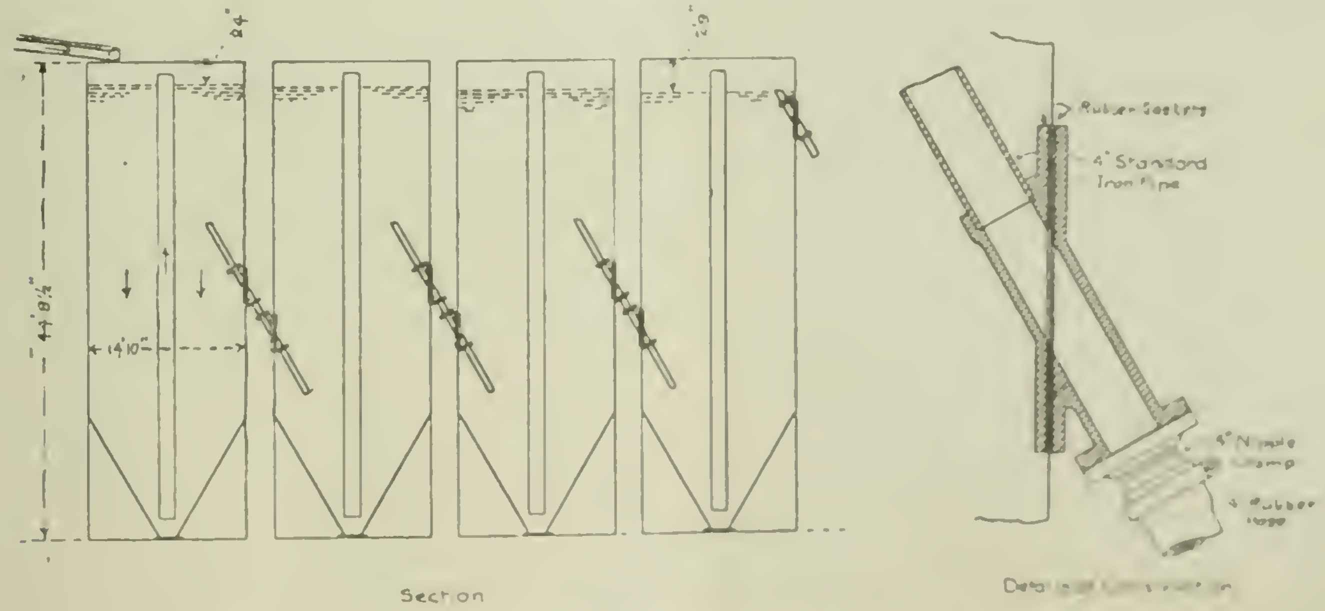

FIG. 111.—Grothe and Carter Vat Connection for Continuous System

contents of the single vat were also agitated for 48 hours, and were sampled at intervals.

The results of the two experiments are here shown side by side in tabular form :—

Hours of Agitation.	Strength of Solution. %KCN.		Cyanide Consumption. Lb. per Short Ton.		Silver Value. Ounces per Short Ton.		% of Extraction.	
	Single Vat.	Series Vats.	Single Vat.	Series Vats.	Single Vat.	Series Vats.	Single Vat.	Series Vats.
0	·39	·39	0	0	10·82	10·91	—	—
12	·26	·30	4·31	2·84	3·66	4·26	66	58
24	·20	·26	5 96	4·09	3·15	3·56	69	68
36	·13	·20	8·27	5·96	2·86	2·95	71·2	71·3
48	·12	·18	8·49	6·50	2·77	2·34	72·7	77·3*

In the above sets of figures those of the series vats refer to the effluents from the four vats respectively. These experiments and the advantages of the new system were described by Mennell in the *Mexican Mining Journal* of February, 1909. Briefly put, the advantages of the continuous system are that :—

1. The time of emptying and filling vats can be employed more usefully in continuously agitating the pulp.
2. Individual attention to the operation of each vat being no longer necessary, all that is required is a watchman to see that all the vats are properly supplied with air, and that there is no wastage of air.
3. The cost of discharging the individual vats is done away with.
4. The loss of head is practically saved, the level of discharge being only about three feet below the top of the last vat. Thus the treated pulp can flow by gravity to the filter plant instead of being pumped to it, as is usual with the intermittent system.
5. In case a sudden extra demand is thrown upon the cyanide department of a mill there is more chance of the residues from a continuous system being lower than those from an intermittent system.

Soon after the publication of the results of the Mennell-Grothe experiments, several mining companies in Mexico arranged to try the system practically, amongst them the Esperanza G. M. Company of El Oro, State of Mexico. This company had formerly six vats in use on the intermittent system, each 14 ft. 10 in. × 44 ft. 8½ in., for treating charges of 90 short tons of finely ground battery sand (without true slime). The pulp, consisting of 1 part of sand to 1·8 of solution, was agitated for 33 hours, this including half the time of filling and discharging, viz., 5·1 hours. If the pulp had been treated continuously, the time of agitation would have been increased to 38·1 hours, an increase of 15·3%. This loss of time in agitating and the great expense of the elevator system incurred in discharging the vats made the company decide to adopt the continuous system, and the vats were connected by 6 in. pipes. The result was quite successful, the transfer of pulp from vat to vat being perfectly regular, whilst, in addition to the saving of labour and cost of elevation, there was an actual saving of cyanide of $\frac{1}{20}$ lb. per ton, while the increased extraction of gold was 1·3% and of silver 1·5%, the original gold and silver contents being 0·196 oz. and 2·34 oz. per short ton. This modified plant has been in successful operation since January, 1910.

Fig. III shows the method of connecting up a series of vats for the continuous system in the way designed by Grothe and Carter, of Mexico City. The connecting pipes are at an angle of 60°, at which angle they are kept clean. The flexible pipe connection outside the vats is introduced to do away with the effect of vibration and unequal expansion.

It will be noted that a small loss of head, due to friction, occurs with a series of vats, the level of pulp in each vat being slightly lower than that of the vat immediately preceding it. In the six vats of the Esperanza mill the friction drop from vat to vat was about 6 in., or a total drop of about 30 in. for the six vats : in the last vat, however, the requisite portion of the pulp, discharged by the air-lift, was cut out for final discharge by a box, 12 in. × 12 in. × 24 in. placed above the level of the pulp in the vat : the loss of head with the six vats was thus reduced to 12 in.

In designing an agitation plant on the continuous system it is advisable to experiment upon the ore in a single vat first. By plotting the percentage extractions, say at the end of every hour of treatment, it will be seen when it would cease to be profitable to further agitate the ore. On dividing the curve by a series of equidistant lines, parallel to the axis of the curve representing the extractions, into as many spaces as the number of vats it is intended to use, it will be seen what extraction in each vat may be expected.

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* This was confirmed by the bulking and re-assay of the whole of the residues.

The air lift system of agitation is not only valuable for the cyanide treatment of current material, but it is also an excellent piece of apparatus for the preparation of accumulated slimes for their subsequent treatment by cyanide.

In the *Journal* of this Society of July, 1897, Dr. W. A. Caldecott, writing upon the chemical conditions existing in accumulated slime, showed how the neutralization of cyanicides in it would be effected by their gradual oxidation through exposure to air, and in March, 1898, in conjunction with John Kelly, he took out a Transvaal patent, No. 1559, for an air lift apparatus for the agitation and aeration of slime pulp or other gold-bearing material. This method of sweetening accumulated slimes by means of atmospheric air is utilized on the Geldenhuis Deep G. M. Co.'s plant and also on that of the Luipaard's Vlei Estate & G. M. Co.

The first Company in the Transvaal to use the air lift principle in the treatment of accumulated slime was the Nourse Mines, Ltd., a flat-bottomed vat 30 ft. in diameter and 12 ft. deep being used. The Geldenhuis Deep G. M. Co. to oxidize accumulated slime uses two conical bottomed vats each 20 ft. in diameter and 20 ft. deep, the vertical depth of the cone being 5 ft. These vats have air lift pipes 16 in. in diameter, in the lower portions of the vats, of one-half the depth of the vats only. Pipes were formerly used in length equal to the full depth of the vat, and then afterwards tried of shorter length with much more satisfactory results. The slime is pulped with 2 parts of water in a knife box mixer and then pumped into the air lift vats, a charge for each vat being 50 tons of dry slime. Each vat requires 50 ft. of free air compressed to 10 lb. pressure per minute.

At the plant of the Luipaards Vlei Estates Co., the accumulated slime after being suitably pulped (lime and lead acetate being then added), is pumped into a conical bottom vat of diameter 30 ft. and total depth 30 ft., fitted with an air lift pipe 18 in. in diameter, 26 ft. high and 22 in. above the bottom of the vat.

The air supply to the air lift is delivered through a ¾ in. air pipe, centrally placed inside the lift pipe. This supply is utilized only when the vat is full and its contents are in agitation. But, when the vat is being filled or discharged, the air lift is not operated, the pulp being agitated by air supplied through four ½ in. air pipes outside the air lift tube. The pulp receives a total aeration of 22½ hours, made up as follows:—Filling 9 hours (partial aeration), air lift agitation 6½ hours, discharging 7 hours (partial aeration).

The charge of the vat is 150 tons of slime and 300 tons of water, and this charge is probably the largest single charge of slime that is agitated in the world. The vat requires about 85 cub. ft. of free air compressed to between 25 lb. and 30 lb. per sq. in. The small amount of sand, which is always present in accumulated slime, is drawn off from the bottom of the cone, as in the separatory funnel of the laboratory, and sent to the tailing wheel of the cyanide plant.

One of the most useful parts of the treatment vat is a conical diaphragm surrounding the air lift pipe, whose horizontal base is 3 ft. 1½ in. in diameter and placed 2 ft. 5½ in. above the bottom of the vat. This leaves an annular space about 9 in. wide between the diaphragm and the sloping side of the vat, through which the pulp passes to the air lift. This arrangement entirely prevents the banking up of material around the mouth of the air lift, and reduces the chance of fine sand collecting on the sloping side of the vat above the diaphragm. In general it enables a vat of larger diameter, and consequently of less capital cost per ton treated, to be used than without its use.

The accumulated slime after aeration treatment is pumped to the current slime plant for cyanide treatment. The following average figures from a three months' run show such highly satisfactory results as to warrant more attention being paid to this method of treating accumulated slime:—

Value of Charge... ...	2·160 dwt.
„ „ Residue ...	0·278 „
Percentage Extraction ...	88·7
Lime consumed per ton .	18·079 lb. at 50 % CaO
Potassium cyanide per ton	0·233 lb.
Lead acetate per ton ...	0·121 lb.

The total working costs in connection with the treatment of this slime is 3/8·1d., which includes 1/5·3d. for collecting and pulping.

The Brown vat is also successfully used for the treatment of black sand. At the plant of the Simmer & Jack Proprietary Mines, Ltd., an air lift vat, 13 ft. in diameter and 18 ft. deep, with a central pipe 16 ft. 6 in. long and 10 in. in diameter is used for treating 12 ton charges of black sand. The black sand is previously finely ground in a small tube mill. After the pulp of sand and water has been got into circulation, lime is added, and later on lead acetate also. Agitation is continued until all soluble sulphides are decomposed, a period of about 16 hours. Cyanide solution is then introduced under pressure into the bottom of the charge, and the solution of the gold is commenced, periods of agitation, settlement and decantation following one another alternately until the value of the decanted solution indicates completion of the treatment. After each decantation, a centrifugal pump, connected with the bottom of the vat, and discharging externally into the top of the vat, is used to withdraw any sand which may have settled at the bottom of the cone below the air lift and put it into general circulation again.

The head samples assay about 10 oz. gold per ton, and over 97% extraction is obtained on this value of sample with a cyanide consumption of only 2 lb. per ton.

The air lift has been in use for the elevation of various liquid materials since 1892 : in Western Australia it was used for pumping battery tailing in 1902 ; and, although F. C. Brown, as mentioned before, introduced the central air lift tube into his tall agitation tas in 1904, Mr. A. F. Crosse actually used the air lift on the Rand for the experimental agitation of slime in 1903.

The following is a list of published articles, bearing directly or indirectly upon the subject of the cyanide treatment of pulp :—

The smaller sizes of these mechanical agitators are often arranged so that the stirring gear can be raised out of the pulp, when a stoppage is imperative.

The Blaisdell agitator is one with a stirring paddle gear, driven by electro-motor and supported on a moving bridge, which can be moved over a row of agitation vats. The stirring gear can be raised out of the vat so that it can be transferred to another of the series of vats, the agitation of charges in this system being intermittent. A centrifugal pump on the bridge, also motor driven, is used to withdraw the pulp and then aerate it by spraying it back into the vat again. This system is very expensive to instal and operate, and is not suitable for a pulp that requires continuous agitation.

The Hendryx agitator has a conical bottom vat and a central tube, through which the pulp is raised by a high speed screw propellor, driven by a vertical shaft in the tube. It is expensive to operate, 18 h.p. being necessary to agitate a pulp containing 100 tons of solids. (Ref. No. 8.)

Ref. No.	Title of Article.	Author.	Publication.	Date Published.
1	Agitation by Compressed Air	F. C. Brown	*Mining & Scientific Press*	Sept. 26, 1908
2	Continuous Cyanide Treatment	J. Leslie Mennell	*Mexican Mining Journal.*	February, 1909
3	Cyanide Lixiviation by Agitation	Walter M. Brodie	*Engineering and Mining Journal.*	April 3rd, 1900
4	Treatment of Ore Slime	Andrew F. Crosse	*The Journal of The Chemical, Metallurgical and Mining Society of South Africa.*	November, 1909
5	Hydraulic Agitation in Cyaniding		*Mining & Scientific Press.*	April 20, 1910
6	Cyanidation of Concentrate	A. E. Drucker	*Mining & Scientific Press*	March 19, 1910
7	Continuous Pachuca Tank Agitation at the Esperanza Mill	M. H. Kuryla	*Mexican Institution of Mining & Metallurgy*	April 1910
8	Methods of Pulp Agitation	Lloyd M. Kniffin	*Mining & Scientific Press.*	June 4, 1910
9	Notes on Cyaniding in Pachuca Tanks and the Continuous System	A. Grothe	*Mexican Mining Journal.*	August, 1910
10	The New Esperanza Mill and Milling Practice	Chas. Hoyle	*Mexican Mining Journal.*	August, 1910
11	A new Cyanide Device	Lee Fraser	*Mining & Scientific Press.*	Oct. 15, 1910
12	A Modification of the Pachuca Tank	Amos J. Yager	*Mining & Scientific Press.*	Dec. 22, 1910
13	A Cyanide Plant constructed of Masonry	John H. Eggers, jr.	*The Pacific Miner.*	Jan., 1911
14	Mechanical and Air Agitation for Slime Treatment	A. W. Warwick	*The Mining World.*	Jan. 1911

Reference has been made before to the use of mechanical agitators, viz., those in which the pulp is stirred by rotating paddles. In these it is not possible to use a thick or very sandy pulp, and in the majority of them air is introduced into the pulp generally by an air jet at the bottom of the vat, but, in some cases, in small jets along the agitator paddles, it having been delivered through the hollow shafting of the agitator gear.

In the Just system of agitation recently introduced and in use in Guanajuato, Mexico, air from a blower is introduced through a special porous brick floor in the bottom of a vat, agitating 9 ft. of pulp above it. It will probably be found in practice that the bricks will become choked with lime salts, and, if so, cleansing would be troublesome.

Mr. A. F. Crosse's method of decanting an overflow from an agitation vat seems to have been useful to inventors.

In the Paterson vat this principle is made use of as well as that of the Brown vat, the decanted solution, free from sand, being passed outside the vat and drawn through a centrifugal pump. This delivers the pulp again to the bottom of the lift pipe, taking the place of the air of the Brown vat. The makers give 4½ h.p. as the probable power required to agitate 50 tons of (dry) slime in a 15 ft. × 45 ft. vat. By way of comparison it may be noted that in a Brown vat of the same size at the Hacienda San Francisco, at Pachuca. 112 short tons of (dry) "slime" are agitated with about 2 h.p. (Ref. No. 5.)

Mention has been made above to the use of a shortened air-lift pipe in the vats of the Geldenhuis Deep G.M. Amos J. Yager (Ref. 12) has recently described some experiments made by the modifying of the air lift pipe of a Brown vat, as shown in Fig. IV., No. 1 showing the normal arrangements. He obtained better extractions on silver ore successively with No. 2, No. 3 and No. 4 arrangements, and finally adopted No. 4 as standard. A. W. Warwick (Ref. 14) also alludes to the shortening of the air-lift pipes in Brown vats at the Stratton's Independence Mill, at Cripple Creek, to two-thirds of their former length. The result of shortening of the air-lift pipe is the spreading out of the rising pulp above the pipe as is shown diagrammatically in Fig. V., and consequently more use is made of the air from the air lift ; but it is doubtful if this modification would be used satisfactorily in the continuous system of agitation.

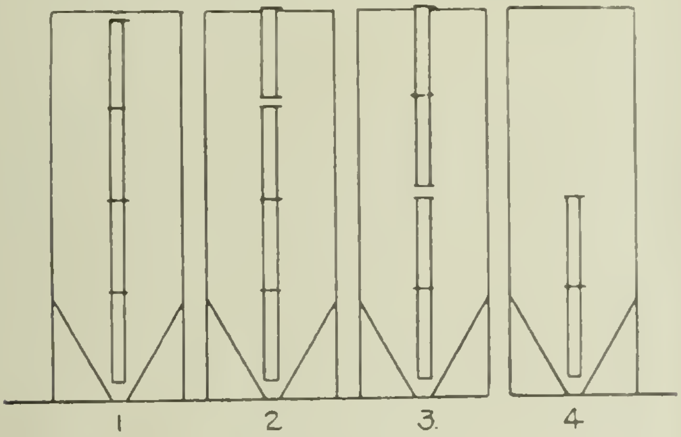

FIG. IV.

Vertical circulation pipes in a Brown vat, as shown in Fig. VI., with their upper orifices, which are protected by goosenecks, just below the surface of the pulp, and lower orifices near the mouth of the air lift, were suggested by W. M. Brodie. Their use was mainly the prevention of choking by heavy sand around the opening of the air lift, and with them less air is required to start agitation after a shut-down. (Ref. No. 3.)

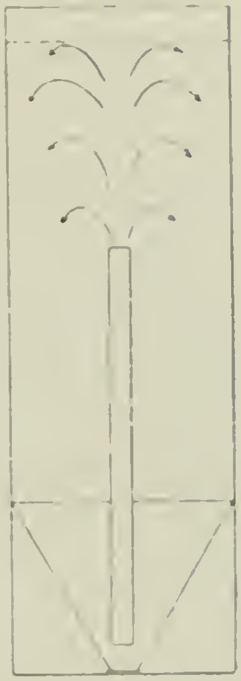

FIG. V.

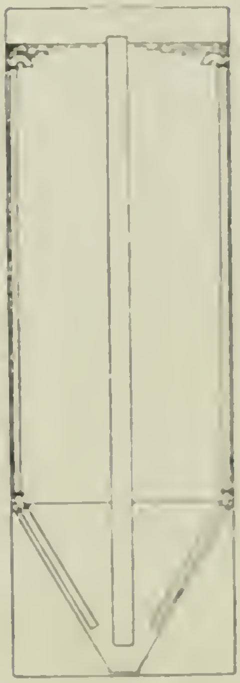

FIG. VI.

The use of air lift vats built of masonry at Zambuna, Mexico, each with two air lifts and continuous decantation and constructed as shown in Fig. VII. has been described by John H. Eggers Ref. No. 13).

In conclusion, I beg to thank Mr. C. D Leslie, Superintending Engineer of the Consolidated Goldfields of S.A., Ltd., for permission to pub-

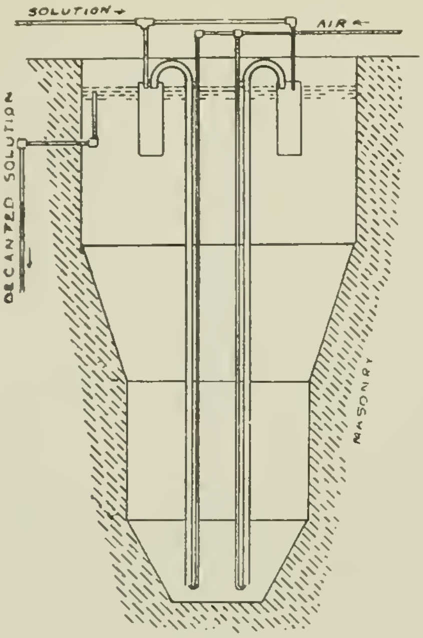

FIG. VII.

lish the working data of the Luipaard's Vlei Estate and the Simmer and Jack treatment plants.

The President: This is another very good paper indeed, and I am sure you will accord a very hearty vote of thanks to Mr. Allen. I believe it is his maiden effort in this Society, but you can see he is already an expert, especially at diagrams.

NOTES ON THE MATTE ASSAY.

(*Read at December Meeting, 1910.*)

By L. J. Wilmoth, A.I.M.M., M.Am.I.M.E.

DISCUSSION.

Mr. T. Graham Martyn (*Member*): Mr. Wilmoth has made a very useful contribution to our methods of assaying cupriferous materials, but referring to the literature of the subject he says, "I . . . . as yet have found no mention of matte except as an unintentional product in the assay of such material as pyrite. In all cases the production of a matte under these circumstances is condemned, and rightly so, as it is always accompanied by unreliable results."

In my opinion the last clause is too sweeping as I have found that provided the slag is sufficiently acid and fluid to permit of clean separation of the matte from it one gets, if anything, a better recovery of precious metals from the slag when the lead button is sulphury, or a matte is formed, than when a soft button is obtained. The reasons why the sulphury button is objected to, are, I take it, first because a further operation is required for purifying the lead before cupelling which requires time and attention and involves some risk of loss; and secondly because if one is not on the look out for it there is great risk of losing part of the brittle lead or adherent matte in cleaning off the slag. It is not at all surprising to me therefore to find that, when care has been taken to produce a slag, a matte and a lead button easily separable from each other, the slag has been unusually clean from precious metal and the matte has carried no more or perhaps less than would ordinarily have been lost in the slag. The surprise to me is that it has not come under one's notice before, as an alternative to the unsatisfactory or troublesome methods in use for the assay of cupriferous materials for gold and silver. The author's results should have led him I think to carry his method a stage further, and I would suggest the retreatment of his matte by his own method instead of his resorting to scorification which in my own experience confirmed by his results, is much less reliable.

Taking the first results given in his paper and comparing the matte assay with the scorification assay a little more in detail than he has done:—

Assume that the result of the "combination" assay gives the true value, viz., 10·49% then the uncorrected matte assay recovered 99·24% of the fine gold, leaving 0·76% in matte and slag, whereas the scorification method only recovered 98·29%, leaving 1·71% in the slag or lost by volatilisation. The above 0·76% was partially recovered by scorification; taking the actual weights instead of percentages out of 7·6 mg. lost in the matte assay, 2·8 mg. were recovered from matte by scorification and 0·17 mg. from the slags. Had he treated this matte as in his later assay of copper matte I am of the opinion he would have obtained a better recovery and might very possibly have improved on the "combination assay" result. I hope that before he replies to the discussion he will find an opportunity of trying the experiment. It would probably improve the gold assay in both stages to add silver before hand if not already present in excess, say, to the extent of at least three times the gold. I have usually obtained a higher result by running down rich sulphide ores with excess of litharge or red lead, a little soda carbonate and silica or borax, if necessary, than by scorification, the latter being a method which in consequence I avoid except for reducing the size of buttons or for assaying rich alloys. It appears that the author has devised a method which may render scorification entirely obsolete and the results of experiments on the lines indicated above would be most interesting.

NATIVE FOOD SUPPLIES AND THEIR QUALITY.

(Read at January Meeting, 1911.)

By F. W. Watson, B.Sc. (Lond.), F.I.C., F.C.S. (Member).

DISCUSSION.

Dr. J. McCrae *(Member)*: With Mr. Watson's general contention that the food stuffs supplied to compounds should be subjected to control by examination, I have the greatest sympathy, but with his idea of purchasing foodstuffs (even excluding meat and fresh vegetables) according to proximate analysis, I cannot agree. The author says that up to the present very little has been done to test the articles of diet supplied to compounds. This statement reflects on those responsible for the conduct of compounds, and to some extent on the Inspectors of the Native Affairs Department. I cannot speak for the compound authorities, but I may say that samples are frequently taken by officers of the Native Affairs Department for examination. I think that if the author were fully conversant with the facts he would modify his view that little has been done; indeed, he would find that many people are alive to the question of the quality of the foodstuffs supplied to native labourers.

The author quotes two analysis of maize, and for comparison, one analysis of flour. These analyses do not teach us anything new, and from such proximate analyses it would be difficult to deduce which is the most valuable foodstuff. It is probable that the whole maize might work out as of the highest monetary value if the valuation were made on the amounts of fat, protein and carbohydrate contained in each foodstuff; but surely a great advantage is achieved by removing the large-piece fibre (in the husk) from the whole maize, even if in the process fat-content is simultaneously reduced owing to removal of the embryo. It is for reasons such as this that I regard it as inadvisable to purchase foodstuffs in general on a proximate analysis. In purchasing cyanide, it is only cyanide which is wanted—anything else present is useless or harmful. With maize meal the case is entirely different: the carbohydrate is the valuable constituent, the protein and fat, too, are useful; but, for its mechanical action, may not the fibre also be of appreciable value (provided it is not in large coarse pieces), and who would be so bold as to assert that the mineral matter is of no use? I suggest that what is of use in the examination of flours and meals is to ascertain that what is required is supplied, and supplied without the addition of other material, and that the flour or meal is in a condition fit for human consumption.

The author tells us that he has not yet examined any samples of maize meal on which any suspicion could rest. I can only think that his experience in the examination of maize meal found in compounds is not extensive. Perhaps in time the author may come across some mouldy weevil-eaten samples!

I would like here to make it perfectly clear that I am no opponent of proximate analysis: on the contrary, I would advocate the frequent analysis of foodstuffs—not in order to decide the price to be paid, but to ascertain that the foodstuff is not overloaded with water (for even on the Rand, water is a cheap adulterant), and that it approximates to the usually accepted average composition.

The author quotes five analyses of ground nuts. Clearly samples D and E, which were adulterated, should be rejected: starches and salt are cheap adulterants for ground nut, and the analyses quoted show, as the author rightly says, the necessity of testing supplies in order that only pure products may be accepted. I cannot, however, agree that the price of ground-nuts to be used as food should be regulated by the fat content, freshness, freedom from disease (plant), and general condition are important factors. Although a definite amount of ground nut may be specified in a diet scale, it is not essential for the human frame that precisely this quantity down to the veriest grain be consumed. The case is wholly different when ground-nuts are purchased for oil extraction: here it is the oil, and the oil only, which is wanted, and the oil maker's return depends entirely on how much oil he obtains for re-sale (or for manufacturing processes). Would the author advocate that the price of milk for ordinary use be regulated by its fat-content? The butter maker in buying milk purchases on fat-content, because it is the fat which he wants, and it is the fat which he re-sells.

The author makes reference to ground nut cake obtained after expression of the fat. This is in itself a valuable foodstuff, because it contains all the original protein, and the fat is never fully expressed. The following analysis of a locally produced ground nut cake will show how useful such cake is:—

Water	... ...	8·6	per cent.
Protein	... ...	42·6	„
Fat	... ...	8·4	„
Fibre	... ...	4·3	„
Carbohydrate	...	31·0	„
Ash	... ...	5·1	„

Clearly a cake so rich in protein could be employed for increasing the protein ratio in a diet, and if the nuts are cold-pressed the digestibility of the protein will probably not be reduced.

In reference to cocoa the author quotes Dr. Hutchison. The quotation appears to me to be inapposite. The "impossible quantity" is not of cocoa, but of water. I do not think that anyone imagines that they derive any large amount of nutriment by drinking a cupful of the beverage, yet for the quantity taken (and surely that must be strictly borne in mind) cocoa is, undoubtedly, a valuable foodstuff.

In his section on Kafir beer the author says that "three per cent. of alcohol by weight is the maximum allowed." It is regrettable that the Ordinance (No. 32 of 1902) is not so specific, for it only states "three per cent. of alcohol," and leaves the question of how the alcohol is to be measured quite open. Personally, I would prefer that the measurement be by volume, because I think that that form of statement has more significance, and because alcoholic beverages are both purchased and consumed by volume.

Evidently, the Kafir beers which have come under the author's notice have been very mild. I have been less fortunate! It has been my lot to examine Kafir beers from many compounds, and I have frequently found as much as 8 per cent. by volume of alcohol, and even 11 per cent. has not been unknown. I should be glad to know if the author followed the increase in the acidity of that sample of Kafir beer in which he made observations on the increase of alcohol during 21 days. It would be of interest to know why fermentation ceased when the alcohol had reached only 3·84 per cent. by weight. One might hazard the guess that acetification was proceeding almost as quickly as fermentation; possibly the acetic acid formed had inhibited the growth of the yeast, or possibly there was some interference with the saccahrification of the starch. There can be no question that Kafir beer ordinarily has the potentiality for becoming as strongly alcoholic as unfortified wine. So long as acetification is prevented, fermentation will normally proceed until the growth of the yeast is inhibited by the alcohol formed, which takes place when the proportion reaches approximately 15 per cent. by volume. I think, however, that the author is very near the mark when he says that Kafir beer remains below the maximum allowed by law (even assuming that the law is to be read as 3 per cent. by volume) up to four days after commencing to brew.

The question of the examination of the foodstuffs employed in compounds is, undoubtedly, of vast importance, and I trust that the author's communication will do much towards directing attention to the subject. Such examinations by leading to the detection of spurious goods can result only in benefit to the native labourers, and it can do much for honest trade.

It would, perhaps, be of interest, and might stimulate investigation if the author could lay before the Society some statistics showing the extent to which adulteration of foodstuffs supplied to compounds is practised. I may say that, during the past five or six years, of the foodstuffs examined under Municipal Bye-laws in the Transvaal, the samples which have been found to be grossly adulterated, or have been condemned as unfit for human consumption, have varied between 15 and 20 per cent. of those examined.

The President: I am sure we should thank Dr. McCrae for his valuable contribution, and I am sure you will all agree with me that his laboratory note books are probably full of other data valuable to the mining industry which he might give us. I have a suggestion to make myself concerning the composition of mealies. Some two or three years ago the *Journal* published an abstract to the effect that the proteid of mealies (namely *zein*) is an unsuitable diet when taken by itself for human beings.* This may have something to do with the inefficiency of the Kafir we hear so much about.

Mr. F. J. Pooler (*Associate*): Mr. Watson's paper has opened up for discussion one of the most debateable questions with which chemistry deals, and while one is struck with the care and industry displayed in his analyses, it seems doubtful whether all such work, even where a fair standard of accuracy is aimed at, as is evidently the case here, has more than an academic value, and such remarks as I have to make will be not in the nature of criticism of results of Mr. Watson's work—such presumption, I hope, is not mine—but a general criticism of the chemical methods at present in vogue, which methods purport to give an indication of food-values of various food-stuffs.

One would ask of what importance is it to know the percentage of fats in various nuts unless one knows the average amount of energy derived from those nuts by an average human system? The human body is not a retort or a distillery, and while it may be extremely interesting to know the percentage of nitrogenous matter in maize, it is of little apparent practical importance without the additional knowledge that 1 gm. of such proteid matter represents so much actual flesh-forming power to the average man. Then the question arises, "What is the average man?" from a dietetic point of view, and it is of importance to know if 1 gm. of albumen forms the same quantity of flesh in a head worker as in a hand and muscle worker. (It is stated in some cases that this is so—has it ever been proved experimentally?)

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* F. G. Hopkins, *The Analyst*, Dec., 1906, p. 386; this *Journal*, Vol. vii., May, 1907, p. 386; also *Journal Chemical Society* (London), Abstract, 1909, ii., 594; 1910, i., 598, and ii., 625.

Again, of what practical value is "calorific values of foods," expressed in kilogram calories per 1 gm. of food-stuff burnt, unless it be first shown that in the human system the power of deriving benefit from the combustion of the foodstuffs is any way a function of the relative potential energies so determined! Quoting from "Hygiene Advanced." R. A. Lyster (M.D., D.P.H.), M.O.H. for Hampshire, I find the following on such determinations:—"The figures thus obtained are not of any great practical value, because the different food-stuffs have different uses, and are not mutually replaceable: moreover, they are not in all cases broken down in the body in the same way as they are experimentally in the laboratory." One might add: The question whether they are broken down in the body at all is an important one, and also, is the metabolism in the intestinal organs of the Kafir the same as that in the white man! This question is not, perhaps, as foolish as it seems, when we consider that the Kafir can live almost entirely on mealie pap, supplemented by a little fat, which is not generally accounted a perfect food by white authorities, and which is said to be unsuited to the needs of our Indian passive resisters.

Further, can it be shown that the human system is able to derive energy from, say, nut oil in proportion of 9·21, while an equal weight of sugar, digested by an entirely different process, gives to the system a quantity of energy proportional to 4·02 on the same scale! Again, it seems scarcely to follow that because the percentage of nitrogenous matter in mealie meal is found to be 10·28 and in pea nuts 29·69, that these values represent the flesh forming powers of the respective foods on the same scale: for, though in general the digestive processes will be the same, it is not yet apparent that the body derives equal benefit from similar food constituents which exist under different conditions and mixed with the different proportions of other constituents, all to be digested together in the various foods.

It seems as though the chemist, working alone, is rather wasting valuable time in attempting to fix food-values by analysis, for since the vegetarian and fruitarian differ from each other and from the mixed diet man in their ideas of relative food values on the human system, and all three differ from the cannibal, of what value is the fat-extraction apparatus and the Kjeldahl estimation except that (in so far as we can accept current ideas as approximately correct) the analyst is enabled to satisfy himself that certain foods are up to average standard. Probably before any considerable advance can be made in the science of dietetics—which at present is an art of guesswork indulged in by all with a delightful empiricism that would be amusing if its results were not so disastrous on the white races to-day—it will be necessary for the chemist and the physiologist to work hand-in-hand, the former analysing and classifying proteids, sugars, &c., into various sub-branches (for it is unthinkable that "nitrogenous matter" is sufficiently informative), while the latter feeds his subjects and measures gain in weight, heat changes, waste excreted, working capabilities generated, and so on, such as, I believe, is now being done in Europe and America. Then, working together, it may be possible to tabulate conclusions of real scientific value, but even such results can only apply to the "average system," where such exists, and the results as determined must in general be subjected to immense variations depending on the particular ferments that find most congenial ground for propagation in the digestive organs of the particular subject. But it is also necessary that food stuffs so experimented upon shall be of the same stock, for, as Mr. Watson has realised, values as quoted elsewhere differ greatly from those in use here. A comparison of Mr. Watson's results with those given elsewhere, shows considerable variations, some of which may be worth while noting, *e.g.*:

MAIZE.

Mr. Watson.		R. A. Lyster.	
Nitrogenous matter	9·51 %	Proteids ...	14 %
Fat	5·52 %	Fat	3·5 %,
Carbohydrates ...	70·64 %	Sugar ...	2 %
		Starch ...	65 %

If this deficiency in nitrogenous principles is characteristic of S. African mealies, the need for pea nuts or some other addition to a mealie meal diet is obvious (accepting current theories as correct) and the flesh-forming powers of our local mealie meal must be considerably less than those of the commonly accepted standard in Europe. It would be instructive to find out to what these fairly large variations are due, especially in the case of our local staples—it may have something to do with local manuring, or the absence thereof. The local maize, however, has an advantage as far as fats are concerned.

A similar variation is noted in the case of cocoa, thus:—

	Mr. Watson (C).	(E).	R. A. Lyster	
Water ..	3·28 %	1·36 %		3·5 %
Fat ...	24·73 %	10·61 %		50·0 %
Starch ...	7·10 %	20·55 %		13·5 %
Nitrogenous matter ...	18·81 %	17·87 %	Proteids ...	15·5 %
			Theobromine	1·6 %

The differences in the fat contents are noticeable, though all Mr. Watson's samples fall within the rule that "No cocoa shall contain less than 20%* of fat." Is this an example of an old saying in England, that "Anything is good enough for the colonies?" The point I would make here is that it is, on the face of it, absurd to say that such and such a food-stuff is of greater value than another, unless, as Mr. Watson states, "some guarantee from the seller" is required, and yet how such can be given is not obvious when such great variations are shown by the analyses, and, finally, if such guarantee is given, what guarantee is obtainable that the food so obtained is going to do the work it is popularly supposed to do, for "one man's meat is another man's poison" is probably as true for the Kafir as for the white man.

In apologising for so lengthy a criticism of certain general principles, my excuse is that this is a matter that concerns us all, not only from a chemical but also from the every-day point of view of general health and working powers, which are, after all, closely allied to working costs.

THE SHRINKAGE METHOD OF STOPING AT THE FERREIRA GOLD MINE.

(*Read at November Meeting, 1910.*)

By G. Hildick Smith, B.Sc., F.G.S. (Member).

REPLY TO DISCUSSION.

Mr. G. Hildick Smith (*Member*): I am sorry to notice that except for several practical points brought forward by Mr. Johnson, nothing worth noting has been mentioned with regard to shrinkage stoping, although there are many points which could have been improved upon in the actual case described. It is very true that often the best way to learn something about anything is to start by making some foolish remark about it and then waiting for the inevitable "rise." I certainly got "rises" in three places over my remark concerning the novelty of shrinkage stoping on the Rand. On this point I stand corrected by the older and more experienced members, but it is certainly unfortunate, in the interests of mining men, that the valuable experiences of such members as Mr. Saner and Mr. Coombe are in danger of being buried with them, instead of being a guide to others who are less fortunate in the way of experience.

As to Mr. Saner's remarks concerning the question of keeping the mill going at any cost I would suggest to him that when "pushed" the best thing to do would be to mill waste, just to keep the ball rolling, but this of course is not new either, and was probably resorted to extensively in the "early days." Poor unfortunate old workers they must by this time know all the moves on the board, but apparently they do not always do the right one. As to keeping the mills going it would certainly be necessary that a certain reserve of broken ore should accumulate underground before milling was commenced, but once having started milling it would not be more difficult to keep breaking ground at the required milling rate by the shrinkage method than by the usual underhand method. As to the question of shortage of labour, circumstances might easily arise in which a good reserve of broken ore underground, broken perhaps at a period of sufficient labour, might help the mine in tiding over successfully a period of shortage of labour.

With regard to Mr. Coombe's remarks as to setts being unnecessary, it was found that setts were much handier, and made a much better job than poles and lagging, and in addition, when the ore is finally run off, the sett planks may be taken out again and used for lagging, shaft setts, etc. Throughout the building up of about 550 ft. of setts not more than two complete setts were destroyed by blasting or what not. A method far preferable to setts or poles and lagging is that employed at the Treadwell mines, where winzes and raises are so placed that by leaving pillars along each side of them the winzes and raises form the pass ways, and require no timber at all. This is one of the points which might have been improved upon. Another improvement I might mention is the use of $\frac{3}{4}$ in. round iron clamps spaced at convenient intervals, in preference to the ladders described in the original paper. Still another improvement is the replacing of the iron doors of the ore pass boxes (see Fig V. of the paper), by iron straps and planks (as shown at *e*, Fig. II. of the paper. Again, with regard to jumpers it was found that a line of pipes down which jumpers could be let fall became too easily jammed with fine stuff to be satisfactory. I just mention these points *en passant* as they may be of interest.

Mr. Coombe seriously questions my costs of box installation, but as he omits to say whether he thinks the costs too high or too low I cannot comment upon it.

Referring to Mr. Johnson's remarks with regard to the ore passes being run up with as little bend as possible, this is important from the point of view of the wear of the wall plates, which quickly become cut through by falling rock wherever an appreciable variation in the angle of dip occurs. Without going into much detail I do not see how Mr. Johnson makes the cost per machine shift for air, etc., come to 9s. From the data he has before him, *i.e.*, 10·48 tons per machine shift, cost of explosive 8·06d. per ton, and a total cost per ton of 3s. 1d., the cost per machine shift might be made up to 12s. 6d. for air, etc., and still allowing a sufficient margin for sundries and white and native wages.

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* Ikin and Lyster, "Hygiene (Advanced)."

With reference to Fig. VIII. of the paper Mr. Johnson has taken it that this figure with the dimensions given refers to an actual average bench in the stope. This was not what I intended it to represent, the idea being simply to show the position of the holes relatively to one another on any assumed size of bench, in this case 7 ft. × 5 ft. The distance between the top and bottom positions of the machine is correct, as shown, *i.e.*, 2 ft.

As to the question of using an arm on a horizontally rigged bar, it will be found better to get a smaller bench perhaps, but a bench every shift, than to try and get a larger bench and fail to get through with the drilling in one shift, due to time wasted and trouble with an arm on a horizontal bar, which is a very awkward thing to handle. With regard to breaking benches from the bottom of the stope to the top it all depends in which way the scheme is looked at, whether we consider we are working from bottom to top or top to bottom. In reality unless there is only one bench left we must follow the bench along and of necessity work from top to bottom. A combination of underhand and overhand stoping might be all right in a sound new mine, but with caved ground all around and in free milling rock as well, and also from past experience in this method as disclosed by blocks shown on the plan in the form of V's in horizontal positions, with their points together, which have been abandoned in the past having presumably become too dangerous to work, it is safer and better to break everything by the overhand method entirely. Although a little less tonnage is obtained at first yet eventually all ground broken can be run as required from a place of safety in the levels, and nothing is lost on the stulls, etc.

In conclusion, my thanks are due to those who were sufficiently interested in my paper to have commented on it.

THE MINE DUST PROBLEM

(*Read at January Meeting, 1911.*)

By Dr. J. L. Aymard (Associate)

DISCUSSION.

Mr. E. J. Laschinger (*Member of Council*): Although Dr. Aymard's paper is a rather rambling treatment of the many factors entering into the problem of health conditions underground, the thanks of the Society and the mining world in general are due to the author. As the author is a qualified medical man of no mean standing, I read into his wide treatment of the title of the paper "The Mine Dust Problem," and more particularly because of his statements :—

> Page 312, "The medical aspect of the dust problem remains to-day very much as it did ten years ago," and "I fear we have all taken the pathology of the disease as definitely settled whereas it is quite clear that it is not so,"

that in the present state of knowledge even medical men have great strides to make in order to arrive at a rational and scientific settlement of the many problems that affect the health of miners and other underground workers.

Underground hygiene, if one may be permitted to use the expression, does not appear to be by any means a simple and well understood branch of the science of health. If this statement be accepted as true, it surely seems absurd that legislators and Government officials should treat the matter by Acts of Parliament and regulations as if the problems involved had been definitely settled by scientific research and the exact determination of natural laws. In the present state of science on this question it will not be presumptuous on my part, even as a layman, to venture on a few remarks regarding underground health conditions, not from the point of view of discussing complex or minute details, but on general aspects. As an engineer I do so all the more readily because it is the engineer's peculiar province to carry out on a large and practical working scale the recommendations of the scientist. I think the time has almost, if not quite, arrived when the recommendations of the medical man with regard to underground labour conditions should be passed on to the engineer to carry out, in the same manner as the recommendations of the metallurgist are, with regard to metallurgical and ore reduction operations. The chief difficulty is that the medical man does not seem to be in readiness to give his recommendations. The medical profession seems to be in a state of unpreparedness, but by this I do not mean to cast any slur on that profession. The blame, if any, rests on those who are responsible for the health of any section of the public in proportion as that section is numerically important. As long as any subject is not known to be an exact science anyone's opinion may be of value, whereas after science has determined laws it is generally rash for the uninitiated in these known laws to enter into discussion with those who have passed the neophyte stage.

With this preamble I will now discuss various points that arise from a perusal of the author's paper, confining myself to matters that affect efficiency in any way connected with the state of the air which the miner has to breathe.

(1) The problem is first and foremost an economic one, and as such deserves the earnest consideration of the employers of underground labour, as affecting the efficiency of an expensive machine—the man. The health point of view in industrial undertakings is only of importance in so far as it affects efficiency in production. This being the case, it follows that any recommendations regarding health conditions must be capable of being carried out in practice without excessive cost.

(2) The health point of view is of first importance to the state in maintaining the happiness, virility and general efficiency of its citizens. If any trade or occupation involves conditions which amount to rapid and unavoidable premature death to citizens it is no more than just that such occupation be absolutely prohibited by law. This is an accepted axiom of modern thought. It is, however, the belief of those competent to judge that the ordinary occupations of mining are not such that they should fall under this extreme ban of the law. Indeed, we find in certain Government Commission Reports that it should be possible to make the miners' occupation as healthful as any ordinary surface work. The very sheet anchor of progress in South Africa is the successful exploitation of the mineral wealth of the country. The vast majority of residents here live and trade on the belief that occupation in mining is not and should not be a prohibited one. I make these general remarks because it is surprising to become fully aware of the intense feeling there is in South Africa generally regarding miners' phthisis and the absurd remedies that are suggested by some people who do not even look the question squarely in the face.

(3) Since in Rand gold mining, poisonous gases from the earth or poisonous minerals are practically unknown, and since sanitary science is a more or less exact science, the serious problem to solve is to maintain the air pure and free from harmful contamination from certain well-known sources. The object should be to maintain mine air in as nearly the same condition as the air off the open veld. What I wish to point out is that the dust danger is not an isolated problem. Although an absolutely dustless mine is a practical impossibility, still a dustless mine might be more dangerous than another with dust. The question seems to be, what characteristics in dust are harmful, what amount of such harmful dust is permissible as a maximum, and by what means can this dust be rendered harmless or comparatively so. That the human system is capable of throwing off dust inspired with the air is self-evident, for any miner who has died of the effects of dust must have thrown off many thousands of times as much dust as was found in his lungs. Even if "fibrosis" is the effect of lacerations of lung tissue by dust particles, this effect is the result of but an infinitesimal proportion of the inhaled dust particles. There must be certain conditions external to the presence of the dust itself which predispose the human system to harm from dust. The chief harmful ingredients in mine air are carbon dioxide and organic emanations from the body, excessive humidity, and the products of combustion of lights and explosives. The "smell of the upcast" is wonderfully uniform all over the world. Again, very few diseases are simple in their origin and pathology and the author's paper together with other treatises on the same subject form an admission that miners' phthisis is the result of a complexity of causes. Perhaps too much has been made of the dust effects. Dust being solid is found to some extent in the lungs of the patient, gases or vapours which may by as much or more have been responsible for the disease are naturally not found in the *post-mortem* examination.

(4) The author touches upon ventilation and makes certain suggestions. Since he admits that all dust found in the lungs is capable of suspension in air currents, and since ventilation, as he refers to it, is change of air by displacement, *i.e.*, currents of air, it follows that the stronger the ventilation the more the dust is carried through the mine. As far as the dust problem is concerned, strong ventilating currents during working hours are only an additional means of spreading the danger instead of minimising it; they have also other harmful effects; it is not healthful to work in strong draughts.

(5) The dangers and expense of over-ventilation bring one to the thought of corrective measures outside the crude and obvious process of simple replacement of air. Why not correct the composition of the air in mines by artificial means? Dust and dangerous gases may be washed out or eliminated by chemical means, excessive humidity may be corrected by artificial precipitation of moisture and excessive temperature may be corrected by artificial cooling. Such expedients in combination with a moderate displacement system of ventilation seem to offer the correct solution, especially for very deep level mines with extensive workings. It is a matter of relative cost and efficiency as to how far the orthodox mechanical or natural ventilation method should be supplemented by means more scientific and complex.

Without going into a lengthy discussion as to the harmful effects of excessive humidity I might point out that it is generally agreed by competent authorities that a wet bulb temperature of 80° F. approaches the limit when a man can not do his day's work efficiently or continue to maintain his bodily health. From an extract in our *Journal* of July, 1910,* it appears that the Socialist Parliamentary Party of Belgium even brought forward the proposition that with a wet bulb temperature of only 67° F. it was proposed to limit a day's work to six hours. In order to avoid excessive humidity the practice in ventilation of the Comstock Mines† is to deliver dry air at or near the working faces in metal pipes so as to prevent saturation by premature contact with wet faces; also nearly all drains are covered in to prevent evaporation of water and saturation of the air.

(6) Over a year ago two of our local workers, Messrs Whitford and Mills, proposed a system of removing humidity from mine air by means of mechanical refrigeration plant underground, and also the removal of carbon dioxide by a spray of alkaline solution (caustic potash or the like). The spraying and precipitation of moisture would also remove to a considerable extent dust and other dangerous gases and organic emanations.

There are, of course, other means of removing moisture and of treatment for gases, and although these mechanical and chemical treatments would involve expense it is quite probable not only that such schemes will be practical, as the alternative to over-ventilation, but also that they may have to be resorted to in the very deep levels as the only solution of the problem of keeping down excessive temperature and humidity.

(7) In view of the remarks made by Mr. Weston at the last meeting regarding the mines at Pzzibram providing up to 200 cub. ft. of air per man per minute, this would be an almost economic impossibility as applied to deep level mines on the Rand, and is only practicable in mines where the number of workers underground is small. To adopt this scheme on the Rand either extremely large special ventilating shafts would have to be provided or else the air forced underground and distributed at very high pressures at great cost.

Underground health conditions would not be improved by having a hurricane of wind blowing through the mine. What is needed in such a contingency is local purification and correction of the air to make it nearly like the open air above. According to an estimate which I have prepared we can take it that in order to provide each worker with the amount of air required by law (70 c.f. min.), the total cost is under 1d. per man per shift. The details of this estimate are as follows:—

	d.	
Power cost	0·35	
Attendance above and below ground	0·15	
Maintenance	0·10	
Working cost ...	0·60	per man shift
Interest and redemption on plant 1% per month...	0·25	
Total cost	0·85	per man shift

The cost per ton of ore mined will of course vary inversely as the useful labour of mining varies. For present Rand conditions it would be anywhere between 0·85d. per ton and double that figure, varying also as regards the capital charges depending on whether one or two shifts per day are worked. The above is calculated on single shift working.

In conclusion, I think the author deserves great credit for his work in connection with his dust catching appliance, respirator and other apparatus for bettering conditions for workers underground but I do think that so far we are only touching the fringe of the real problem in regard to miners' phthisis.

Mr. E. M. Weston (*Member*): I should like to know if the decision to put down a circular shaft on the New Modderfontein was not influenced largely by the question of ventilation? and I should like to remind Mr. Laschinger that there are two ways of supplying increased quantities of air up to 200 cub. ft. per minute to underground workers in the deepest mines. One is by using very high air velocities or by largely increasing the size of the shaft which Mr. Laschinger regards as impossible, and the other is by decreasing the number of underground employees per 1,000 tons of ore raised. I am quite certain that we are going to do this in the future, and that with the aid of electrical haulage and mechanical means of breaking and transporting rock the Kafir will be more and more reduced in numbers. The one man drill will tend to eliminate the hammer boy who will become too inefficient a machine to be used in the deepest mines, and with the drills or Mr. Robeson's rock-pounders will need more fresh air to operate them and thus aid ventilation.

Mr. E. J. Laschinger (*Member of Council*): I am very sorry I cannot answer that. You must ask that question of a consulting mining engineer.

Mr. R. Gascoyne (*Member*): There is one point arising out of Mr. Laschinger's remarks. I believe mechanical engineers, judging from a recent after-dinner speech, are very anxious to get below ground and see what they can do to lower costs and improve methods of mining even without employing labour. I suggest that Mr. Laschinger, seeing that machine drilling is a mechanical operation, should take into consideration whether it is not possible for the mechanical engineers to devise some means of, say, using water in connection with drilling machines so as to prevent the production of dust altogether.

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* From the *Colliery Guardian*, 1910, p. 1267.

† Proceedings A.I.M.E., Nov. 1900, by G. J. Young.

Mr. E. J. Laschinger: We might do so and pick up some other kind of disease which is as bad or worse.

Mr. W. Price Griffiths (*Member*): The legislation contemplated by the Union Parliament must make "miners' phthisis" one of the most serious problems in the industrial economics of the Transvaal. Dr. Aymard's paper is, therefore, most timely, and will, I hope, promote such a discussion and an exchange of views amongst those engaged in the industry as will go a long way towards a solution. Two vitiating factors in the mine air are recognised as the cause of this disease, viz.:—

1. The fine particles of dust, and
2. The noxious gases present in the air.

The former, I suppose, is the direct cause of silicosis, while the latter by producing ill-health, and hence a low standard of vitality, increases the susceptibility of the mine worker to the disease, promotes its growth, and in general materially assists in bringing about the fatal end.

Apart from these well-known causes, it is well worth while perhaps to consider the disease from another aspect. Is there a predisposition to the disease in the constitution of some men? If so, can it be discerned on medical examination? The opinion of the medical fraternity on this point would be valuable. If there exists a predisposition to the disease which can be medically ascertained, then I consider it a national duty to exclude all men so affected from entering a mine where phthisis is known to have been produced. Such a course would probably meet with a great outcry, but the success of the industry, apart from social obligations, demands it. To find a remedy for the dust evil of the mines is a duty: to prevent those who are already constitutionally weak, and who are naturally-adapted "toy-things" of the disease, from becoming accessible to the "germs" of silicosis by entering the mines, is another duty equally as important, and incumbent on us from national, moral, and economic considerations.

Turning to the vitiation of the mine air, I consider that the gaseous part of it presents no insurmountable obstacle. This must be readily admitted if one considers for a moment the very large quantities of dangerous gas removed daily from some of the South Wales coal mines and elsewhere. On the Rand it has now become fairly well recognised that natural ventilation is inadequate for maintaining the air in a healthy state. This has naturally led to the consideration of ventilation produced by mechanical means, which has been so very ably and fully discussed before this society that there remains little or nothing to add. It is highly satisfactory to know that, its necessity being recognised, its cheapness and contribution to mining efficiency justifies its introduction. With an improved trunk ventilation, such as mechanical ventilation is bound to produce, the air at the faces of "ends" will become considerably improved.

The necessity of valuing the ore developed should ensure winzes being kept at reasonable and regular distances, and wherever the poorness of the reef driven on is such that economy requires these distances to be increased, recourse may be had to local ventilation by means of small fans driven by compressed air, which is always on hand. The ventilation of these "ends" does not depend, as the author seems to think, on the velocity of the air in the shaft past their entrances. It is well recognised that they depend for their supply of air on difference of temperature, which is the principle of natural ventilation. The ventilation of a drive leading into the side of a hill is produced by the same natural causes. To deny that the velocity assists may be wrong, but to credit it with being the active principle is entirely erroneous.

He also wrongly attributes the stifling atmosphere sometimes experienced at the working face to the slight tendency to vacuum produced by the rapidity of the passage of the air down the shaft. The ventilating pressure which produces the ventilation of one of these levels is, say, about an one-hundredth part of the whole pressure responsible for the ventilation of the whole mine. Let us presume this to be 2 in. w.g. If this be converted into barometer reading, it will be found to be less than ·15 in. Thus the author must readily see that such a small difference of pressure is imperceptible, and in the case of a level it should be one hundred times less.

The author might be interested perhaps in extending his experiment with the two tubes at right angles to one another. By placing at the end of the horizontal tube a U tube half filled with water, he may ascertain the "tendency to vacuum" produced by the rapidity of the passage of air down the vertical tube by measuring the difference in the height of the water in the two limbs of the U tube. This may be ascertained for various velocities if a Biram's anemometer be suitably introduced. Should the author carry out this experiment, I should very much like his results to be contained in his reply to discussion. He appears unnecessarily alarmed at the stalactitic condition of the drives in places. Will he go so far as to suggest a glazed brick lining or the employment of a gang of dressers to dress down the sides and roofs? I think he will agree with me that the cost of both is prohibitive. In these days, when large crushing plants are the order of the day, haulage considerations should always ensure the sectional areas of airways being more than is actually required for the conveyance of air, and thus an efficient ventilation will be maintained at a sufficiently low water-gauge.

Contrary to the author, I certainly agree with the Commissioners as to the advisability of filling in old stopes and drives. The noxious gases of the Rand mines are chiefly heavier than air, and their tendency, when ventilation is weak, is to congregate in the lower unused workings of the mine. When a fall of the barometer occurs, and especially when this is sudden, they become a source of nuisance, if not of actual danger, by invading the nearest working places. Many miners have experienced this phenomenon. There is yet another reason. These old stopes by being open increase the sectional area, and therefore materially reduce the velocity. The tendency of the heavier than air gaseous impurities diffused in the air for conveyance outside is to separate out. In course of time these stopes would become a reservoir, not of cool refreshing air, but of deadly foul gases.

Coming to the root of the evil— the dust—we are confronted with serious difficulties at the start. Efficient ventilation aggravates it through the increased absorption of water. Deep mining produces the same effect. The dust, therefore, threatens to become more of a "thorn in the flesh" than ever. How may this be remedied?

In discussing this question it is most important to know the habits of the dust in the mines. A large quantity is no doubt carried to the surface, but this is no advantage if it has to go to other working places before it reaches the upcast. Our aim should be to provide air with no dust. I know this is as impossible as the production of a perfect vacuum, but aiming at a high ideal is the surest way to get near it.

The velocity of the air at the working places where a large quantity of dust is being constantly produced is generally slow. On this account the largest proportion of dust settles not far off. Water applied to it here temporarily damps its ardour, but it quickly revives under the influence of a dry atmosphere. Can we not make the effect of water more permanent? It has occurred to me that a soluble viscous substance dissolved in the water would materially assist in this direction. By this means the dust would still remain a captive after the water had evaporated, owing to the adhesive or "sticky" residue remaining. This solution should be applied in the same manner as water is now. All places known to produce dust should be specially attended to daily. Periodical spraying of all the air courses should be adopted, and this more lavishly at those places where the velocity of the air is slow. The effect of this would be to present a very wide adhesive area for the dust to settle on, and it should be the means of withdrawing a big percentage of dust from circulating in the air.

I regret not being able to suggest as well a substance suitable for this purpose, but that there is one which can be successfully applied is, I think, more than probable. Doubtless there are quite a few which are worth experimenting with. Anyway, until such time as we can drill holes and blast them without producing dust, it seems to me the only reasonable direction to seek a solution of the problem. We all agree with the author that the practice of leading dust laden air through the mines is bad. Air should be portioned out to do a certain amount of work, after which it should be regarded as sufficiently contaminated, and should be conducted to the upcast by the most convenient route. With suitable doors and bratticing, with air-crossings to avoid the downcast, this can easily be effected.

Every one must realize that the solution of this perplexing dust problem will never be attained by any one effective line of defence, but probably by a dozen or more; each one appearing by itself an insignificant detail, but, added together, forming a very efficient whole. They must be accompanied by strict rules, and any infringement must be made punishable. On this account respirators, however successful they may be in supplying the lungs with dustless air, must be regarded of doubtful value. They are admitted to be cumbersome and inconvenient during exertion, and therefore no strict rules can be applied to their use, but to men who value their health they will probably prove invaluable.

It is to be regretted that the author has finally despaired of allaying some of the dust at its origin. One had become so assured of its success from Press comments that the news of failure is unpleasantly disappointing. The idea of allaying the dust, had it been successful, would have effectively trapped a great deal of dust, and I trust his ingenuity will yet become master of the situation.

Whatever may be said of the justice of the Miners' Phthisis Bill, now before Parliament, it will have one good result, and that is, it will finally solve the great dust problem. Men engaged in the mining industry have been so engrossed in the task of solving problems relative to working costs that social duties were forgotten and disregarded. Phthisis will now become a factor in the working costs, and, like all other problems, it will inevitably suffer the same fate —a solution.

The Act may also have one bad result. It may tend to make the miner (if that be possible) more regardless of his health, and hence inattentive to the means devised for his protection, if they imply any little extra work. Whatever these means, stringent rules must be used to enforce them

I join hands with our President in thanking the author for his paper.

PRACTICAL NOTES ON COAL.

(*Read at February Meeting, 1911.*)

By Michael Dodd (Member).

DISCUSSION.

Mr. Tom Johnson (*Member of Council*) : One must agree with Mr Dodd's remarks in his paper about the waste of coal in working, also, how applicable these remarks are to our gold mining. I have in mind a mine where on a change of management taking place, the new manager started to work out the old workings which he had been told could not be worked. However, he cleared up some of the drives and started to clean out the old workings, the result being that 10% of the tonnage of the mine was recovered from these workings the value being so good as to increase the value of the total tonnage of the mine by 1 dwt. per ton and gradually increase the profits from £1,000 to £8,000 per month. This is a quite modern case. We have other cases not so bad as the one just mentioned, but bad enough, where ore of less value than average cost is left for the future, forgetting that this ore will in the future have to be mined separately at increased cost instead of a reduced cost.

In coal mining at the present time much better methods are in vogue, better care is taken about the working of the upper or lower seam first (a point we people need to look into), the more systematic opening out of the seams and the choice of system of working. Formerly, districts were very conservative about the system of working, but latterly this conservativeness is being broken down so that even in one colliery two systems of working may be seen due to change of conditions.

In the use of coal on the gold mines of the Rand there is room for much improvement. How many horse power are thrown away yearly by our winding engines exhausting to atmosphere, by using too many engines to hoist the rock, by letting compressors run nearly twice the time the drilling machines run, or more correctly running the rock drills little more than half the shift. It is some years ago since I advocated gas engines at the collieries as prime movers for the generation of electricity to be distributed to the gold mines. Then how much coal is wasted by using coal not suitable for the boilers or not altering the grates to suit the coal procurable? Some time ago I mentioned before the Society the successful alteration of grates at the Langlaagte Deep, by Mr. Denby, the engineer.

Much money is wasted on coal by taking the worth on calorific value, forgetting that it is not the heat that is in the coal, but the amount of the heat that can be used that determines the value of the coal to the plant. In the case of the mine mentioned by Mr. Laurie Hamilton working at a cost of 10d. per ton milled, one of the reasons for this low cost was that there was no contract for coal, the manager buying what suited him and the boilers in open market, the resident engineer doing the rest. If this was more general I think the results would be much better.

The President : Before we go I should like to make the complaint that so many of the mining men are not quite playing the game with us, seeing that we arranged to make special meetings for them. I hope the mine managers will come along and give us their views.

Visit to the Roodepoort United Main Reef G. M. Co., Ltd.

By the kind invitation of the Managing Director and the Management, about 150 members of the Society visited the Roodepoort United Main Reef Mine on Saturday afternoon, the 25th March, for the purpose of inspecting the new surface equipment of the property.

The party was received by Mr. P. Q. Treloar, the General Manager, Mr. M. Torrente, the Consulting Metallurgist, Messrs. E. Farrar and J. N. Bulkeley, the Consulting Mechanical and Electrical Engineers, and Messrs. Roger Price, T.

Johnston, A. E. Davis, and E. Musgrave, of the Head Office Staff, under whose guidance the members were shown over the works.

The following is a description of the surface equipment of the mine :—

Kimberley Shaft Plant.—The boiler plant consists of six 250 h.p. Stirling boilers with economisers and stack, and is provided with a Locke damper regulator. The feed pumps are of the well-known Weir type.

The engine house contains :—

One Ingersoll compressor with steam cylinders 18 in. and 34 in. diameter and air cylinders $18\frac{1}{4}$ in. and 32 in. diameter, all 42 in. stroke.

One Hoerbiger compressor with steam cylinders $31\frac{1}{2}$ in. and 51 in. diameter, and air cylinders $30\frac{1}{2}$ in. and 49 in. diameter, all $47\frac{1}{4}$ in. stroke.

One Worsley Mesnes direct winder with cylinders 20 in. diameter × 42 in stroke, and drums 8 ft. diameter × 2 ft. 4 in. wide.

One Robey direct winder with cylinders 30 in. diameter × 60 in. stroke, and drums 12 ft. diameter × 3 ft. 11 in. wide.

The condenser for the air compressors is provided with a centrifugal circulating pump and a three-throw Edwards air pump, both driven by a small vertical engine (Allen) at 500 r.p.m.

The condensing water is passed through a natural draught cooling tower standing in a dam on the west side of the boiler house.

The feed water is passed through a vertical feed heater, where it absorbs the heat from the exhaust steam of the winders.

Headgear.—The headgear and bins are of steel, and were made locally in the works of the United Engineering Company, and erected in record time.

Electrical Driving.—It may be pointed out that from the time the ore leaves the headgear bin, it is handled and dealt with entirely by electrical power until it reaches the sand dump and slime dams.

The total number of motors employed in the plant is one hundred and fourteen, all supplied by the A.E.G. Electrical Co., the whole of the switch gear being furnished by the S.A. General Electric Co.

Sorting and Breaker Station.—This is to the west of the headgear bins from which the ore is carried by conveyor belts.

Main Fine Belt.—The ore from the fine bin under the grizzly passes to a 20 in. conveyor belt driven by a $7\frac{1}{2}$ h.p. motor, and running the whole length of the sorting house and delivering to the loading bin belt which leads from the breakers to the loading bin.

The main fine belt also receives the fine from the washing trommels and the de-watering trommel, which are taken by two small belts referred to later.

The coarse ore is delivered to a 30 in. belt passing under the main ore bin and also under a small ore bin on the east side, which receives ore from the shaft of the eastern section of the mine about half a mile away.

This belt, which is driven by a 25 h.p. motor, conveys the ore to a Y-shaped chute which, in turn delivers it to either or both of a pair of washing trommels 11 ft. long × 4 ft. diameter, carried on Lea's patent rollers. Each of these trommels is driven by a $7\frac{1}{2}$ h.p. motor.

The trommels—supplied by S. Sykes & Co.—have two sets of perforations : the first, of comparatively large size, are intended to take out any fine which may have remained with the coarse ore and to drop it into a hopper beneath, from which it falls on a 16 in. conveyor belt, driven by a 1 h.p. motor, and is carried to the main fine belt already referred to.

The water for washing the ore is applied over the second set of perforations, which are of smaller size, and through which the water and the small fine washed from the ore passes into a second hopper and thence to a de-watering trommel 30 in. diameter × 6 ft. long covered with fine mesh screen. The water passes through this screen to a receiving tank, from which it is pumped up to the main launder in the stamp mill.

The small fine which passes through the trommel is received on a 14 in. belt, driven by a 1 h.p. motor, and delivered to the main fine belt.

The washed ore on passing through the washing trommels is delivered to two 40 in. sorting belts, which convey it to the chutes leading to the breakers. The waste rock sorted out from the belts is dropped through openings in the floor to a bin beneath, from which it is carried by trucks to the waste rock elevator (driven by a 15 h.p. motor), and thence to the dump.

The breakers—five in number, and each driven by a 25 h.p. motor—are of Grusonwerk manufacture, swinging jaw type, size No. 7, and deliver their product to a 30 in. belt leading to the loading bin, and driven by a 25 h.p. motor.

Loading Bin.—For filling the trucks of the mechanical haulage, a bin has been erected, to which the product from the breakers and the fine from the grizzly are conveyed by the loading bin belt already referred to.

The bin has been arranged with a view to further extension, and the head pulley of the belt is carried at a sufficient height to admit of a shuttle belt being placed on the top of the bin so as to distribute the ore to any desired length of bin.

Mechanical Haulage.—For conveying the ore to the mill—a distance of 2,900 ft.—a powerful mechanical haulage gear has been provided, to carry two-ton saddle-back trucks on a 24 in. gauge 30 lb. rail track, laid on steel sleepers. The line passes under the Main Reef Road to a point about 500 ft. west of the end of the mill. To this point also runs a steam locomotive haulage for transporting ore in similar trucks from the main incline shaft, a distance of about one mile.

The trucks are then transferred to a second haulage system, and after passing over an Avery Automatic Weighing and Recording Machine, are taken up to the mill bin on an inclined steel trestle 400 ft. long, the ore is dropped into the bin, and the trucks, after passing over two turntables, are returned down the inclined trestle, passed over a second Avery automatic weigher, and returned to the respective shafts for another load. It will be observed that by this system an exact record is provided of the weight of ore sent to the mill.

The haulage gears were made and supplied by the Austral Ironworks, the trucks by Robert Hudson & Co., and the rails and sleepers by the United States Steel Products Co.

No. 1 haulage is driven by a 75 h.p motor, and No. 2 by a 25 h.p. motor, both of which can be started and stopped from the weighing machine station, which is connected by an electric bell system to the loading bin and to the mill bin platform.

Stamp Mill.—The mill building and bins are of steel throughout, the only wood used being for floor platforms, ladders, king posts and cam shaft pulleys.

The mill contains 100 stamps arranged back to back. The falling weight of the stamps is 1,901 lb., the cam shafts—each for five stamps —are of nickel steel, and the mortar boxes of cast steel, weighing 14,000 lb. each, the whole of the battery parts being of Grusonwerk manufacture.

The pile blocks are of re-inforced concrete, and the timber king posts are carried on cast iron stools.

Two crawls are provided on each side of the mill over the stems and the cam platform, and a third crawl runs under the cam platform for handling heads, shoes and dies.

A strong rail track runs in front of the mortar-boxes to facilitate their removal and replacement.

Each set of ten stamps has a separate line shaft, and is driven by a motor of 50 h.p. The line shafts are carried on Hyatt roller bearings, and tighteners are provided for the cam shaft belts.

No. 1 Tailing Wheel.—The mill pulp is carried in launders with a 10% grade to the No. 1 tailing wheel, which is of steel, and built on the lines of a bicycle wheel with tension spokes.

The wheel (55 ft. diameter) is carried on steel towers, which also support the steel catch box launders ; these are lined with wood and with balata belting to resist the scour of the coarse sand.

The wheel is driven from the centre shaft with two sets of steel spur gear and a belted countershaft. A 50 h p. motor is provided for driving the wheel.

In addition to the wheel, a 10 in. Morris steel-lined sand pump is provided to allow of repairs being made to the tailings wheel buckets without stopping the mill ; the motor provided for this pump is 100 h.p.

A spill elevator of the bucket type is installed to lift and return to the wheel any drip from the launders or buckets, and is driven by a 1 h.p. motor.

A small wheel 9 ft. in diameter receives the coarse tube-mill pulp returned by the spitzlutten and delivers it into the main circuit (big wheel).

A drainage tunnel has been driven from the tailings wheel pits so as to prevent their being flooded in the event of any stoppage to the wheels; and slat doors are provided at the entrances to the tunnel to retain the sand, while allowing the water to pass. This tunnel has also been carried to the sites of the wheels for the future extension of the plant. It is connected to the pipe trench under the slime vats, and then runs out into the open on the east side of the extractor house. It may be remarked in passing that this tunnel has proved of the greatest service during the construction work in preventing the flooding of the excavations in the rainy season, thus obviating the necessity of pumping them out and the consequent delay in the execution of the work.

The pulp leaving the wheel is delivered to a pulp distributor of a novel type, replacing the usual coned spitzlutte.

The overflow from this separator is led to a separator of the usual type, which also takes the overflow from the tube mill dewaterer.

The underflow from the primary separator passes to the tube mill dewatering cone, whence the coarse sand is sent to the tube-mill.

The tube-mill discharge passes to the secondary separator already referred to, which returns to the tailings wheel any coarse sand which may have passed the mill, and delivers the overflow to the launder leading to the shaking tables.

Tube Mill Plant.—Five tube mills are provided each 5 ft. 9 in. diameter × 16 ft. 5 in. long

(No. 17 Grusonwerk Mills). Each mill forms part of a separate unit of the plant and is arranged to take one-fifth of the total mill product passing as underflow from the primary separator. Each mill is provided with a dewatering cone, and at the discharge end with two spitzlutten, where a final classification of the product is effected. The product overflowing the spitzlutte (finished product) is delivered on to the amalgamated copper plates. The underflow from the spitzlutten (coarse product) is returned to the main pulp circuit as already described.

Each mill is arranged to deliver to seven shaking tables with amalgamated copper plates, thus providing one tube mill and seven tables for each twenty stamps.

Plate House.—Thirty five shaking tables are installed here. The frames are of steel, and each is driven by a 1 h.p. motor placed beneath it so that no countershafting is required in the building.

At the lower end of the copper plate is provided a short length of wooden table with blankets for retaining any mercury or gold which may escape from the plates.

At one end of the house a strong room is provided, together with small office, clean up plant and furnace room, so that the amalgam can be retorted and the gold smelted without going outside the building.

No. 2 Tailing Wheel.—After passing over the tables the pulp from each set of seven tables passes an amalgam trap and is delivered to No. 2 wheel, which is practically identical in design and arrangement with No. 1 wheel already described.

Sand Plant.—From No. 2. wheel the pulp passes to spitzkasten, which deliver the overflow to the slime launders, and the underflow to the settling vats.

The spitzkasten are carried on a steel tower, from which project four cantilever platforms carrying the sand launders to the centre of the settling vats.

The settling vats are four in number, each 50 ft. diameter × 8 ft. deep, and fitted with Butters' distributors mounted on ball-bearings.

The overflow from the vats, together with the overflow from the spitzkasten, is passed to another separator, which returns any particles of sand to the wheel, and passes the slime on to the slime settlers.

The sand is discharged from the settling vats into trucks, which deliver to a 24 in. conveyor belt, driven by a 25 h.p. motor.

This belt delivers to a Blaisdell distributor over the treatment vats, which are six in number, each 50 ft. diameter × 11 ft. deep.

The sand from the treatment vats is discharged into trucks and taken by a mechanical haulage to the dump. This haulage is driven by a 25 h.p. motor.

Slime Plant.—The slime settling vats, three in number, are 70 ft. diameter × 12 ft. deep, with a conical bottom, giving an additional depth of 5 ft. 6 in.

Four similar vats are provided for slime treatment.

The decanted solution from these vats is sent to two clarifying vats, each 10 ft. × 5 ft., and is pumped from these to a receiving vat 10 ft. × 10 ft., from which it gravitates to the extractor house.

The decanted water from the settlers is sent to a return water vat 10 ft. × 10 ft., from which the return water pumps lift it to the three mill supply vats—each 60,000 gallons capacity—standing north of the mill.

These vats are also supplied from the Saxon Dam, about three miles distant, where are installed two 6 in. Rees Roturbo, four-stage pumps, each with a capacity of 30,000 gallons per hour, and each driven by a 100 h.p. motor.

Pump House.—The pump house is arranged to take all the pumps required both for sand and slime plant, together with the return water pumps.

The pumps—each driven by a separate motor—are arranged along the sides of the building, and the motors in two lines near the centre, with a passage between them. The slime pumps are in a pit at the end of the house, sunk to the level of the pipe trench.

The motor switchboards are grouped together on a platform over the slime pumps and communicating with the platform on the top of the slime vats, from which all the valves on the vat pipes are operated.

A five ton travelling crane—supplied by Hubert Davies & Co.—covers the whole of the pumps and motors.

The following is a list of the pumps installed:

Two 12 in. Morris Slime Pumps, with two 100 h.p. motors.

Two 8 in. Morris Slime Solution Pumps, with two 60 h.p. motors.

Two 6 in. Morris Sand Solution Pumps, with two 20 h.p. motors.

Two 4 in. Morris Solution Pumps, with two 10 h.p. motors.

Two 8 in. Rees Roturbo Return Water Pumps, with two 50 h.p. motors.

One 5 in. "Eureka" Pressure Pump, with one 30 h.p. motor.

One 3 in. × 4 in. triplex Gould Pump gland supply, with one 7½ h.p. motor.

Extractor House.—The extractor house is a steel building with concrete floor, and under the same roof is the furnace room, where the zinc slime is calcined and the gold smelted; a strong room and store is also provided. The bisulphate plant is in an annexe on the south side of the house, and the small vertical boiler provided here for heating purposes is the only steam plant on the whole of the reduction works. Twenty-two steel extractor boxes are provided, each 19 ft. × 4 ft. Zinc lathe, ball mill, filter press and a small centrifugal pump for charging are also provided, and are driven by a $7\frac{1}{2}$ h.p. motor through one of the few countershafts to be found on the plant.

Four solution sumps 60 ft. × 10 ft. are placed south of the extractor house.

Two large dams are provided for receiving the treated slime.

Power Distribution.—Power is supplied by the Victoria Falls and Transvaal Power Co., at 10,000 volts. It is led into the transformer house, equipped with three transformers 10,000/500 volts, and two 10,000/3,000 volts, the former being for the power supply to the reduction plant, and the latter for the Saxon Dam and underground pumping plant. The transformer house is arranged so that all the primary switchgear is placed in cells on the transformer side of the house, and the secondary gear on the opposite side, leaving an operating passage down the centre with no live parts accessible. Both primary and secondary bus bars are in duplicate, and arranged in reinforced concrete bus bar cells on the floor immediately above the switches, while the lightning arrestors are above these cells again. The secondary distribution switches are all of oil break type, and arranged in two compartments of a lean-to on the main house. Here also all bus bars are in duplicate. The circuits have all been arranged so that the power supplied to each department can be accurately metered. Recording instruments are also installed to give a graphic record of voltage and total power.

The contract for the excavations for foundations was given out on June 10, 1909, and the plant was started on August 15, 1910.
Engineering Department.

After the tour of the works, the visitors were invited to the dining-room of the mine boarding-house, where light refreshments were hospitably served. Dr. J. Moir (President), seized the opportunity to express the appreciation of the members of the pleasure and information the hospitality of the company had afforded them. The plant, he said, showed what the Rand was coming to. Two features were pre-eminent, the almost complete electrification of the plant, and its compactness, neatness, and efficiency on the metallurgical side.

Mr. Treloar said the plant could handle well over 1,500 tons a day if necessary, but part was at present hung up, as the mine was not yet far enough advanced to feed it fully. The present rate was about 1,000 tons a day. In eight months' time they hoped to be doing something like 1,500 tons. Already the reduction costs had been reduced by nearly one-half as compared with those of the old Roodepoort United Mine, and they hoped to lower them still more as the volume dealt with was increased. They knew what it meant to the industry in particular and to the country in general to be able to recover from 21s. to 21s. 6d. from 5·4 dwt. ore, and to make a profit on it. He hoped, even on that basis, to raise the profits of the mine to £10,000 a month. That could only be accomplished by working on a good scale. The feature of his work at the Roodepoort United, however, of which he was most proud was that since he took charge four months ago he had only made one change in the staff. That was to say, he only brought one man with him. He had every confidence in the staff as it stood. It had done good work, and worked well together, and he was confident they would make the mine a success.

Mr. Torrenti eulogised the work of the Society as one of the greatest factors in the advancement of metallurgy and of the gold-mining industry. They enjoyed an *esprit de corps* which made for co-operation, and they did their best, first because they were metallurgists with high ideals of their profession, and secondly because they had a sense of duty to their employers.

Mr. Edward Farrar invited criticisms of the plant, and emphasised the fact that practically all the work on the plant had been done departmentally. Only some minor and special features had been done on contract.

Mr. E. J. Laschinger also spoke, and reminded the members of the Society that the old Roodepoort Mine was one of the first to adopt the cyanide process. He thought the Society should discuss the merits of the plant at its next meeting. Such visits would thereby be enhanced in value.

Mr. A. F. Crosse said he had been 22 years on these fields, and had never before seen a plant which pleased him so much. Everything was so compact and neat, and so proportioned to the service required of it. The Romans built things to last, it was true, but they overdid it. The plant they had just seen would last as long as the mine, and not too much longer.

Mr. W. R. Dowling doubted whether the saying that a plant was out-of-date before it was completed was correct in the case of the plant they had just visited. The high stamp duty attained and the use of electrical power were all thoroughly modern. Even the shaking tables in the amalgamating house had impressed him so much that he felt he should have seen them before reading his paper on the respective merits of shaking and stationary plates.

The proceedings closed with cordial expressions of thanks to the hosts and pilots of the afternoon.

Obituary.

The news has just been received by cable of the death of Mr. JOHN HENRY JOHNSON, a former associate of the Society. The deceased gentleman, who was the father of our Past-President, Mr. Edward H. Johnson, and Mr. J. Hayward Johnson, paid a lengthy visit to his sons on the Rand some years ago, and attended many of the meetings of the Society, in the work of which he was greatly interested. The sympathy of the members of the Society is extended to Messrs. E. H. and J. H. Johnson in their bereavement.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

DUST, SOOT AND SMOKE.—The increasing attention now being given to the allied questions of smoke-prevention and the improvement of the purity of the atmosphere in all large towns renders the introduction of correct and scientific methods of recording the amount of dust and soot suspended in the air and the blackness or density of the smoke emitted from factory chimneys of the highest importance.

In the past there has been no attempt made to obtain comparative and permanent records of this kind on the part of the health or sanitary authorities of any town, and 'smoke prosecutions' at the present date are still too often based upon incorrect or unscientific evidence. The first step in any reform of the existing system is to show that more satisfactory and scientific methods of observation have been developed and are now available for general use. In the first portion of this article several methods of estimating the amount of soot and dust held in suspension in the atmosphere will be described, while in the second section of the article the newer methods of observing and recording the density of the smoke emitted from individual chimneys will be dealt with. Until the health or sanitary authorities of all large towns and cities take up the study of smoke emission in this manner their statements regarding the degree of purity or otherwise of their towns' atmosphere must be accepted with some reserve, for there is no subject of common observation upon which independent observers are more disposed to differ than upon the comparative density of the smoke emitted from their own and their neighbours' chimneys, or the relative purity of their own town's atmosphere as compared with that of their nearest rival. The personal factor must, in fact, be removed before accuracy can be hoped for or arrived at.

I.—Scientific Methods of Estimating the Amount of Soot and Dust in the Atmosphere of Towns and Cities.—A—Rubner's Method.—The Rubner method of soot and dust determination is based upon the aspiration of a large volume of the air which is to be tested (200 to 300 cub. ft.) through a filter paper placed in a special holder. Upon the surface of this paper the soot and dust are collected. (Fig. 1*a*.)

The aspiration is effected by a water-jet pump, and the volume of air is measured by an ordinary dry meter. The determination of the amount of soot and dust collected on the paper in twelve or twenty-four hours is carried out by comparison with a standard scale, since the amount is too small, even with the large volume of air used, to be determined by direct weighing. Renk has employed for the comparison a mixture of a known amount of soot and oil which, when used in a glass vessel of wedge-shaped design, gives a colour scale of gradually increasing density. This method of estimating the soot and dust in the atmosphere is now being employed by the officials of the Hygienic Institute in the city of Hamburg, the results being carefully recorded. It is stated that a series of these tests will shortly be published.

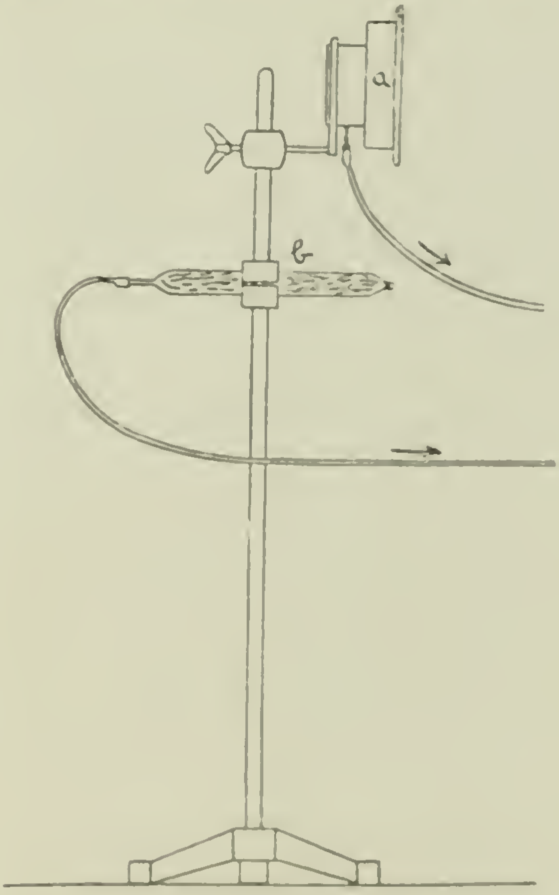


FIG. 1.—The Rubner and Hahn Apparatus.

B.—Hahn's Method.—The apparatus devised by Hahn consists of a double-cylinder pump worked by an electric motor with current from accumulators. The volume of air in this case is ascertained by multiplying the cubical capacity of the two cylinders of the pump by the number of revolutions of the crankshaft, as recorded by a counter attached to the same. The soot and dust are retained by drawing the air through a tube containing collodion wool. (Fig. I*b*.) The estimation of the soot and dust is made by dissolving this collodion wool with its deposit of soot and dust in a mixture of two parts of ether and one part of alcohol. The final step is to compare the turbid fluid thus obtained with various standard solutions containing collodion and known amounts of soot and dust. The apparatus for carrying out this method is more costly than that of Rübner, and the use of ether adds considerably to the expense of the final comparison.

C. — Liefmann's Method. — Liefmann's method differs essentially in principle from the two methods already described, since it is based upon deposition or 'sedimentation' of the soot and dust particles upon prepared plates and not upon filtration.

The method is described in considerable detail in the Habilationsschrift' (Degree Thesis), published by Liefmann at Halle University, in 1907. Considerable space is devoted in his pamphlet to the theory of 'sedimentation' as a means of measuring the soot and dust particles suspended in the air and to the reasons why *two* surfaces are necessary (a vertical one and a horizontal one) in order to obtain correct results. The practical operation of the method only can be described in this article. The apparatus required is simpler than in the case of the Rübner and Hahn methods, and it is therefore, more easily moved from place to place or erected at any desired point of observation. A firm, three-footed base of cast iron is provided with a vertical centre rod or spindle. This carries a horizontal plate on the edges of which are fixed four thin uprights carrying a light roof or cover. Under this cover is fixed the vertical disc of glass covered with a thin film of oil for catching and retaining the particles of soot and dust blown by the wind in a horizontal or inclined direction. These particles would not be caught by the horizontal disc fixed on top of the apparatus, since only the particles falling in a vertical direction impinge on this disc. The vertical plate must, of course, be kept facing the wind or air current, and for the purpose of causing it to turn with any change in the direction of the wind a large vane is fastened to the back of the spindle which carries it. (Fig. II.) The glass discs used by Liefmann in his trials of the method were the usual chemical clock glasses 100 sq. cm. in superficial area, and the oil film was obtained by painting their concave surfaces with a perfectly white-bleached oil by means of a small brush.

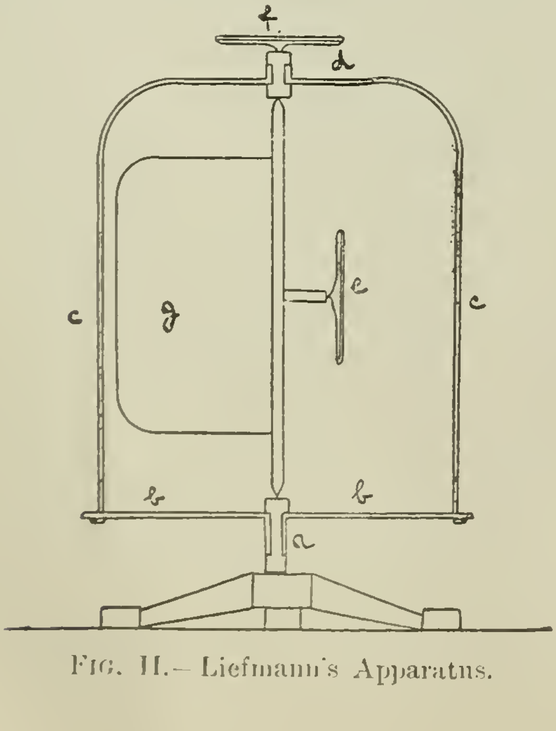


FIG. II.—Liefmann's Apparatus.

After twelve or twenty-four hours', or longer, exposure to the atmosphere the two glass discs are removed from the apparatus and newly-prepared ones are inserted, if the observations are to be continued. The exposed discs, with their films of soot and dust-covered oil, are then held over a small glass mortar, and the film is removed by aid of ether, which dissolves the oil and holds the soot and dust in suspension.

This ether solution is then evaporated by placing the mortar containing it in a warm place. It is most important to note that no light must be near this place, and that, as ether-vapour is heavier than air, it flows *downwards.* The residue is then rubbed up with 5 cub. cm. of the originally used oil, in order to yield a mixture for comparison with the standards. These standards are prepared by taking a weighed quantity of dry soot prepared from naphthalene and by mixing this carefully with 5 cub. cm. of white bleached oil in a glass mortar. These mixtures are then placed in a series of glass-stoppered tubes having flat feet, the quantities of soot used giving a scale reading from 1/10 mg. up to 5/10 mg. While the first tube of the scale is only slightly tinged a dark colour, the last tube will be perfectly black, and the difference in tint between successive tubes of the series is clear and distinct. The oil-and-soot mixture obtained from each of the exposed discs is now placed in a similar glass-stoppered tube and is compared with the standard tubes, in order to determine the amount of soot in suspension. The results must be calculated upon 5 cub. cm. of oil and multiplied by 100, in order to give the quantity of soot falling upon an area of 1 sq. metre (10·75 sq. ft.) during the time of exposure.

According to Liefmann, a test by this method can be completed in ten minutes, exclusive of the time taken for the evaporation of the ether; the accuracy is within 0·05 mg.

The following are some of the figures obtained in Hamburg in the winter of 1903-4 by this method :—

Date 1903.	Weather.	Results calculated upon one square metre and 24 hours mg.	Remarks.
Dec. 18	Bad	23·59	On horizontal disc.
Dec. 18	Bad	18·85	On horizontal disc.
Dec. 19	Bad	18·94	On horizontal disc.
Dec. 19	Bad	16·14	On horizontal disc.
1904.			
Jan. 15	Windy	188·45	On vertical disc, wind blowing from harbour.
Jan. 22-23	Good	35·29	On horizontal disc.
Jan. 23-27	Moderate	298·70	On horizontal disc.
Jan. 24-25	Moderate	76·40	On vertical disc.
Jan. 24-25	Moderate	19·11	On horizontal disc.
Jan. 27-28	Moderate	69·60	On vertical disc.
Jan. 27-28	Moderate	13·90	On horizontal disc.

The above tests show that, as a rule, more deposit is collected on the vertical surface than on the horizontal one, which is in accordance with the theory of the sedimentation process, since on most days there is sufficient motion of the air to cause a drift of the dust and soot particles in a horizontal direction. On perfectly calm days the horizontally-placed surface will collect the larger portion of soot and dust.

II.—New Methods of Observing and Recording the Density of Chimney Smoke. — The need for more scientific and accurate methods of observing and recording the density of the smoke emitted from factory and works chimneys was well brought out in the address delivered by Mr. James Swinburne, F.R.S., before the Association of 'Engineers in Charge' in October of last year. In the course of some comments upon the evidence given in 'black-smoke' prosecutions under the existing acts, Mr. Swinburne made the following pertinent remarks:—

"Smoke records are, naturally, chiefly of use in giving evidence about smoke, as, without them, the magistrate has to decide whether there has been 'black smoke,' or smoke amounting to a nuisance, from the evidence of a number of witnesses called an each side. This is rather apt to be unfair to the defendant, because the prosecution can bring evidence that, at such a time, on such a day, a witness saw 'black smoke.'

"The defendants had not their attention called to the smoke at that moment, and all they can do is to give evidence that at other times they did not emit 'black smoke,' or that their furnaces are so good that they do not believe they ever emitted 'black smoke.'

"The great difficulty is that, up to now, there has been no way of making any sort of estimate as to the blackness of the smoke. One witness may say, quite fairly, that he considered the smoke dense black, while another witness, quite as fair, looking at the same smoke, may say it is only grey or, at the most, dark grey smoke. There must always be the difficulty that the witness for the prosecution sees the smoke of which there is complaint without the defendant being able to produce any evidence about it."

Within recent years several instruments have been devised for over-coming this difficulty and for eliminating the unsatisfactory personal factor from the observation of the smoke emitted from the chimneys of factory or works. These will now be described seriatim.

(*a*)—*The Swinburne Photographic Smoke-Recording Apparatus*.—A full and detailed description of this method and of the apparatus required for carrying it out was given by the inventor in the issue of *The Times* (Engineering Supplement), of March 20, 1907. The following description is drawn from that article, the 'smoke-scale' referred to being that issued by the committee of the Institution of Civil Engineers in their 'Report on Steam Engine and Boiler Trials,' published by Messrs. Clowes & Co. The principle of the method is that of photographing the smoke-scale and the smoke upon the one plate, so that any variation in the depth of tone of the negative, due to the method or time of exposure, or to the methods and chemicals used in developing will affect both the chart and the smoke equally, and a fair comparison will be obtained between the two. As Mr. Swinburne says:—

"To photograph the smoke in the ordinary way would be quite useless as a record, because the appearance of the smoke in the resulting print would depend on the kind of plate used, whether it is slow or fast; whether it is isochromatic, and used with or without a screen; how far it is developed, and whether the negative is intensified. Then the print may be on a print-out paper giving great contrast, or on a platinum one, or a bromide paper may be used, properly exposed and fully developed, or over-exposed and incompletely developed. Most of these sources of error can be completely avoided by photographing the smoke-scale with the smoke. They then both receive the same treatment, and the smoke on one print is compared, not with the smoke on another print, but with its own smoke-scale. This smoke on one print may be very light. The scale is then very light, too, so that the smoke corresponds with the scale at, say, 0·4, which means that the smoke stops four-tenths, or 40 %, of the light. On another print taken by someone else at the same time the smoke may look very dark: but the scale will look very dark, too, so that the smoke will still correspond with the part of the scale marked 0·4.

"The scale is made by putting a process plate in a printing frame with a paper screen in front of it. This screen has a rectangular hole to correspond with the smoke scale. The hole is covered with a paper shutter, which is withdrawn in steps. A division into eight steps is probably as many as will ever be wanted. For a process plate a candle, about 2 ft. off, will do. The exposures for eight grades will then be sufficient if the first tint is exposed 64 seconds, the second and first 32, and so on. This gives exposures of 1, 2—128 seconds in geometrical progression. There is no special reason for exposing in geometrical progression like this, but photographers always do it. The finished plate is then clear glass, except that there is a scale along one side, so as to be in the sky in a photograph. The scale plate is mounted in the camera so as to be in front of the sensitive plate. It may be loose, so that when the camera is held lens upwards it falls right on to the sensitive plate. On turning the camera with the lens down, it falls back on to stops in the camera frame. A film camera would most likely be more easily fitted.

"A cheap pocket quarter-plate camera does quite as well. Unfortunately, the shutters have generally only one speed, chosen for beginners' home portraiture, and this may be far too fast for practical smoke and sky work in this climate. In taking the chimney the camera is held so that the smoke comes close to the scale. To get good results, the smoke should be taken against a background of uniform sky. Fortunately, it is easy to get a uniform dull grey sky in this country."

(*b*) *Lowden's Smoke Tintometer.*—This instrument is comparatively simple in construction and use, and will be found of value in making the preliminary observations of the blackness of the smoke issuing from a chimney before attempting to obtain a photographic record of the same by the Swinburne method and apparatus.

For it may be noted here that the emission of dense black smoke from most factory and works chimneys is *periodic in character*, and that if a chimney is observed closely it will be found that the periods (of longer or shorter duration) when dense smoke is emitted agree very closely with the periods during which fresh fuel is being thrown upon the fires. If black smoke is then observed issuing from a chimney at 11 o'clock a.m., say, for a period of one, two or three minutes, it is probable that smoke of the same density will again be emitted at 11.15 or 11.30, according to the interval between the times of firing. The smoke inspector or observer having once decided that the chimney is an offending one under the local acts, by observation with such an instrument as the Lowden tintometer, can then proceed to obtain a photographic record of the smoke by the Swinburne method and apparatus.

Lowden's apparatus consists of a metal tube having at one end an eye-piece and at the other end two apertures. One of these is fitted with a revolving diaphragm, by means of which it is possible to bring different pieces of tinted glass in front of the aperture. The diaphragm has six holes corresponding in size to the object aperture, and five of these are provided with tinted glasses of varying density, marked No. 0 to No. 5, according to the scale of the well-known Ringelmann smoke chart.

'In estimating the density of smoke with this instrument the observer approaches as close to the chimney as will enable him to see through the observing aperture of the instrument a circular disc of smoke without any of the surrounding sky. The instrument is held so that the other aperture is to the windward side of the smoke column, and by revolving the diaphragm the observer can match the varying densities of the smoke as it issues from the chimney top.'

(*c*)—*Bonham and Weber's Method.* — The two methods of observing and reading the density of the smoke described above are not continuous in character, and are further handicapped by the fact that an *observer* is required to operate them.

Messrs. Bonham and Weber have devised a continuous and automatic method of recording the density of smoke issuing from a chimney which is well worth installation in every large works, since the records obtained with it can be of great assistance and value in the general over-sight of the firing of all boilers and furnaces.

A perforated ring pipe is inserted in the main flue or breeching at the base of the chimney, and from this point a small proportion of the gases passing up the chimney are abstracted by an electrically-operated fan or pump, and are forced under pressure to the point where the recording apparatus is installed. This should be near to the chimney, to prevent deposition of soot in the pipe, and the flow of gas should be rapid.

The apparatus consists of two large drums revolved by clockwork and of a long piece of white tape, chemically treated to retain the soot and dust. The tape is wound upon one drum and the free end is attached to the second drum in such a manner that the chemically-prepared surface of the tape stretched between the two drums is upwards and is exposed to the chimney gases as they issue, *under pressure*, from a narrow opening in the mouth of the pipe which leads them to the apparatus.

The clockwork and the electric pump are then set in motion, and the apparatus is left working for a period of twelve or twenty-four hours, when the tape wound upon the second drum is cut off and is removed and examined for its record of soot and dust. It is obvious that this apparatus must be enclosed in a dust-proof cupboard and be kept under lock and key, in order to obtain accurate records. A still simpler modification of this method consists in the use of the tape without any chemical treatment, the record being obtained by drawing the tape *tightly stretched* over the orifice in the smoke pipe and by employing sufficient pressure in pumping the gases to force them *through* the tape.

(*d*)—*Methods for Guidance of the Firemen and Boiler Engineers.*—A difficulty which handicaps many firemen in the discharge of their duties is that the chimney top is quite out of sight from the firing plate of the boilers, and that they have no means of knowing, unless they leave their boilers and go outside the boiler house, whether black smoke is being emitted or not. The simplest method of overcoming this difficulty in those works where it exists is to cut openings in the boiler shed and to fix silvered glass mirrors at one or two points on walls, so that the fireman can see the chimney top, either directly or by reflection, *without leaving the firing plate of the boiler house.* The writer first saw this arrangement in Gallagher's tobacco factory, in Belfast, in 1902, and it has now, on his suggestion, been adopted in many works in the United Kingdom and in America. When this plan is impracticable, owing to the large size of the works and to the number and height of the intervening buildings, that proposed by Messrs. Bonham and Weber should be adopted. The mechanism already described in sub-section C is utilised not only to supply a sample of the smoke-laden gases to the recording apparatus, but a portion of the gas is forced through a small chimney stack erected in a convenient spot on the floor of the boiler or furnace house, and the top of this miniature chimney is provided with a white background, so that the colour of the smoke and issuing gases can be instantly seen.

It is necessary, however, when this method is adopted to arrange that the volume of exit gases travelling to the smaller smokestack shall be proportioned to its smaller diameter in relation to the larger one; otherwise, the indications will be wrong. With this device, both the firemen and the engineer in charge of the boiler or furnace house will have constantly under their eyes an indication of the density of the smoke emitted from the chimney of the works. Allowance will, of course, have to be made for the difference in the diameter of the two chimneys. A column of smoke which appears brown when seen through a depth of 6 in. will seem to be much darker, and may even appear quite black when viewed through a depth of 6 ft. or more, and the firemen will have to regard the slightest sign of smoke at the top of the chimney stack as indicative of a much worse effect at the top of the works chimney. However, there is not the slightest doubt in the writer's mind that, if every boiler house possessed this simple addition to its equipment, there would be a great diminution in the amount of black smoke emitted from the chimneys of works and factories on both sides of the Atlantic, to the material benefit of the surroundings.—JOHN B. C. KERSHAW.—*Cassier's Magazine*, January, 1911, p. 251. (R. A.)

METALLURGY.

THE DEVELOPMENT OF GOLD EXTRACTION METHODS ON THE WITWATERSRAND.—"It may be assumed that members of this Association are familiar with the main outlines of the methods employed in extracting the precious metals from the Witwatersrand banket reefs.

A reference to the last paper on this subject will make clear to anyone the underlying principles of the practice then adopted, and these have undergone no considerable change. I refer to the paper read in April, 1903, before this Association by Dr. Caldecott, who, as is well known, has taken a large share in the developments of the last seven years, with which this paper chiefly deals. It will perhaps be useful to take this statement of the facts of 1903 as a base on which to develop the principal features of present-day practice, and to follow the same order of treatment of the various processes to be considered.

As it is usually necessary in mining to stope out some quantity of inter-bedded, but valueless, rock, the first operation to be performed is to remove, as far as practicable, all such 'waste' from the valuable ore. The use of sorting tables and belts for this operation is still maintained, and in 1909 the average amount of material so discarded was 14·45% of the total mined, against 14·24% in 1903. This left nearly twenty-one million tons of ore to be further dealt with. Somewhat more than half of this passed over the screening arrangements to be broken down into 1½-in. cubes in the rock breakers.

There has not been much change in the devices employed for sorting and breaking; and improvement in detail and the use of larger units have been chiefly responsible for the saving effected in cost of operation. In the transport of the crushed ore from the breaking plant to the mill bins, belts have been introduced where the distance was short, and full-sized railway trucks and engines are employed where the mills are further away, and may be seen in operation at the Simmer and Jack or the Simmer Deep.

It is proposed to use electric haulage on a similar large scale for the new mill being erected at the City Deep, and this is in keeping with the development of electric power utilisation, consequent on the formation of large power companies whose sole business is the production and distribution of electric energy.

The considerable saving effected by generating electricity in large quantity, by huge machines centred in suitable localities, will ensure that this means of reducing capital expenditure and economy in cost of operation will be rapidly extended in the near future.

Many mills are now driven by electric motors, either in large units driving 100 or more stamps, as at the Angelo, or by means of smaller motors for each 10-stamp battery.

It is now becoming usual to arrange 10 stamps to be driven from one cam-shaft, but the mortar-box for each 5-stamp unit is still retained. Breakage of these cam-shafts is still a constant source of expense and of stoppage, though this has been considerably reduced as a result of careful investigation of the quality of the steel used by means of a microscopic examination and the utilisation of the latest knowledge in production and heat treatment of this metal. It is proposed at the City Deep mill to put an inch hole right through the cam-shaft centre, and to supply extra bearings between the cams, in accordance with Mr. Laschinger's designs, in order to obviate breakage as far as possible.

The greatest changes that have taken place in the battery are in the weight of the stamp and in the size of the screen used. The average weight of the stamps running in 1903 was 1,100 lb., and no stamp heavier than 1,500 lb. was in operation. The duty, 4·91 tons crushed per stamp per 24 hours, was thus small compared with the average for 1909, which amounted to 6·79 tons. This increase of duty is not only due to the heavier weight of the stamp employed or to the fact that the weight is rendered more effective by concentration near the crushing point by increasing the weight of the head and using shorter stems; it is also due to the fact that since the introduction of tube mills it is possible to use a much coarser screen in the mortar-box. The average weight of the stamps now pounding rock within 20 miles of Johannesburg is probably not more than 1,350 lb., but stamps up to 1,800 lb. are now at work, and in the near future a weight of 2,000 lb. will be employed at the City Deep mill, with a duty as high as 20 tons a day, or possibly more, the limit being set by the coarseness of the screen employed.

In 1903 a screen mesh of 600 to 1,000 holes to the square inch was usual, and anything coarser than 500 mesh was rare. To-day screens with nine holes to the square inch are actually in use, and the highest efficiency has been determined by the Mines Trials Committee to be correlated with this aperture. The same stamps which give a duty of 5·88 tons with a screen having an aperture of 0·015 in. show the high figure of 20·03 tons with the 9-mesh screen, having an aperture of 0·27 in. Even this result may be improved by separating from the ore fed to the battery that portion already fine enough to pass the screen employed (about 30), so that in future it is possible that a duty of 30 tons per stamp per diem may be fairly common.

It may be pointed out that the present economic limit of nine holes to the square inch is to some extent governed by the diameter of the tube mills employed. The standard size at present is 5 ft. 6 in. in diameter and 22 ft. long, but it is possible that in the future tubes of a greater diameter may be successfully employed. This will tend to increase the 'size of unit,' which is a favourable means of decreasing both capital required and working cost and may render economic the use of even coarser screens. Proceeding on these lines, a time may come when the stamp mill will be improved entirely out of existence, and the heavy rock-breakers employed will be set to crush finely enough to pass directly to the larger tube mills, or rolls might be used for the slight reduction still required.

No signs of such a progress are yet visible, though it appears merely a logical deduction from the success which has attended the efforts of our metallurgists in the direction of improvement in the operation of the tube mill plant, which is growing to be a more significant part of every new mill erected.

Coincident with the increase of coarseness of the pulp leaving the mortar-box is the difficulty of directly amalgamating the product without further crushing, and it is becoming clear that the advantage of removing the apron plates entirely away from the boxes will emphasize its necessity in the near future. This removing of the amalgamating tables from proximity to the mortar-boxes and the relegation of their duty to shaking plates, placed after the tube mills, is contemplated in many mines, though I believe it was first carried into effect at the West Rand Central mill, where it has proved entirely successful. The only possible drawback to this arrangement—that the lighter portion of the crushed ore, called slime, largely evades the tube mill shaking plates—can be obviated by special arrangement, as proposed for the 600-stamp mill being erected at Randfontein. At the City Deep new plant the shaking plates are placed under the same roof as the extractor-boxes of the cyanide plant, an arrangement which enables all gold recovered to be superintended and handled, till it is in smelted bars for the banks, in one building.

The tube mill is of course the principal novelty in the history of the reduction plant since Dr. Caldecott's paper was read in 1903, though he foreshadowed it when he wrote advocating the finer crushing of the coarse and pyritic portions of the mill pulp.

Mr. Denny first advocated the use of the tube mill for this purpose in this country, and referred to its successful use in Australia by Dr. Diehl.

At first introduced cautiously, after careful observation of its performance and of the improvement in extraction shown after the finer crushing obtained by its means, it is now used almost universally, not only on account of this increased extraction of the gold, but also because it has been proved here to be, in conjunction with the heavier stamps and coarser screens, a much cheaper method of reducing ore to the fineness demanded by economic considerations.

An endeavour has been made, on purely theoretic grounds, to show that the crushing efficiency of the tube mill is only one fourth of that of the stamp battery, but it is certainly true that its extended use is always accompanied by lower cost for reducing the ore to the utmost fineness now considered economically sound. Thus one or two tube mills were at first employed for every 100 stamps running, but at the end of 1909 158 tubes were in use, against 9,545 stamps erected, and it must be observed that there are many mines, whose 'lives' are now too short for the repayment of the necessary capital expenditure, where no tube mills are yet put up, and that the present tendency is in the direction of one tube to every 15 or 20 stamps; it is possible that the future will see even this proportion of tube mills increased.

A change of some importance has been made in the method employed for elevating the pulp leaving the mill for classification and subsequent treatment. Formerly tailing-wheels were almost universally employed, though even seven years ago plunger pumps were successfully used in one or two cases. The great disadvantage of the tailing-wheel, beside its high capital cost, was total lack of elasticity when the introduction of improved appliances demanded an increase in the total elevation necessary.

After considerable experimenting, a type of centrifugal pump, with cheap and easily-renewable wearing parts, was found suitable for elevating to any required height the coarsest material yet produced, and many are now in operation. Though the wearing parts may only last 20 or 30 days, the total cost, including capital redemption, of elevating pulp is less than by any other method, and any increase in elevation subsequently required is easily arranged by adding a length of pipe to the delivery column and suitably increasing the speed of the motor. One disadvantage which can, however, be overcome without very great difficulty is the lack of regular flow from the pump if the amount of the pulp to be lifted is varied by hanging up stamps or from other causes. This lack of regularity interferes considerably with the effectiveness of any arrangement for classifying the pulp and separating the coarser and heavier portions as is required for their different treatments.

The use of compressed air in the 'Pohle lift' is quite effective for elevating pulp, as at the East Rand Mines, and, though the efficiency is low, a similar method is largely used where the aerating effect is also of value in the solution of gold, as in 'Brown' agitators or similar apparatus.

The method of separation by means of cones has in the last seven years largely displaced the use of spitzlutte and spitzkasten, formerly chiefly employed for this purpose, and the introduction by Dr. Caldecott of a diaphragm partition near the bottom of the cone enables a much steadier and more uniform produce to be returned to the tube-mill plant for re-grinding, and increases the efficiency of the apparatus for other separations required. The underflow may be thus arranged to retain even less water than is suitable for the best tube-mill work, so that a second set of cones for removing the excess water may now be dispensed with, and any required adjustment may be accurately made by adding the amount of water experiment has shown to be requisite. A further advantage is found in the increased size of discharging nozzle that may be used when diaphragms are suitably employed. This avoids the frequent choking up by small pebbles of the smaller nozzles necessarily employed with less efficient arrangements.

The increased efficiency of the cone used in place of an inverted pyramid is largely connected with the fact that it is always arranged to overflow all round the circumference, thus ensuring greater tranquility for settlement and a smaller height of overflow. The spitzkasten has been invariably used with an overflow over one edge only, though of course it would be quite simple to arrange it to be used, similarly to the cone, with central inflow and baffle, and overflow over all four sides, where only wood is available for construction material, and in some other cases such an arrangement would present considerable advantages.

It has always been understood that successful treatment of crushed ore by cyanide solutions depended not only upon the ultimate fineness attained, but also upon the completeness of the separation of the leachable sand from the finer slime which tends to clog up the material to be percolated and to render the access of the air required more difficult.

By the older methods, still largely in use, it is found impracticable to separate sand containing less than 3% to 5% of slime, and even this efficiency is rarely attained. This difficulty is incidentally largely obviated in the method of collection by means of the Caldecott table. This appliance consists of a slowly-moving table with a peripheral band from 1 ft. to 3 ft. wide, upon which the thick pulp, containing less than 30% of moisture as separated at the underflow of a deep cone provided with a diaphragm, is allowed to fall. This band is provided with a filter-bottom, under which a vacuum is maintained by suitable air and water pumps. By these means the excess moisture is sucked out, and the dried material, with about 12% or less water, is continuously removed by a plough-scraper just before the point where the new material is being deposited. A few inches of sand are allowed to remain to assist the filtering medium, and this has to be removed at intervals when cleaning is necessary. The overflow of the separating cone employed carries away the greater portion of the slime, but in some cases a second cone is employed in which the underflow of the first is washed in clean water, and by this means less than 1% of slime is allowed to remain with the sand. The clean sand removed from the table is washed by means of a stream of weak cyanide solution to a centrifugal pump, and thus delivered to the collecting vats through Butters distributors, a large portion of the gold getting dissolved in transit. The treatment of the charge may be completed in this collecting vat, as is done at the East Rand Mines, but some little advantage is found in transferring it to one of a series of second treatment vats, as at the Simmer and Jack and elsewhere.

The advantages gained in the cleanness of the sand, and the fact that it is in contact with cyanide solution a few minutes after being crushed enable a considerable saving in capital expenditure to be made in the vat capacity required. Incidentally also an increase in the tonnage of slime separated this way, which may amount to 8% to 12%, permits a saving in working cost due to the fact that slime can be more cheaply treated than sand.

It is probable, as the greater part of the advantages of this method of collection are really due to the increased efficiency of separation, that its principles will be more largely availed of in the future, either by increased use of the whole apparatus or by other means of dealing with the same problem.

A radically different method of solving this problem of most economic treatment of the crushed ore is that being tried at the East Rand Mines under the name of the Arbuckle Process. This involves the treatment of the pulp as a whole without separation of slime from sand, and thus solves the difficulty by the method of evasion. The pulp, mixed with about three times its weight of weak cyanide solution, is agitated by means of compressed air in deep vats with conical bottoms, circulation being maintained by means of the 'Pohle' air lift principle. About 20 hours is required for a sufficient solution of the gold, when the whole pulp and solution are transferred to an ingenious separating apparatus, the solid portion with as little as 25% moisture is sent to a further apparatus in order to wash out all the dissolved gold, and the clean solutions are sent to the precipitation plant.

The possible drawbacks are extra cost for power for air compression, and for the mechanical arrangements involved, and the inherent defect that the light slime, which really needs only about one hour's treatment, must remain, undergoing needless circulation for the full time required by the most refractory portions of the ore. The general decrease in working costs, owing to larger units, greater skill in handling material, and other important reasons, render it probable that in future this method, perhaps modified in the direction of greater simplicity in the mechanical details, which promises lower capital cost for plant, which is very suitable for the finer grinding, gradually coming into the region of practical economy, and which evades the difficulty of successful percolation with extremely fine sand, will in this country become of as great importance as very similar methods in other parts of the world. It is possible that, as in Australia and elsewhere, some method of mechanical filtration may be found suitable for extracting the dissolved gold from the material treated, and already on these fields the vacuum filter is undergoing successful trial at the Crown Mines, where, however, it is in use for handling slime only. It is yet too early to form an opinion as to whether the more compact apparatus will affect the abolition from future plants of the enormous vats now used for the slime decantation process. It is certainly generally felt among metallurgists that the large and expensive plants used for treatment of slime by the standard process offer tempting opportunities in the matter of reduction of capital cost, but so far it is the general experience that a more than countervailing increase of working cost is involved. When it is considered that a shilling will cover all cost by the standard methods on the large scale now usual, with an extraction of 85 to 89% of the total gold, it is clear that the field for economy, either by increasing extraction, where residues are now worth 1s per ton or less, or in decreasing working cost, is not a wide one. The fact must also be borne in mind that in all feasible methods so far suggested the large vats employed for collection of the slime are still necessary, and even those for solution of the gold are either still required or a more expensive treatment in smaller vessels must be substituted.

With reference to the precipitation of the gold from the working solutions, it may at once be remarked that little change has been made during the last seven years. As, however, this work is performed at a cost of less than one penny per ton, and as the value is brought so low as 0·01 dwt., here again the field of future economy is restricted. Experiments have been made with zinc dust, in place of zinc shavings, and the results obtained so far promise economy of space, less gold locked up, and greater security of the bullion produced. Certain difficulties in the clean-up may possibly be evaded, and it is this weak point in our present methods that leads to the pious hope that in the near future the plan of using zinc dust and filter presses may have a brighter prospect.

A glance at the whole series of improvements effected in the past few years will render evident the fact that progress has been almost entirely in the domain of mechanics, and that nearly all proposals for the future emphasise the reduction in capital cost, and in some cases this is pressed so far as to partly ignore the fact that the increased working cost involved is more than counterbalancing. The origin of this state of things can no doubt be referred to the fact of the demonstrated economy in providing a much greater ratio of reduction capacity to mine area, and to the fact that much larger areas are now worked under one company. These facts combine to render necessary very huge capital sums, and all possible means of reducing the amounts necessary are so much the more welcome.

The most interesting question at the present time refers to the disposal of the residue after extraction of the gold has been pushed to its economic limits. Up to the present time nothing has been proposed or done with the slime but store it in dams, where it is useless, unbeautiful, and a source of expense. The coarser portion of the pulp, that is the sand, is now being sent back direct to the mine at the Simmer and Jack, which pioneered this improvement, and at several mines in the Central Rand sand is being removed from the old dumps and sluiced into the worked-out stopes below. This method of sand-filling was inaugurated this year at the Village Deep, and the success obtained is causing the adoption of similar methods on many of our leading mines. The disadvantages of using old residue for this purpose are the double handling of material involved and the expense of supplying lime to neutralise the large amount of acid and acid salts generated after long exposure to the atmosphere. On the other hand, when using sand directly from the treatment vats, the cyanide present must be effectually destroyed in order to avoid generation of prussic acid gas in the underground workings. It is fortunate that several methods of performing this work are now available. Dr. Moir and Mr. Gray have suggested the use of sulphate of iron and an alkali, and if this method is found sufficiently safe in practice it will cost an insignificant amount for the chemicals necessary. At the Simmer and Jack permanganate of potash has been found to completely destroy the cyanide, so far as even delicate indicators can determine, at a cost of about one half penny per ton, and experiments with an alkaline solution of bleaching powder, which promises even lower cost, are now being made.

It is certain that by some of the means suggested above it is perfectly feasible to solve the question of direct return of sand to the mine, but at the present moment there is little hope of rendering the slime available for underground support, and though the future may see the disappearance of the dismal, dusty sand dump, the permanent presence of the unsightly slime dam seems an assured feature of the landscape of the Witwatersrand.

Since so little progress, except in details, has been made in the chemical side of our work, it is much to be deplored that we have no University in the State where the research work, so necessary for imparting life to the dull routine of teaching, and so urgent for a correct understanding and development of the practice of gold solution and recovery, can be carried on; and it is to be hoped that this defect may in a short time be remedied.

In conclusion, the interesting observation may be made that in no period of its history in the past has so long a vista been possible as now of the future life of these, the most important goldfields yet discovered."—HENRY ARTHUR WHITE.—*The South African Journal of Science* (S.A. Association for the Advancement of Science), January, 1911, Vol. vii., p. 96. (W. R. D.)

LEAD ACETATE IN THE CYANIDATION OF SILVER ORES.—"I should like to suggest for the consideration of fellow-workers in the cyanide field that the advantages of the addition of lead acetate to cyanide solutions for the treatment of silver ores is considerably over-rated. It seems to be taken as an axiom at some plants that the use of a lead salt is almost as essential as the use of cyanide, and it is accordingly added as a matter of course; in some instances without any trial to ascertain its exact usefulness. Again, if on starting a new plant, working experiments are made with lead acetate and the advantage of its use demonstrated, the question is usually considered permanently settled, and is not subsequently reopened. In the course of a wide experience in the cyaniding of silver ores, I have found many cases where lead acetate appeared to have no effect whatever on the solution of the silver and sometimes even a deleterious effect on the extraction of the gold. In some instances the slight gain in silver extracted would be counterbalanced by the gold loss, leaving a net loss on the treatment amounting to the cost of the acetate used. I need hardly state that the above conclusions were arrived at by averaging the results of many parallel tests. Another phenomenon I have had occasion to notice is this: that sometimes on starting a new plant and making careful experiments with parallel charges the advantage of lead acetate seems abundantly proved, and yet if six or eight months later the acetate be discontinued for several weeks to give time for the elimination of lead salts from the stock solutions, and then the foregoing tests be repeated, no advantage will be indicated. I remember that two or three years ago a certain cyanide superintendent who had just started a new plant wrote me enthusiastically about the splendid profit he was making by using lead acetate; but when eight or nine months later I found the opportunity to put the question 'Are you absolutely sure you are gaining anything by the use of acetate?' his reply was "No, I am not sure, but I am trying to find out." I ascertained a few months later that he had discontinued it.

The use of lead acetate is based on an assumed presence of soluble sulphide in the solution due to the reaction, $Ag_2S + 4KCy = K_2S + 2KAgCy_2$

This reaction is no doubt correct as far as it goes, but is probably little more than momentary, and represents merely a stage in the process. The K_2S present, as it is in such extreme dilution, is very sensitive to oxidation, and except in the case of ores having a marked reducing action there will probably be sufficient dissolved oxygen in the solution to oxidize it all at once. When making some experiments on a large scale with Crosse's regeneration method, part of which consists in adding Na_2S to a sump solution to precipitate the zinc as sulphide, I noticed that solution which after treatment would yield an unmistakable brown coloration, on addition of a lead salt, was, by the mere process of decanting into another vat, entirely purified from any such visible reaction. I have never yet succeeded in finding soluble sulphide in any solution used for cyaniding a silver ore, though KCNS is almost invariably present. In view of these facts, therefore, I am led to think that with silver ores of average cyaniding value which have not a reducing action on the solution other than that due to Ag_2S there will usually be sufficient oxygen present to satisfy all requirements for the solution of the silver without excessive aeration or addition of foreign chemical salts.

In the case of ores that tend to impoverish the solution in regard to oxygen, the addition of lead acetate may be more or less beneficial. I say 'more or less' because, as stated above, I have found it 'more' beneficial in laboratory tests and and when starting a new plant, and 'less' beneficial, or not at all so, when the same plant has been running for a number of months. The explanation of the fact, I believe, lies in the presence or absence in the treatment solution of zinc absorbed during precipitation which would react with soluble sulphides in the same manner as lead acetate. Thus, when the solutions are freshly made up the acetate added will perform the function ascribed to it, but after they have been in use long enough to accumulate a considerable amount of zinc, the latter acts as a protective, and the effect of the lead salt ceases to be apparent. I was led to this explanation by a curious experience related to me by James S. Colbath, manager of El Rayo Mining and Developing Co. When working a certain stope in the mine he noticed a serious falling off in his extraction, coupled with a disorganization of his zinc precipitation. At the same time the zinc in the solutions gradually diminished to the vanishing point, as evidenced by the cyanide titrations. The readings for 'free' and 'total' coming closer and closer till they coincided. He at once suspected soluble sulphides caused by a peculiar susceptibility of the sulphides in the ore to rapid decomposition, and of course applied the stock remedy, lead acetate. The improvement, however, was very slight, and his gold extraction suffered somewhat from its use, so he began to argue from the unwonted absence of zinc in solution that it would probably be beneficial to add the latter from outside sources, which he did in the form of zinc cyanide manufactured on the spot from zinc sulphate. The effect was immediate and remarkable. As soon as zinc was present in excess the extraction and precipitation became normal and no further trouble was experienced. The outcome of these observations seems to be that lead acetate is not a panacea for all the ills of silver cyanidation. In the majority of instances where I have made careful comparative tests I have not been able to find that it was of the slightest aid in the extraction of the silver. I do not deny that there are cases where its use is indicated, but I maintain that such advantage cannot be taken for granted and must be verified from time to time. I am not prepared to offer any explanation of the harmful action of lead acetate on the extraction of the gold, but I have noticed it sufficiently often to accept it as a fact, at least in some cases. As stated above, Mr. Colbath also noticed the same thing, so it is a point worth looking into."—E. M. HAMILTON.—*The Pacific Miner*, Sept., 1910, p. 340. (A. R.)

CYANIDE EXPERIENCES IN NORTHERN MEXICO.—"We have experimented with various types of grinding mills, and are convinced that, for usual conditions, the best combination is single-unit stamps, Chilian mills, and small-diameter tube mills. Fuller Leigh mills have low capacity when grinding dry. The results on one installation showed $2\frac{1}{2}$ h.p. required per ton per day, whereas the standard arrangement mentioned above will produce the same results with $1\frac{1}{2}$ h.p. per day. Grinding wet, the mill has a prohibitive consumption of iron in wearing parts.

After careful observation of the 8-ft. conical mill under working conditions, we have been able to find no sorting of the pebbles to produce the claimed 'sledgehammer-tackhammer' effect, namely, an arrangement of the large pebbles at the large end and small ones at the discharge end. For producing slime it has about the same efficiency as the tube mill. A test conducted on the same lines as those described by V. B. Sherrod in the *Trans. del Instituto Mexicano de Mines y Metalurgia* gave the following results on a hard silicious ore. The feed contained 30% moisture in each case and the discharge 55%, 200 mesh:

	Kgs. per min. through 200-mesh per h.p.
Allis-Chalmers— $4\frac{1}{2}$ × 16-ft. Tube Mill . .	·30
Hardinge Conical Mill 8 ft. diameter ...	·306

As a competitor of the tube mill we believe that there is no advantage in the conical mill, but it is probable that with coarse ore, say with 4-mesh, there is a profitable field for the conical mills. This is as it should be, because the forces of impact and sliding increase with the increased diameter of the mill, and for efficiency should expend themselves in breaking larger particles of ore.

My experience is that in nearly all cyanide mills the best opportunity for increasing the efficiency of the work will be found in the concentrating department. Usually little care is taken to effect a clean separation, still there is no ore which yields the values under the same conditions for concentrates as for concentrate free ore. I cannot find that many companies are systematically determining the portion of the value in the tailing which is still locked up in the sulphides after treatment. The results are usually surprising, and justify the making of a daily assay of concentrates in the tailing.

The use of weak sulphuric wash for removing cyanicides from concentrates before treatment, as worked out by Hutchinson at the Goldfield Consolidated mill, ought to increase the field for cyaniding concentrates in Mexico, where in some cases the process is prohibited by a cyanide consumption of 60 pounds per ton.

For filter-pressing slimes the Butters and the Moore are the only machines doing good work on high-grade ore in the northern part of the Republic, but the Oliver continuous rotary filter has a useful field on ores up to 15 pesos per ton in value. It is limited to this figure, because with this machine it is hard to get a satisfactory washing of the cake to remove the dissolved values. This is especially true when the filter is used in connection with pneumatic agitation, as rapid deposition of lime takes place, which rapidly decreases the efficiency of the washing. Acid treating must be conducted with care in order to secure good results, and the tailing must be washed each day to determine the amount of dissolved value contained. Daily washing of the tailing to determine the work of all classes of filters is essential for consistent work, and this is a point that is neglected in many plants.

At one plant where the Oliver filters are installed we found it necessary to maintain the protective alkali at 0·1% CaO to prevent zinc cyanide from accumulating in the solution. The pulp is agitated in Pachuca tanks, and under these conditions it is necessary to acid treat the filter canvas every third day in order to keep the dissolved values in the tailing below 3 oz silver per ton.

Acid treating usually takes about two hours' time. It is first necessary to scrub the canvas to remove all the slime adhering, and then apply the acid through the wash pipes, scrubbing the canvas constantly in order to bring the acids in contact with all parts. It usually pays to save the spent acid as it is discharged from the pump. When all effervescence ceases we wash the canvas with water.

Difficulty has been experienced in keeping the pulp in agitation beneath the revolving drum. Small air jets have been used, but have the bad feature of introducing a large quantity of carbon dioxide, which continuously deposits calcium carbonate in the band around the drum of the jet. Agitators of the Trent design are now being used, and give good results. We are at present experimenting with a pulp box of concrete with small clearances between the sides and the drum, and the feed of pulp will be at the bottom, so that the current will serve to keep the sand in agitation. It is necessary to have the sides steep, so that the sand will not accumulate at any point and stop the drum.

Using either zinc dust or shavings for precipitation, there is a great economy that is not appreciated in most cyanide plants. It is usual to hear cyanide men speak with pride of their zinc-box work in getting tailings that contain only a trace of gold and silver. But I claim that this implies an unnecessary consumption of zinc, because it is really only necessary to have a small portion of the total solution barren, namely, that portion which is necessary to wash the filter press cake. This should never exceed one ton per ton of ore treated. Still some of the largest plants in Mexico are precipitating several tons of solution for each ton of ore, to traces. The following table will show comparative results on a month's run made at the Dolores mill:—

Class of solution.	Tons.	Value in pesos per ton.		Lb. zinc.	Lb. zinc per ton solution
		Before ppt	After ppt		
Strong agitated solution	3,757	28·00	1·50	1,368	0·37
Mill solution					
(1) Press barren solution	3,359	8·00		1,813	0·51
(2) Battery	5,174	8·00	1·50	889	0·17

Tons of ore treated, 3,174.

Total zinc per ton, 1·28.

Zinc dust is particularly efficient when used in this way, so as to leave about an ounce of silver in the solution. It is then possible to cut the zinc so closely that only 5 per cent. of the metal will be left with the precipitate of gold and silver. The product is then easily melted, and affords a high grade bullion.

We have a method for using the short zinc that is produced in the zinc boxes which may be of interest. The short zinc as produced is washed in a trommel and stored under solution. It is then fed in small portions to a 4-ft. agitator, into which a small stream of ·2% free cyanide solution enters at the bottom and overflows at the top of the tank. This solution contains about 10 oz. of silver and gold per ton, which is partly precipitated by the short zinc, and then flows to the zinc boxes to be completely precipitated. It is thus possible to do useful work with the short zinc rather than by using acid to dissolve it or even the roasting and melting process which is so well carried out at the Dos Estrellas mill, but which requires a large amount of fluxes.

For ordinary melting of precipitates we find that a flux calculated as half of Na_2O, ZnO, SiO_2 and half Na_2O, ZnO, B_4O_6 is the least expensive. Here ZnO represents all the impurities The average portion of fluxes to form the slag are 6½% borax glass and 4% dehydrated sodium carbonate. The silica costs nothing, and is usually about the same amount as the borax glass. Percentages here are based on the wet weight of the precipitate. At times the slag contains 50 oz. of silver, but all of the slag is milled and any shot metal recovered.

A by-product of matte from the bullion smelting is being treated at the Dolores mill by an interesting process. It pays to give a rough treatment, because the matte assays 150 oz. gold and 6,000 oz. silver per ton, and in the quicker realization of money value is a considerable advantage. The treatment consists of melting the separate portions of matte together in a graphite crucible and oxidizing the sulphur by a stream of compressed air introduced into the charge in the crucible in a graphite tube. This requires about two hours' time, although all of the matte is not oxidized, and the remaining part assays about 1·5 oz. gold and 1,000 oz. silver per ton. This is granulated in water and shipped to the smelter. The bullion separates by gravity and is easily detached from the low-grade matte."—L. M. Kniffin. —*The Pacific Miner*, Sept., 1910, p. 313. (A. R.)

MINING.

Coal Mining in Nova Scotia.—"A new era in the history of coal mining is rapidly developing in Nova Scotia, and the hand-pick, wherever the conditions are favourable, is being supplanted by machines for under-cutting coal. Where coal-cutting machines cannot be used, improved methods of mining have been adopted which tend to greater economy and better results.

The larger coal companies, such as the Dominion and the Nova Scotia Steel & Coal Companies, having coal-fields lying at easy angles and, therefore, favourable to machine mining, have equipped their collieries with the most modern and improved coal-cutting machines, and in common with many of the smaller coal companies, have adopted the latest methods of handling, cleaning and assorting coal on the surface in preparation for the market.

The principal reasons for the installation of mining machinery are—to reduce the cost of production, to increase the output, and to obtain the largest percentage of lump coal.

The introduction of machinery into the coal mine has not reduced the number of men employed, but as in other trades, the stimulus of labour-saving devices has greatly helped the industry, and a scarcity of miners rather than a surplus now exists.

If we make a comparison of, say, one machine colliery with a hand-pick, or a number of hand-pick collieries, the advantages of coal-cutting machines will be fully demonstrated. Dominion No. 1 colliery, Cape Breton, during the year 1909, employed underground 463 men. The output for the year was 544,499 tons. The hand-pick collieries of Cumberland combined, only produced 508,202 tons in the year 1908, yet they employed underground 1,806 men.

While mining machines are of great value in coal getting, they are a most important factor in the rapid development of the collieries.

The introduction and extended use of coal-cutting machines in thin 'seams.' or beds have made the working of these profitable, and placed many of them on a competitive basis with the thicker seams. The British Royal Commission on Coal Supplies in their report stated that in the year 1900 about 17·7 per cent. of the total output of the United Kingdom of Great Britain was obtained from the seams of less than three feet thickness. If seams less than three feet can be worked successfully in any other country, there is no reason why they should not be worked in Nova Scotia, conditions being equal.

Coal mining being the chief industry of Nova Scotia, the province's greatest asset is her mines and minerals. These should therefore be protected, as they are the heritage of the people. Along the eastern seaboard of the County of Cape Breton there are valuable coal-fields. Some of these fields contain five or six workable seams overlying each other with a variable thickness of strata intervening. The question arises, which of these seams should be first attacked? In my opinion, if the market affords it and other conditions permit, the descending order of working is the most practical. Working the top seam first is beneficial to the seams below, forming a cushion over them, especially over the thick seams. By thus relieving the pressure above, the proportion of slack in the underlying seams is reduced. The liability of a crush is averted, and the risk of men's lives is lessened by the lessening of the vertical pressure of the strata. Fatal accidents have often resulted in times of 'creeps' or 'crushes' in coal mines when sudden pressure has been exerted, and large lumps of coal suddenly forced out upon the miner without any warning whatever.

By working the lower seams first, all the strata right up to the surface is disturbed and broken; and while this settling process is taking place, the upper seams are damaged and sometimes wholly destroyed.

The possibility, therefore, of danger to other seams must be carefully considered in opening up a coal-field containing seams overlying each other, and the method least calculated to seriously affect the commercial value of the coal-field should be adopted. The tendency of past years seems to have been to work the most profitable seams first, but it has had very baneful effects, and has not always been the most profitable system of mining. Of course, I am well aware that competition plays a large part in determining what seams must be opened up first. But if keen competition forces the opening up of the thicker seams at the beginning, might it not be well to consider the workings of thin seams at the same time? This would be a decided improvement on the methods of mining coal, and wherever tried in the province has met with good results.

The method of mining is generally determined after a careful study of the thickness of the seams, the nature of the strata above and below, the angle of the seam and the texture of the coal.

The method of mining coal in Nova Scotia is generally either bord-and-pillar, or long-wall.

The conditions favourable to long-wall working are thin seams which freely part from the roof, and tender strata overhead. A band of 'dirt' which is easily separated from the coal, makes material for stowage with which to fill up the goaf when the coal has been extracted, so that the roadways, which sometimes must be naturally very long, can be supported. Particular pains are, however, taken to support the roadways and packs, or pack walls are built on each side of the roadway from the floor to the roof. These are usually of stone blasted from the roof or the pavement. In the long-wall method, ventilation is simple, as the full current sweeps along the working-face without the aid of brattice, and free from obstructions met with in the bord-and-pillar system. In very deep workings the vertical pressure is such that long-wall is the only system that can be adopted.

When conditions are not favourable to long-wall the most practical system is the panel system. When passages are being driven to win the coal, these should be continued back to the boundaries. At the same time headways or balances should also be driven from the main levels to the rise, a distance of four or five hundred feet, and rooms turned off to suit the cleavage of the coal. As soon as the rooms have been driven through to the adjoining balances, pillars should be drawn at once and all the coal extracted. This would save the expense of ribbing, would relieve the pressure and prevent a 'crush' or 'creep' in that district, and, as usual, where work is concentrated, the cost would be reduced to a minimum and larger profits obtained.

The size of pillars varies in different collieries and is determined by the nature of the coal, the thickness of the seam and of the cover over it. Small pillars often induced 'crush' and 'creep' and brought great loss to the owners; but this lesson has been so well learned in Nova Scotia that the present tendency is now all the other way. Especially must large pillars be left where the roof and floor are very strong and the nature of the coal is soft and tender. Where these conditions are the opposite, a smaller pillar may be left; but at no time should any risk be taken, and the margin of strength in the pillar be less than that stated by leading mining authorities.

As passage-ways are driven to win the coal, and districts opened up, all coal should be extracted, except where it is necessary to leave pillars for the support of surface plants, residences, main roads and railways. Under sea working (and we have at the present very much of it in the Island of Cape Breton) requires special attention and has to be dealt with according to the nature and thickness of the cover. The fact that much of the coal that is now being extracted is taken from collieries opened up fifty years ago, is proof that present methods are superior to those of our predecessors, and it is only natural that they should be."—NEIL A. NICHOLSON, *Mining Society of Nova Scotia*, October 15, 1910, p. 627. (A. R.)

MODERN AMERICAN COLLIERY EQUIPMENT.—"A typical example of the most modern and efficient American colliery equipment is furnished by the Marianna mines, owned by the Pittsburg-Buffalo Company, Pennsylvania. They are situated within the boundaries of a coalfield, which it is estimated has an area of some 800 square miles, and which is expected to provide the future fuel supply of Pittsburg, the great iron and steel centre. For economical operation with a large area such as that available at Marianna, it was necessary to install the most modern machinery, as 1½d. per ton saved in operation more than covers the entire cost of this equipment. The installation of absolutely fireproof buildings for boiler, engine and power houses, and other buildings, prevents the possibility of delays by fire and is an insurance against stoppage of work from such cause. In equipping the plant at Marianna the above factors have been taken into consideration, and notwithstanding that the most modern machinery and fireproof buildings have been provided, the cost in proportion to the annual output has been less than the average mine in the district, or in the United States, being under 1½d. per ton of coal production. The force of the above statement will be better appreciated when it is considered that many mines having an annual capacity of only 50,000 tons to 100,000 tons per annum, and having a field of only 100 acres of coal, have equipment costing £20,000, or from 5d. to 7½d., per ton of coal produced. When it is realised that the production of the Marianna pits will be 1,500,000 tons per annum, or 5,000 tons per day, as against an average production of all mines in the United States of less than 300 tons per day of 280 working days per year, the magnitude of the operations will be apparent. Eighteen tons have been wound in one minute, or at the rate of 1,000 tons per hour, or more in one hour than the average American mine produces in three days.

Shafts, etc.—For the development of the seams two main rectangular winding shafts, each 22 ft. × 33 ft. outside the lagging and about 450 ft. deep, were sunk 4,800 ft. apart. These shafts are so situated that the coal from the entire field can be delivered to them with the gradients almost entirely in favour of the loads. The shafts have cageways 7 ft. × 20 ft., so that two trams can be raised on each cage in tandem; they are also provided with an airway 11 ft. × 20 ft., and with stairways from the surface to the bottom of the coal. A small shaft 12½ ft. × 24 ft. was sunk for the purpose of taking men in and out of the mine, as well as for handling slate and supplies; so that the larger shafts could be given over entirely to the winding of coal; while at the same time all supplies could be lowered in daylight. A circular brick-lined shaft, 16 ft. in diameter, ventilates the entire mine, avoiding all partitions and brattices in the shaft, the air leakage inevitable with such partitions, and at the same time making the air current independent of the motion of the cages. The tipple, washery and coke ovens have been located with regard to the duty of each in relation to the others.

The principal factors which determined the design of the surface plant were—that it should load a large tonnage of coal with minimum breakage in handling; that all impurities should be removed from the coal, and that there should be the least possible delay on account of breakdowns. The cages carry two loaded trams, each with a capacity of three tons of through-and-through coal, so that a large tonnage can be delivered with the least amount of handling. The coal is dumped over four cross-over dumps of heavy pattern, on to 1¼ in. bar screens. The lump coal, after going over the 1¼ in. screens is dumped on to the picking tables, where all impurities are removed and the coal delivered into railway wagons. The impurities which are taken out are thrown on to a lateral conveyor which delivers them to a crushing plant, where they are made serviceable for use in the boilers. The picking tables are 6 ft. in width, about 80 ft. long, and are run at a speed of 40 ft. per minute. The small coal falls through the 1¼ in. bar screen on to conveyors, whence it is elevated to revolving screens. These separate the nuts from the slack. From the revolving screens the nuts and slack are distributed by separate conveyors to the loading bins, the washer, or to the small bins over each picking table. The surface equipment includes a Luhrig coal washer, with a capacity of 100 tons per hour. It has 16 primary jigs, and four secondary jigs which rewash the refuse from the primary jigs, thus recovering the "bone" coal.

Amusement Hall and Bath House, etc.—The houses in the town of Marianna are built almost entirely of brick and vary in size from four to fourteen rooms. Each house is provided with hot and cold water and bath. The houses are also equipped with electric light, and, if the tenants desire, with natural gas.

The amusement hall is a brick building of three storeys, 106 ft. 8 in. long by 62 ft. 8 in. wide. The basement contains eight bowling alleys, with lavatories, a barber's shop and restaurant. The floor above contains a billiard room and theatre, while the greater portion of the third floor is given over to a roller skating rink and dancing hall. On this floor there is also a reading room, and a lecture room in which it is proposed to give instruction in mining.

The bath house is modelled after the most advanced German shower or rain baths. It is a brick building 42 ft. × 125 ft. in size, and of four storeys. It is so arranged that men coming out of the mine are landed on a covered concrete bridge leading to the fourth floor of the building, which constitutes the assembly room. In the centre of the assembly room is the rack containing the safety lamps. The men going into the mine here receive lamps which are prepared for them by the attendants, and the men working coal-cutters receive their bits and oil for operating the machines. The men coming out of the mines deposit their lamps, empty oil cans and worn machine bits with the attendants, who clean and fill the lamps and exchange worn bits and empty oil cans for sharp bits and full cans of oil for the following day. The men coming out are checked by the time-keeper and pass down stairs to the bath rooms, where they bathe and leave the building with clean, dry clothing, leaving their soiled mine clothing suspended from the ceiling and drying out for use the next day. The building contains 99 shower baths for the men and two separate bath rooms for boys employed as slate pickers, etc., on the surface. There are also baths for the officials and visitors in separate rooms. An emergency hospital on the ground floor is a valuable adjunct.

The machine shop is really a collection of shops all under one roof, as the building contains a machine shop, carpenters' shop, blacksmiths' shop, tram repair shop, brass foundry and four large store rooms. The building is equipped with one 25-ton electric travelling crane, three small travelling cranes and a jib crane. The blacksmiths' shop and forge equipment is large enough to make all kinds of forgings for railway wagons, mine trams, and mining machinery.

Underground Workings.—The underground workings are on the same scale as the surface equipment. The main shaft bottom has a capacity of 250 loaded, and as many empty mine trams, and is arched for 1,100 ft. on the rise and 600 ft. on the dip side of the shaft. The gradient is ideal for handling coal, being 1·1 per cent. in favour of the loads for the entire length, so that no grading was required. The loaded trams are fed on to the cages by a chain haulage and the empties will be returned by a rope haulage which is now being installed. All the coal is gathered by means of 6-ton compound gathering locomotives. Steel track is laid into all workings, and there is not a mule in the mine. The main line hauling is done by compound compressed-air locomotives 15 tons and 19 tons in weight. These machines deliver the coal to the shaft bottom."—*Iron and Coal Trades Review*, Nov. 25, 1910, p. 860. (A. R.)

MISCELLANEOUS.

The Ruby.—"Mineralogically the ruby belongs to the corundums, of which the sapphire, topaz, Oriental emerald, and Oriental amethyst are also members. Corundum is a sesquioxide of aluminium, and the various tints are due to the presence of different colouring matters. In the ruby oxide of chromium in greater or less degree is responsible for the ruddy colour so sought after.

A ruby of perfect colour is a very rare and beautiful gem, and its value is commensurate with its rarity. Indeed, a flawless 'pigeon's-blood' ruby weighing 5 carats would be worth ten times as much as a first-water diamond of the same weight, while perfect rubies of 6, 8, or more carats are of such unusual occurrence as to bring very high prices. By the term 'pigeon's blood' when applied to the colour of the ruby is meant that shade which is by international taste accorded the post of honour, though many of the gems have a darker tint which is quite as beautiful.

Burma, in India, is conceded to be the principal source of the true pigeon's-blood ruby. There the mining is in the hands of the Burma Ruby Mines, Ltd., an English corporation which secured the right of working the district soon after Burma became ceded to Great Britain in 1886. Previous to that date the location of the famous mines was unknown, and all attempts to learn about them resulted in failure, for the rulers exercised a most strict surveillance over foreigners, and saw to it that any such were kept in ignorance of the exact location, etc. It is known, however, that the methods used were very crude and tedious; the natives merely dug out the 'byon,' or ruby-bearing earth, hoisted it by crude derricks to the top of the pit (these 'pits' were from 50 ft. to 60 ft. deep), and there left it to dry in the sun before searching for gems. To-day essentially the same methods are used by the native miners, in whose hands such labour is still left, and who work under terms much more advantageous to themselves than were permitted under the old regime. In defence of the antiquated methods used it may be said that they bring results, and do away with the necessity of costly machinery.

Besides the true gems, large quantities of spinel and balas rubies are found. These spinels, though often (and quite excusably) mistaken for the genuine stone, are really minerals of very inferior qualities and entirely different composition, being in this latter respect aluminate of magnesium. The spinel is much less rare than the ruby and not so hard. Besides red, they exhibit a great variety of colours, some being bright cherry; a rare variety is a deep violet, others have a cinnamon shade; and a white spinel comes from Brazil mixed with diamonds. Many of the large historic rubies in royal regalias (including that in the Maltese cross of the English crown) have been pronounced spinels by modern mineralogists, and are consequently of little value. The balas ruby is merely a spinel of proper quality, having a rose or deep pink colour. The best balas are found principally in Ceylon, though both they and spinels are generally found wherever the ruby occurs.

From Ceylon also comes a true ruby of a rich rose colour, but not many of the desired pigeon's-blood shade are found, though in all other respects they are beautiful and attractive stones, suffering, of course from the flaws which occur in all rubies. The Ceylon stones often belie the name, for in colour they run from very light red almost to a pinkish, being otherwise very brilliant.

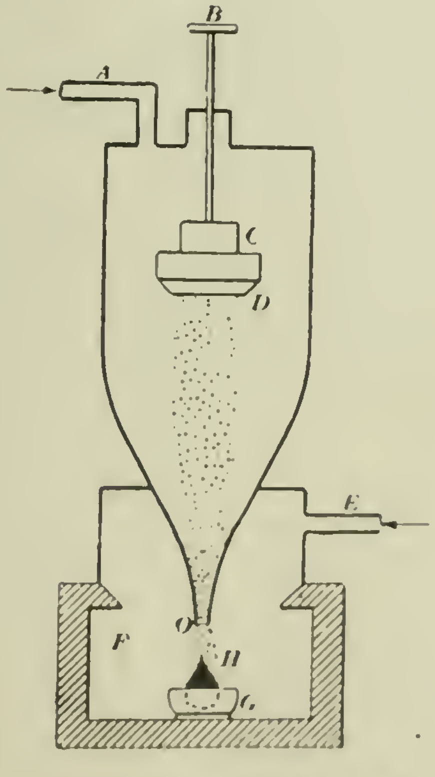
Similar to the Ceylon stone in many respects is the ruby of Siam. This country, though principally noted for its beautiful sapphires, often produces rubies that rival the best Burma stones, though their colour is more frequently a deep red (often called 'ox blood'). It is believed that any really good stones which may be discovered in Siam are sent overland to Burma and sold as Burma rubies.

The United States has brought forth rubies as well as sapphires. The principal districts are in North Carolina and Montana. In the former State gems of 3 and 4 carats were found, free from inclusions and of good colour; indeed, the quality of these stones often equals the pigeon's-blood rubies of Burma, though their occurrence is rare.

The detection of artificial rubies, as they are made to-day is an exceedingly difficult task, for the reason that the best artificial ruby is made of precisely the same materials of which the natural ruby is composed—pure alum and oxide of chromium or manganese, which latter form the colouring matter.

The furnace used in making these 'synthetic rubies' is illustrated in the accompanying diagram. It is a very ingenious application of the principles of the oxyhydrogen blow-pipe, and by its means a temperature as high as 2,000° C. (over 3,000° F.) may be obtained. The process of manufacture is as follows:—

A certain amount of powdered alum is placed within the box *C*, along with a proportionate measure of oxide of chromium to produce the red colour of the ruby. By means of a small hammer operated by an electromagnet, the tapper *B* is touched at various times. This causes the box to vibrate, and the powdered mixture is thus sifted through a fine gauze sieve *D*. Thence it passes down to the orifice *O* within the firebox *F*. At that point the combining of the oxygen gas (admitted through the pipe *A*) with the hydrogen, which enters at *E*, on ignition causes a heat so intense that the powdered alum and oxide of chromium are instantly melted and fused into crystal globules, which fall into the platinum crucible *G*. In this manner a pear-shaped gem *H* (called a 'brut') is built up, the size depending upon the amount of material used. Some of these synthetic rubies are 80 carats in weight.



Furnace for making Rubies.

After this 'brut' has been allowed to cool (if touched when warm it would fly to pieces owing to the great strain of its molecules), it is sent to the jeweller's, by whom it is cut into one or more gems which in colour, refraction, hardness, durability, and general beauty are identical with the natural ruby. Indeed, so absolute is this identity that the pawnbrokers of Paris and other great cities refuse to take rubies on pawn, for they cannot distinguish the artificial from the genuine.

Rubies can be made in this way at an expense of about 40 cents a carat. In the little laboratory of M. Pasquier in Paris (who introduced this method) 100 carats a day may be produced, and 8 to 10 carats an hour is a fair average in all such factories.

Such a discovery as this will have a serious and lasting effect upon the legitimate mining of stone. According to Mr. R. K. Duncan, in his 'Chemistry of Commerce': 'The ruby mines on their present basis of profitable working are absolutely doomed.' We can only hope that those who control this process will exercise good judgment in regulating the manufacture of their beautiful product, so that the market may not become unstable or overstocked.

Already the use of these synthetic gems is very widespread, and the natural ruby bids fair to become the exception rather than the rule. This fact, however, is not to be greatly deplored, for the artificial product is not an imitation; it is an equal of the genuine, even excelling it at times in beauty of colour and crystallization, and in freedom from flaws. If this were not so, the synthetic ruby could never have become the success which it is to-day."—MORRIS R. WARD.—*Mines and Minerals*, Dec. 1910, p. 319. (A. R.)

POTASSIUM CYANIDE FROM BEETS.—"The only factories where potassium cyanide is made from a by-product of sugar beets are two in Germany and one at Kolin, 40 miles east of Prague, Bohemia, according to a report made by Consul Joseph I. Brittain of Prague. The molasses by-product is sold to factories manufacturing alcohol, which by fermentation and distillation produce a first quality 97·7 % pure alcohol for medicinal and chemical purposes, and a second quality denatured for fuel and light. The refined wholesales at $12.27 and the denatured at $9.43 to $9.54 for 26·41 gallons. The former pays an internal revenue tax of $18.27 per 26.41 gallons. After making the alcohol there remains a still thicker and darker coloured molasses, resembling pine tar in appearance. This residue is sent from the various alcohol factories in tank cars and barrels to Kolin where, after subjected to steam heat, it flows into long metal troughs and to retorts where it is burned for several hours, until the gas escapes into pipes. The material then passes through an extended system of pipes, undergoing various processes until it reaches the place where it is mixed with lye, when the potassium cyanide is formed, after which the moisture is extracted by centrifugal force. The powder is then conveyed to another room, where it is hydraulically pressed into cakes and packed into boxes for shipment. The output of the Kolto factory is about 240,000 lb. a month, and is sold in gold and silver mining companies in South Africa."—*Mining Science*, Sept. 25, 1910. (K. L. G.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

ONE THOUSAND TWO HUNDRED MINING EXAMINATION QUESTIONS, VENTILATION PLANS AND HINTS TO CANDIDATES FOR CERTIFICATES. By G. L. KERR, M.E. (London: Crosby, Lockwood & Son, 7, Stationers' Hall Court, E.C.)

"The questions in this book have been compiled with the view of assisting those who are studying mining with the intention of preparing for a manager's or under-manager's certificate, and have been selected principally from the papers set at the examinations held in the different districts of Great Britain. They are divided into various sections, dealing respectively with the following subjects:—The Coal Mines Regulation Acts; arithmetic; geology and search for coal; blasting and explosives; generation and transmission of power; sinking and fitting shafts; modes of working; supporting and timbering underground workings; winding; haulage; pumps and pumping; mine gasses and ventilation; surface arrangements; surveying and levelling; and miscellaneous Prior to the questions, the qualifications of candidates who propose to present themselves for examination for first or second-class certificates are enumerated, and then follow useful hints to candidates. The compilation is certainly a helpful one for students whose aim is the manager's certificate."—*Iron and Coal Trades Review*, Feb. 10, 1911. (A R.)

Beadle, Clayton, and Stevens, H. P. Rubber Production and Utilisation of the Raw Product. Cr 8vo, pp. 142. *I. Pitman's.* Net 1s. 6d.

Beadle, Clayton, and Stevens, H. P. Rubber Production and Utilisation of the Raw Product. (Pitman's Common Commodities of Commerce). Illust. Cr 8vo, pp. x.-132. *I. Pitman.* Net 1s. 6d.

Bowles, O. Tables for the Determination of Common Rocks. 18mo, limp. *Spon.* Net 2s.

Deerk, N. Cane Sugar: A Text-Book of the Agriculture of the Sugar Cane, the Manufacture of Cane Sugar, and the Analysis of Sugar House Products. Together with a Chapter on the Fermentation of Molasses. Cr 8vo, pp. 592. *N. Rodger.* Net 20s.

Donington, G. C. A Class Book of Chemistry. Cr 8vo, pp. xi.-399. *Macmillan.* 3s. 6d.

Donkin, Bryan. A Text-Book on Gas, Oil, and Air Engines. 5th Edition. Revised and Enlarged. 8vo, pp. 664. *C. Griffin.* Net 25s.

Evans, John Castell. Physico-Chemical Tables. Vol. II., Physical and Analytical Chemistry. Roy 8vo, pp. 702. *C. Griffin.* Half-Morocco. Net 36s.

Harden, A. Alcoholic Fermentation. Roy 8vo. *Longmans.* Net 4s.

Henthorn, J. T., and Thurber, C. D. The Corliss Engine and its Management. 6th Thousand. 12mo, limp. *Spon.* Net 1s. 6d.

Hornby, John. A Text-Book of Gas Manufacture for Students. 6th Edition. Revised and Enlarged. 8vo, pp. 436. *Bell.* Net 7s. 6d.

Isler, C. Well-Boring for Water, Brine, and Oil. 2nd Edition. Revised and Enlarged. 8vo, pp. 282. *Spon.* Net 10s. 6d.

Journal of the Institute of Metals. Vol. LV., 1910. Edited by G. Shaw Scott. Cr 8vo, pp. 340. *Institute of Metals.*

Kemp, H. R. The Engineers' Year Book of Engineering Formulae, Rules, Tables, Data, and Memoranda, 1911. Cr 8vo, leather. *C. Lockwood.* 10s. 6d.

Kershaw, G. B. Modern Methods of Sewage Purification. 8vo, pp. 370. *C. Griffin.* Net 21s.

Macfarlane, Walter. A Practical Guide to Iron and Steel Works Analyses. Cr 8vo, pp. 196. *Longmans.* 4s.

Marshall, W. J., and Sankey, H. R. Gas Engines (Westminster Series). 8vo, pp. 294. *Constable.* Net 6s.

Mitzakis, John. The Russoal Oil Fields and Petroleum Industry. Illust. 8vo, pp. 106. *Pall Mall Press.* Net 3s. 6d.

Wager, H. On the Effect of Gravity upon the Movements and Aggregation of Euglena Viridis, Ehrb, and Other Micro-Organisms. Illust. 4to, sd. *Dulau.* Net 4s. 6d.

Warr, J. W. The Electrical Ignition of Petrol Engines. 12mo, pp. 76. *Alabaster.* Net 2s.

Watson, Hugh S. Sewerage Systems, their Design and Construction: A Practical Treatise. Roy 8vo, pp. 330. *Lockwood.* Net 10s. 6d.

Watson, M. The Prevention of Malaria in the Federal Malay States. With a Preface by Ronald Ross. 4to, pp. 139. *Liverpool School of Tropical Medicine.* Net 7s. 6d.

Wilson, F. R. F., and Hedley, G. W. Elementary Chemistry: Progressive Lessons in Experiment and Theory. Part II. 3rd Edition. 8vo. (H. Frowde). *Clarendon Press.* 5s.

Wood, T. B. A Course of Practical Work in Agricultural Chemistry for Senior Students. 8vo, pp. 56. *Cambridge University Press.* Net 2s. 6d.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 18/11. James Forrester Brown. Sliding jump set. 13.1.11.

(C.) 20/11. Harry Salmon (1), The Expanded Metal Co., Ltd. (2). Improvements in the manufacture of expanded metal and apparatus therefor. 13.1.11.

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(C.) 46/11. George Harlow (1), Karl Martin Faye-Hansen (2). Improvements relating to protective apparatus for alternating current distribution systems. 27.1.11.

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(C.) 53/11. Herschel Merle Comnr. Improvements in vapour generators and burners. 31.1.11.

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(P.) 62/11. Osborne Ernest Wilson. Improvements in bending stirrups and tie-bore for use in reinforced concrete constructions. 2.2.11.

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(C.) 66/11. Lewis Condict Bayles Improvements in combined throttle valves and oilers. 3.2.11.

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(C.) 68/11. William Appleton Lawrence. Process and apparatus for extracting rubber or rubber like substances from the vegetable sources. 3.2.11.

(C.) 69/11 Albert Edward Montagu Call. A process for cleaning mine workings of dust and gases after blasting. 4.2.11.

(C.) 70/11. Arthur Reginald Angus. Improvements in or relating to railway safe running devices "A." 4.2.11.

(C.) 71/11. Arthur Reginald Angus. Improvements in or relating to railway safe running devices "B." 4.2.11.

(C.) 72/11. Arthur Reginald Angus. Improvements in or relating to railway safe running devices "C." 4.2.11.

(P.) 73/11. Nicholas Chard Thomas Juniper Automatic transportation system. 6.2.11.

(C.) 74/11. Gordon Ashcroft. Improvements in the manufacture of nitrogen containing compounds (amides, cyanides and cyanide of alkali metals, and also of alkali metals. 6.2.11.

(P.) 75/11. Allan Campbel. Projection draughting instrument. 7.2.11.

(P.) 76/11. Francis Ernest Drummett. Improved process for extracting the gold from black sands. 7.2.11.

(P.) 77/11. George Anderson Robertson. Improvements in driving means of automatic feeders for feeding ore and the like to crushing and grinding machines. 7.2.11.

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(C.) 88/11. Robert Davy (1), Frederick Davy (2), John David McKenzie (3), Clement Stanhope Bertram (4). An improved process for extracting gold from refractory ores. 11.2.11.

(C.) 89/11. Stephen William Smith (1), John Duff Innes (2). Improvements applicable to the actuating fluid supply connections for rock drilling machines, pump connections and the like. 11.2.11.

(C.) 90/11. Paul Millington Jerrard. Improvements in explosion engine actuated machines and cars and the like, and apparatus therefor. 13.2.11.

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(P.) 106/11. John Galilee Barclay (1), Archibald Newton Barclay (2). Means for preventing the over winding of pit cages. 17.2.11.

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Beatty, G. H., *Notices to* Reitzburg, near Vredefort, O.F.S.; *Journals to* c/o Park and Jumper Streets, Belgravia, Johannesburg.
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Jameson, J. I., *l/o* Rooiberg; Mining and Metallurgical Club, St. Ermyns, Westminster, London, S W.
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Rand, E. T., *l/o* East Rand; Post Office, Boksburg.
Saner, C. B., *l/o* Johannesburg; Royal Colonial Institute, Northumberland Avenue, London, E.C.
Simmons, T. W., *l/o* Langlaagte; Meyer and Charlton G. M. Co., Ltd., P. O. Box 27, Jeppestown.
Spence, A. G. B., *l/o* Germiston; Nourse Mines, Ltd., P.O. Box 32, Denver.
Tennant, A., *l/o* Boksburg; Roodepoort Central Deep, Ltd., P. O. Box 114, Roodepoort.
Thomas, C. Fred., *l/o* Johannesburg; Zaaiplaats Tin Mines, Potgietersrust Station, Pietersburg Line.
Twining, D. H., *l/o* Barberton; The Village Mines, Penhalonga, Rhodesia.
Webb, H. H., *l/o* London; 71, Broadway, New York, U.S.A.
Woolf, M. P., *l/o* Wiluna, W.A.; Nararoa G.M., Norseman, W. Australia.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

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Vol. XI. APRIL, 1911. No. 10.

Proceedings
AT
Ordinary General Meeting, April 22, 1911.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the South African School of Mines, on Saturday evening, April 22nd, Dr. James Moir (President) in the chair. There were also present :—

58 Members: Messrs. W. R. Dowling, R. Allen, Tom Johnson, E. J. Laschinger, O. P. Powell, A. Whitby, H. A. White, J. A. Wilkinson, James Littlejohn, A. F. Crosse, W. A. Caldecott, A. McA. Johnston (Members of Council), A. J. R. Atkin, J. W. S. Beatty, J. H. Brown, J. Chilton, F. W. Cindel, W. M. Coulter, W. J. Creasey, T. D. Delprat, E. A. Furner, H. Goodwin, James Gray, Noel Griffin, B. C. Gullachsen, J. H. Harris, G. G. Hewitt, A. B. Inglis, G. A. Lawson, Henry Lea, J. Lea, A. Lilley, L. Marks, T. G. Martyn, G. Melvill, C. E Meyer, S. Newton, H. W. Pidsley, J. F. Pyles, Alex. Purser, W. Russell, G. A. Robertson, R. W. Robinson, O. D. Ross, A. Salkinson, C. O. Schmitt, A. Schwarz, A. M. Sim, G. Hildick Smith, A. L. Spoor, C. T. Stephens, J. A. Taylor, A. Thomas, G. R. Thompson, F. W. Watson, John Watson and C. F. Webb.

16 Associates and Students : Dr. J. L. Aymard, Messrs. E. A. Barry, J. P. Beardwood, H. G. Brickhill, C. A. Damant, F. E. Doble, H. J. Filmer, A. King, G. F. Mathews, R. A. Porter, A. G. Rusden, H. Rusden, H. Stadler, E. H. Tamplin, A. M. Thomas, and S. A. Woolf.

15 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as printed in the March *Journal*, were confirmed.

Messrs. H. Rusden and R. Allen were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows :—

Bell, Kenneth Somerset, P. O. Box 4281, Johannesburg. Mechanical Engineer.

Clesham, Thomas Henry, Simmer East, Ltd., P. O. Box 47, Germiston. Sampler.

Damant, Douglas Edward, Aurora West United G. M. Co., Ltd., P. O. Box 26, Maraisburg. Mill Manager.

Given, Ralph David, Messrs. H. Eckstein & Co., P. O. Box 149, Johannesburg. Engineer.

Kratz, Arthur Murray, Messrs. H. Eckstein & Co., P. O. Box 149, Johannesburg. Foreman Draughtsman.

Morrow, George William Offord, South Randfontein Deep, Ltd , P. O. Box 45, Randfontein. Foreman Timberman.

Peterson, Charles, Messrs. H. Eckstein & Co., P. O. Box 149, Johannesburg. Electrical Engineer.

Tester, William Andrews, Messrs. H. Eckstein & Co., P. O. Box 149, Johannesburg. Engineer.

Tom, Isidore, Messrs Lewis & Marks P. O. Box 1033, Johannesburg. Mining Engineer. (*Transfer from Associate Roll.*)

Webb, William Morton, Princess Estate and G. M. Co., Ltd., P. O. Box 112, Roodepoort. Mine Manager.

The Secretary : Since the last meeting of the Society the following have been admitted by the Council :—

As Associates.—

Purser, Alexander, East Rand Proprietary Mines, Ltd., P. O. Box 67, East Rand. Fitter.

Russell, William, 13, Park Circus, Ayr, Scotland (*temporary address*, Rand Club, Johannesburg.

Weeks, Robert Foster Jeffrey, Champion Reef G. M. Co., Ltd., Mysore State, S. India. Chief Reduction Officer.

As Student.—

Woolf, Samuel Adolf, 100, Van Beek Street, Doornfontein, Johannesburg. Student, S.A. School of Mines.

The President then distributed the book prizes presented by the Society to the successful evening class students at the recent examinations conducted by the S.A School of Mines, as follows: Chemistry, Stage I., Theoretical, £1 10s., S. Stein : Practical, £1 10s., J. P. Beardwood ; Stage II., Practical, £3, W. E John Metallurgy, Stage II. (Gold), £3, W. A Martin. Assaying, Stage I., £1 10s., J. Ingram. No awards were made for Chemistry, Stage II. (Theoretical); Metallurgy, Stage I.; Assaying, Stage II.; and Metallography.

THE ASSAY OF USED PLUMBAGO CRUCIBLES FOR GOLD.

By John Watson, F.I.C (Member).

This matter was brought before our Society, in the pre-war days, when Mr. Andrew F. Crosse one of our earliest presidents, read a paper entitled :—" Notes on Assaying Ground Graphite Crucibles." The subject was discussed at the meeting in September 1897, and in the following month, when Mr. Crosse replied to his critics. Curiously, since the war, this matter has not again been brought forward.

Plumbago crucibles are still in general use for melting the gold before pouring into moulds; though the larger number formerly used in the reverberatory furnaces in cyanide works for smelting zinc-gold slimes have been displaced largely by the almost general adoption of the Tavener, or pan furnace, and the capelling furnace.

The Phœnix of old was said to rise again from her own ashes, and some of us who are engaged in mine assay work are still called upon at odd intervals to assay this material which, after grinding and mixing with old cupels, " mabor " or cement, is now used as a bed, underneath the fire-brick lining of the Tavener pan furnace.

I suppose, of all the assays which the assayer on a mine is called upon to make, this material is one of the most tricky and ticklish to value with accuracy. It might be thought from the silence which has reigned during the eight or nine years of peace, that we have reached finality in this matter, and that we assayers all rest satisfied with the method of our esteemed Past-President, that in fact the last word had been said on this subject.

It recently fell to my lot to have to assay a parcel of some 2·2 tons of ground bricks from the bed of a Tavener furnace, containing a large proportion of old ground plumbago crucibles. A fair sample had been taken at the reduction works, buyer's and seller's representatives being present. This sample was properly mixed, pulverised to – 60 mesh, and divided into three portions. I used Mr. Crosse's method, working on four assay portions of ·30 assay ton each. I did *not* rely on the assurance that the lead buttons contain all the gold. Perhaps, because I am naturally somewhat sceptical, I kept the slag in each case, had it pulverised and assayed, and found quite a large proportion of the gold in the slag.

In the first portion about one-third of the gold found was in the slag; in the second portion, more than two-thirds; in the third portion, about one-third; and in the fourth portion, about two-fifths of the total gold found was in the slag.

The results of this parcel reported were as follows :—Fine gold per ton :

Seller.	Buyer.	Umpire No. 1.	Umpire No. 2.
107·38 oz.	127·2	85·85	130·00 oz troy.

The contents were paid for on the mean between the assays of umpires No. 1. and 2, *i.e.*, 107·92 oz.

The fine gold in this material is paid for at the rate of £3 17s. per oz. troy, therefore, it is highly important that we should have accurate assays.

Mr. Crosse, I hope, will not think me unduly critical of the method he described; on the contrary, when I had to assay this material, I felt thankful to have his paper at hand as something tangible to work upon.

Having now got a sample checked by two experts on this material—I mean the assayer of the buyer and umpire No. 2—I felt myself in possession of a valuable asset. This is a case where most of you will agree, I think, that the higher assays are the more likely to be correct. Experiments were tried using potassium nitrate and chlorate respectively, as oxidising agents, but these gave too low a result, especially the latter. Out of fairness to Mr. Crosse's method, I tried two more portions, using a smaller proportion of manganese dioxide; as his paper referred to ground graphite crucibles, not a mixture of that material with its own bulk of old cupels. These two portions, however, gave results, again, much too low from the buttons, and I did not try the slag from these.

Composition of Plumbago Crucibles—That veteran text book, *Mitchell's Manual of Practical Assaying*, edited by Sir Wm. Crookes (6th Edition, 1888) says :—" Black-lead crucibles are generally composed of 1 part of refractory clay, and from 2 to 3 of black-lead." Never having seen a published analysis of plumbago pots, I took a piece cut out of a No. 40 Salamander plumbago crucible of a well-known brand. This was pulverised to – 60 mesh and analysed. Its composition in the dry state I found to be :—

Carbon		42·95
Silica		34·68
Alumina and Ferric Oxide		18·08
Lime...		·32
Magnesia		·48
Alkalies, &c., undetermined	by difference	3·48
		100·00

It is the carbon in this material which forms the bugbear of the assayer, and I would recommend :—

Proc. Chem. and Met. Soc. of S.A., Vol. 11., p. 123., *et seq.*

(*a*) To work on small portions of the sample. say ·10 assay ton, in the case of a rich sample like this.

(*b*) To carry out the roasting operation with manganese dioxide in a roasting dish, the mixture being spread out in a shallow layer and roasted at a bright red temperature in the oxidising atmosphere of a muffle furnace for at least half an hour.

(*c*) To add enough reducer to bring down a fairly large lead, say 35gm. to 40gm. in weight, and, in all cases, to keep the slag, have it pulverised to – 60 mesh and assay for gold.

The following method will be found, I think, to give reliable results when properly carried out :

Place ·10 A.T. of the sample, with ·20 A.T. manganese dioxide, in a Wedgwood mortar and mix thoroughly, transfer to a 4 in. roasting dish, brushing clean the pestle and mortar, roast in muffle furnace at a bright red temperature for at least half an-hour, then withdraw and allow to cool. Scrape out the roasting dish with a small spatula, the contents being pulverised in a Wedgwood mortar, add 2 A.T. by measure of flux as used for daily routine samples ; add also 1 A.T. pulverised potassium carbonate, 1 A.T. silica pulverised, and 4 gm. mealie meal or flour. Grind the whole together in a Wedgwood mortar to thoroughly mix, transfer to paper bags and add silver (according to richness of sample). Introduce into red-hot H or J pots, heating for about 20 minutes, till fusion is complete, then withdraw the pots, tap gently, allow to cool in pots, when they can be broken. Cupel the leads, keep the slags and re-flux. Working on ·10 A.T. the ·01 gm. rider will give ounces to the ton direct.

Herewith I give comparative results :—By older method, as described in *Journal*, 1897 :—

From	I.	II.	III.	IV.	oz. troy
Button,	50·7 oz.	35·7	73·0	60·8	fine gold
Slag,	51 0	73·7	36·7	48·0	per ton.
Total,	101 7	109·4	109·7	108·8	

Average 107·40

By newer, modified method :

	I.	II.	III.	IV.	oz. troy
From	Oz. troy				fine gold
Button,	126·00	128·20	130·00	126·20	per ton.
Slag,	·20	·90	·25	·10	
Total,	126·20	129·10	130·25	126·60	

Average 128·04

My thanks are due to Mr. C. Glyn, General Manager of the City and Suburban Mine, for permission to publish these results

That great master of science and lucid English, the late Prof. Huxley, used to say that free and open discussion is the very life and soul of truth ; this Society has generally acted up to that admirable maxim. We have on the Rand several assayers who have had special experience in valuing the products I have brought to your notice this evening. I hope they will come forward and give us their best method. What we mine assayers want is a method which will give accurate results in the shortest possible time. I take it, that one of the chief advantages of belonging to a scientific society is, that there exists, or *should exist*, a feeling of freemasonry among its members. In these matters it would be well to emulate the spirit indicated by the late W. K. Clifford, when he said : "Let us take hand and help, for this day we are alive together."

Mr. Andrew F. Crosse (*Past President*) : In proposing a vote of thanks to the author for his paper, I would like to say that as my name has been mentioned several times by him I would mention that I altered my opinion as to my then proposed method. This fact is stated on p. 207 in the 2nd Volume of our Proceedings. The analysis of the plumbago crucible is interesting, though I expect the Morgan Crucible Co will be interested to learn that the alkalies, etc., amount to 3·48% by difference. I would advise that the publication Committee delete this statement, of course with the author's consent. The rest of the analysis might remain.

I quite appreciate the author's remarks about free discussion, and those here to-night who have been members for many years will allow that I for one have always given way when I have been proved to be in the wrong.

As regards rich bye-products, the highest assay is not necessarily the correct one. If ten assays are made each may be correct on the given sample, though none may agree.

Mr. J. Watson (*Member*) : With reference to Mr. Crosse's remark that the figure given for alkalies is too high, I must mention that the sample piece analysed had been cut out of an *old* pot, which had been used for smelting several times with borax, soda carbonate and possibly nitre as flux.

Mr. A. Whitby (*Member of Council*) : Having been furnished by the courtesy of the author with an advance-proof of his paper, I think I may be entitled thus early to offer a few remarks.

The author appears to be unfortunate in his choice of title. It is not plumbago crucibles he is assaying, but Tavener furnace bottoms, mixed according to the author with graphite crucibles. Even were he correct in this respect, he still has no justification for assuming that the assayer of 1911 rests satisfied with Mr. Crosse's method of 1897. The chief points to be observed in reading this paper seem to me to be :

1. That the author has only dealt with one sample.

2. That it was probably incorrectly labelled.
3. That he makes no mention of metallics.
4. That the sample only passed the 60 linear sieve.
5. That the analysis published of Salamander crucible is beside the question.
6. That the use of MnO_2 in roasting has no justification and that with such a small quantity as ·1 A.T. direct fusion with red lead is more satisfactory using a little Na_2CO_3 and borax.

It would be manifestly unfair to the assayer to submit a sample of such complex nature without precise information as to its derivation since an analysis is not paid for. Why the author uses such an enormous excess of flux, 2 A.T. of the daily routine flux (whatever that may be) 1 A.T. of K_2CO_3 and 1 A.T. of SiO_2 in order to melt down 0·1 A.T. charges or at most 0·2 A.T. is beyond comprehension.

Setting aside any question of imperfect sampling, I think that had the various assayers concerned being supplied with correct data there would have been no necessity for this paper, and certainly I can see no reason for such unwieldy treatment, always preferring to flux direct when ·1 to ·2 A.T. charges require to be taken.

I may remark that plumbago pots themselves when properly scraped usually do not assay more than 4 oz. and present no difficulty in roasting. When not freed from adhering slag I admit they are troublesome and cannot be roasted with or without MnO_2. Personally I should not use MnO_2. One is compelled in such a case to the use of small charges and direct fusion. I would like here to make a suggestion which I fancy is novel. Expense being hardly the main point it would almost seem that there is a field in such products as plumbago pots or even rich concentrates for the use of the puce-coloured oxide of lead PbO_2 instead of the other oxides. We know that even with such a slight relative preponderance of oxygen, red lead possesses a distinct advantage over litharge, therefore why not use the dioxide in cases where the ore has reducer in excess.

One further remark I must make. No assayer having such rich material to work upon would be correct in neglecting his slags.

Mr. J. Watson: In reply to Mr. Whitby's remark that he considers the choice of the title of this paper unfortunate, I have often assayed other samples from Tavener furnaces and have agreed all right with the reduction works assayer, except when the product contains these old plumbago crucibles. It is then only that I have found differences in our results to occur. Mr. Whitby suggests that the sample was wrongly labelled. I personally was not responsible for the marking of the samples. As a matter of fact this particular sample was marked: "Brick, Lot 2." I asked our Mine Secretary to inform both the umpires, when the sample was sent, that it contained a considerable proportion of old plumbago crucibles, and he assures me that he did so, in both cases.

Mr. A. Whitby (contributed): It appears from Mr. Watson's own showing that the material could not have contained, being constituted, say, of 50% brick and 50% pot carrying 47% graphite, more than, say, ·75 gm. to 1 gm. of carbon giving a 25 gm. button at most with litharge. Therefore I must repeat that no roasting was required and there was no necessity for reducing agent or complexity of flux. His error arises from inefficient crushing, imperfect mixing and bad fluxing. The question of mesh is serious. No such material could be obtained through a 90 or 120 mesh sieve without yielding metallics, and practice indicates the use of the fine mesh in all cases where values rise above 50 oz. to the ton.

FUTURE ECONOMIES IN RAND REDUCTION PLANTS.

By C. O. Schmitt (Member.)

Introduction:—It is an acknowleged fact that with the ever increasing depth of the mines on the Witwatersrand, with a consequent increase in the cost of bringing the gold bearing ore to the surface, and accompanied by a steady but insistent decrease in the value of the ore deposits, the profit derivable from the exploitation of the mines at great depths tends to taper off and must disappear entirely when working costs plus capital charges equal the value of the gold recovered.

There is not sufficient information available to give the average screen value of the ore for the Witwatersrand for all depths, but from the published returns of the Chamber of Mines it appears reasonable to estimate it as shewn in Table I:*

At the present day the average value of the gold recovered per ton of ore milled is about 28s.

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* It should be clearly understood that the screen value of the ore given in Table I. is not based on any particular section of the Rand nor obtained from any definite mines, being merely the writer's interpretation of the published returns for the whole Rand, which obviously do not make any allowance for a deliberate reduction in grade rendered possible by the low standard of working costs now prevailing. The writer, in order to more clearly show the benefits derivable from the proposed innovations, has naturally taken a low value for the ore derived from great depths which, together with the estimated increase in mining costs, emphasises the importance of reducing working costs on the surface. Apart from this, the value of the ore has no bearing on the subject of these notes.

TABLE I.

Average Depth of Workings.	Screen Value of Ore per ton Milled.	Screen Value of Ore per ton Hoisted assuming that 15% of Waste is Sorted Out.
feet	s. d.	s. d.
1,000	40 0	34 0
2,000	30 0	25 6
3,000	25 0	21 3
4,000	22 6	19 0
5,000	20 0	17 0

and working costs, including development redemption, but not including capital charges on the surface equipment. are about 17s., leaving a gross profit of 11s., of which about 9·9s. is available for distribution, after allowing for the profit tax.

It is impossible to estimate the average depth from which ore is mined at the present day, but from Table I. it would appear that it is in the neighbourhood of 2,100ft. At any rate it is known that while some mines work at a comparatively shallow depth others obtain the ore from below 3,500ft. and even 4,000ft.

If working costs remain at the present standard the end of profitable mining would be reached at or before a vertical depth of 5,000ft., and at the present rate of working the time needed to reach this point can readily be estimated. Whatever this period of time may be, it is evident that in order to lengthen it, it will be necessary to reduce operating costs and capital charges. Unfortunately the capital spent on existing plants can not be affected by any improvements to be made and, if anything, capital charges will be greater in these cases should further capital be required to give effect to necessary improvements. In other words, as far as existing plants are concerned, any benefits derivable from improvements are minimised by additional capital charges, although it will probably be seen that these benefits will more than pay for the increased capital charges. On the other hand new mines, not yet equipped on the surface, will fully share the advantages to be derived from the improvements to be indicated later on.

It will be interesting to recall at this point the position of the mining industry on the Witwatersrand in 1905, as shown by Ross E. Browne[*] in a paper read before the South African Association of Engineers. He gives the following (p. 290):—

	s.
Working costs	24·0
Capital redemption ..	5·0
Total expenses	29·0
Yield	35·7
Profit	6·7
Profit tax 10%	0·7
Company's profit	6·0

At this period (September 1905) the mines were working at considerably less depth than at present, yet the profit, if we ignore the capital redemption charge of 5s. was not much greater than to-day as appears below:—

	s.
Yield	35·7
Working costs	24·0
Profit	11·7
Profit tax 10%	1·17
Company's gross profit ...	10·53

Although the yield fell from 35·7s to 28s. the gross profit only fell from 10·53s. to 9·9s., this favourable result being due entirely to the reduction in working costs.

If the redemption of capital is taken into account the position is at the present day even more favourable. R. E. Browne gives the capital expenditure for the average plant of the Rand Mines Ltd., milling 15,654 tons per month with 131 stamps at £620,132 or £1,127 per ton milled per day, of which £356 was for the reduction plant and £771 for the remainder, including shaft sinking and development prior to starting milling operations.

Omitting the cost of development the capital expenditure was £430,000 or £782 per ton milled per day, of which £356 was for the reduction plant, leaving £426 for the remainder. Assuming that this plant represented the average for the whole Rand at that time (1905) it can be compared with the average at the present day. As a considerable number of mines have started milling since 1905, all of which were able to build the surface plant at less cost, the charge for capital redemption, which Ross E. Browne takes at 5s. can be reduced, and it remains to be seen by how much. The tonnage milled has increased from 1,000,000 tons per month in September 1905 to 1,750,000 tons per month in January 1911, and while the cost of reduction plants was £356 per ton milled per day in 1905, the cost of the additional reduction plants may be taken at £200 per ton milled per day. The total expenditure per ton milled per day would now be £426 and £200 = £686 as against £782 and corrected for the increased tonnage an average for the whole Rand, at the present day, of £716.

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[*] Ross E. Browne "Working Costs of the Mines of the Witwatersrand," *Journal of the S. A. Association of Engineers*, Vol. XII. p. 289.

If a capital expenditure of £782 requires a redemption charge of 5s., £716 requires only 4·5s., and we can now compare the position of to-day with 1905:—

	1905. s.	1911. s.
Yield	35·7	28·0
Working costs	24·0	17·0
Capital redemption ...	5·0	4·5
Total costs	29·0	21·5
Profit	6·7	6·5
Profit tax	·67	·65
Company's profit ...	6·03	5·85

Thus, although the yield per ton fell 7·7s. the profit only fell 0·18s. This result has been obtained chiefly at the expense of mining costs, and it is hardly to be expected that mining costs will be able to keep pace with the reduction in grade due to the lower value of the ore deposits at greater depths, in fact, it can safely be assumed that mining costs will go up rather than decline for many reasons, but chiefly the following:—

(*a*). Increased cost of skilled and unskilled labour.
(*b*), Do. do. explosives.
(*c*). Do. do. ventilation.
(*d*). Do. do. mine drainage,
(*e*). Do. do. hoisting rock to surface.
(*f*). Do. do. underground transport
(*g*). Do. do. power underground.
(*h*). Do. capital charges.

Labour:—The scarcity of native labour used for drilling, etc., tends to raise the cost of same apart from the fact that with the increased depth of the mines, more time is lost in reaching the working face, the latter factor also applying to skilled miners. Further, with increased depth the temperature rises, and labour is liable to become less efficient.

Explosives:—Owing to the increased hardness of the rock to be broken more explosives will be needed and the increased demand together with an increase in the price of the raw materials, particularly glycerine,* tends to raise the cost per unit.

Ventilation:—Outcrop mines and shallow deep levels have not had to provide for artificial ventilation, but the deeper mines are compelled to adopt same in order to maintain the efficiency of the labour force quite apart from general considerations of health and safety. The cost of this item alone is liable to be from 2d. to 3d. per ton hoisted.

Mine Drainage:—Although as a rule the cost of mine drainage is not heavy on the Rand the cost must obviously increase with the depth of the workings.

Hoisting Rock to Surface:—The cost of hoisting rock will undoubtedly increase with the depth and at an accelerated rate for several reasons. It is now generally acknowledged that the most economical depth or length of lift for a single stage of hoisting is about 3,500ft., either in the vertical or on the incline, and if we take a mine with vertical shafts 3,000ft. deep, two stages of winding on the incline will be required to reach a depth of 5,000ft. vertical, making in all three stages. Although less power is required for hoisting 3,000ft. on a comparatively flat incline than in a vertical shaft 3,000ft. deep, yet the cost of hoisting per ton will be as great if not greater due to greater wear on skips, ropes and track, apart from the increased cost of power due to duplicate cables, losses in transmission, etc.

Underground Transport:—Shafts will in deep mines be farther apart than is now the case, due to the great cost of sinking same, with the result that the distances over which ore must be transported are increased.

Power:—The cost of power must increase, as already pointed out under "Hoisting." This will be most evident in the cost of compressed air, because not only will frictional losses increase due to longer pipe lines, but more air will be required for drilling holes in the harder rock.

Capital Charges:—The expenditure on winding plant, power mains, air mains, water mains, ventilating plant, shaft sinking, naturally increases with the depth of the workings and this in no small degree.

At the present date working costs average 17s. per ton milled, made up as follows:

	s.	d.
Mining costs	8	9
Reduction costs	5	0*
General expenses	1	3*
Development redemption ...	2	0

As already said (page 465) the average depth of the workings is probably 2,400ft., and estimating that mining costs to increase at the rate of 6d. per ton milled for every 1,000 feet increase of depth we obtain the data shewn on Table II. The diagram, Fig. 1, shews this graphically. It is obvious from the table that unless it is possible to reduce working costs and capital charges mining ceases to be profitable at the comparatively shallow depth of 4,000 ft., because redemption of capital alone will swallow up the profit.

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* W. Cullen, "The World's Glycerine Supply," this *Journal*, Vol. XI., p. 302.

* E. J. Way, Trans. Inst. M. & M., Vol. XIX., p. 120.

TABLE II.

Probable Net Profit from Mining Operations at Various Depths.

1.	Average depth of working...	1,000 ft. Shillings.	2,000 ft. Shillings.	3,000 ft. Shillings.	4,000 ft. Shillings.	5,000 ft. Shillings.
2.	Mining costs	8·0	8·5	9·0	9·5	10·0
3.	Reduction Costs	5·0	5·0	5·0	5·0	5·0
4.	General expenses ...	1·25	1·25	1·25	1·25	1·25
5.	Development redemption ..	2·0	2·0	2·0	2·0	2·0
6.	Total costs	16·25	16·75	17·25	17·75	18·25
7.	Yield	40·0	30 0	25·0	22·5	20·0
8.	Profit	23·75	13·25	7·75	4·75	1·75
9.	Profit tax 10%	2·375	1·325	·775	·475	·175
10.	Company's profit ...	21·375	11·925	6·975	4·275	1·575

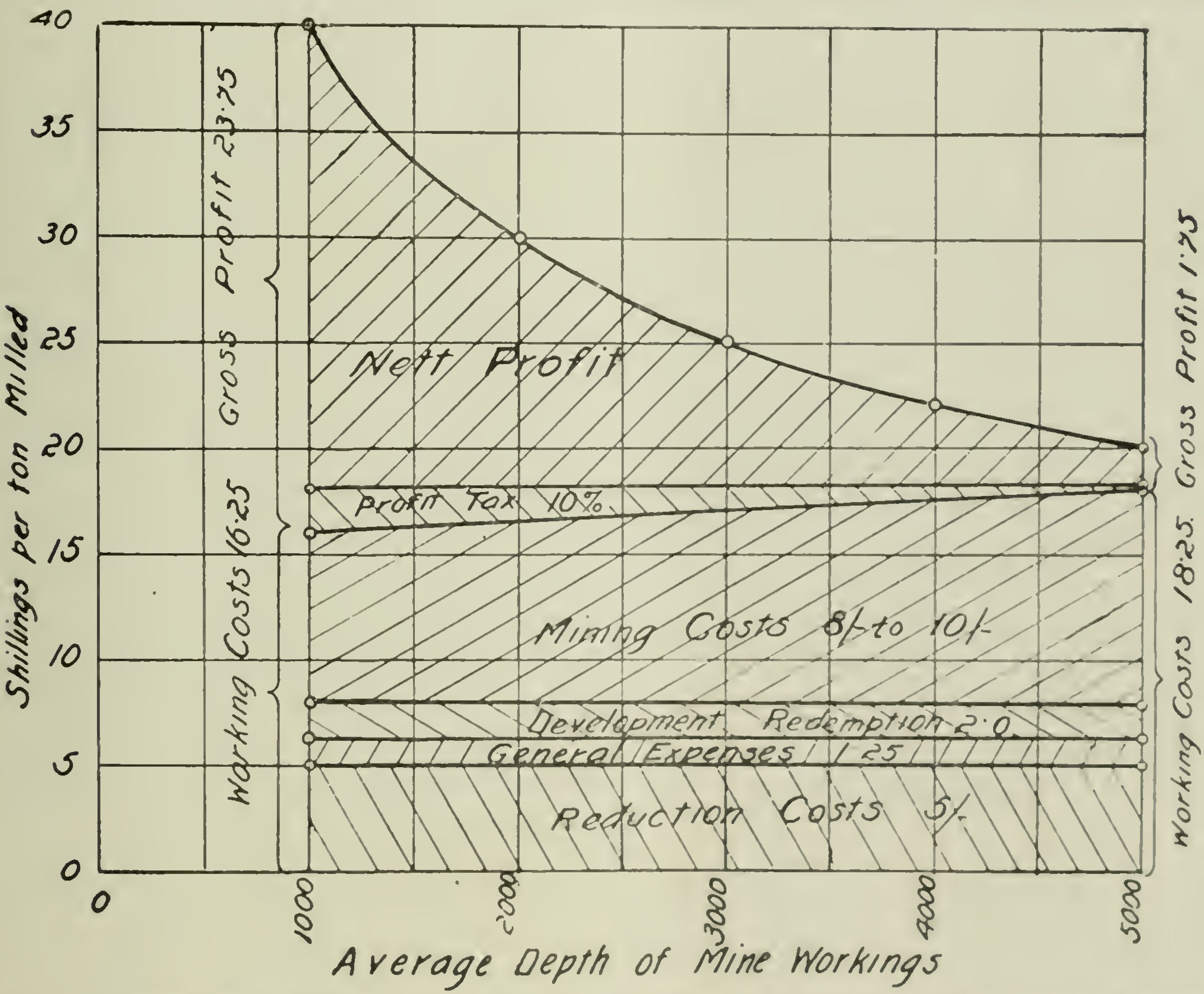

FIG. 1.—Diagram showing Screen Value of Ore, Working Costs, and Profit per Ton Milled when Working with Modified Sorting and Breaking Plant.

It might be pointed out that the assumed increase in mining costs of 6d. per 1,000 ft. is too high, and the writer hopes that this is the case, but in view of the foregoing remarks it seems reasonable to assume that 6d. per 1,000 ft. of depth is not by any means an exaggerated amount. No doubt the tendency towards increase can be counteracted to a certain extent by mining a greater tonnage, but this carries in its wake the danger of mining barren rock, which must be a real temptation to underground workers. However, as the grade would be lowered, nothing would be gained beyond distributing general expenses over a greater tonnage, and a probable loss would result from the increased quantity of waste rock sent to the reduction plant. Under the circumstances, the adoption of what is known as the big mill policy, first put to the test on the Rand by Mr. H. H. Webb on behalf of the Consolidated Gold Fields of South Africa, Ltd., was an obvious step to reduce working costs, and the results have been satisfactory. However, the successful carrying out of this policy requires a very considerable amount of advance development, for, unless the mine is well opened to conveniently supply the mill with the increased tonnage without being tempted to mine barren rock, the object in view is defeated.

At the present day the standard for comparison generally adopted on the Rand is the ton milled, but it has already been pointed out by various writers that this is not well chosen for comparing costs and efficiency of working. In the first place, the tonnage milled by no means expresses the work done underground, because the difference between the tonnage hoisted and milled, representing the waste sorted out, varies considerably in different mines, namely, from the practically nothing to over 30%. It seems fair to assume, always provided the work underground is carried out honestly and with every endeavour to reduce the quantity of barren rock sent to the surface, that the tonnage hoisted forms a much better basis for comparison, and this standard the writer proposes to use in the following investigation.

Seeing that mining costs have a tendency to increase, it will be necessary to counteract this on the surface by reducing operating costs in and capital charges for reduction plants. On p. 465 has been shown that capital expenditure for the reduction plant is a comparatively small portion of the total expenditure. Using again the figures of Ross E. Browne, we have for the average mine of the Rand Mines, Ltd., but leaving out shaft sinking and development, the following :—

Surface Equipment ...	£426 = 55%
Reduction Plant ...	£356 = 45%

It has been estimated that at the present day, for the whole Rand, the proportion would be :—

Surface Equipment ...	£426 = 60%
Reduction Plant ...	£290 = 40%

Assuming these figures as correct, any saving to be effected in the reduction plant would therefore only be a percentage of the 40%, the amount represented by the reduction plant, so that a considerable reduction must take place to materially affect the whole capital expenditure. The position is still worse if we include shaft sinking and advance development.

Referring to working costs, the position is also unfavourable if substantial reductions are to be made, because reduction costs are barely one-third of the total costs, as appears from the following :—

	Shillings.		
Mining Costs	8·75	10·75	11·5 = 68%
Development Redemption ...	2·00		
		·75	
General Charges	1·25	·50	5·5 = 32%
Reduction Costs	5·0		

Therefore, any decrease in reduction costs affect total costs only in the proportion mentioned.

Under the circumstances, the task of substantially reducing working costs and capital charges is by no means an easy one, but in view of the magnitude of the interests involved, worthy of the most serious consideration by all interested in the mining industry.

Considering an investigation of the whole subject as beyond the scope of these notes, the writer proposes to deal primarily with the sorting and breaking plant and the milling or crushing plant, merely indicating economies in other portions of the plant at the hand of data published in current papers.

I.—*Sorting and Breaking Plant.*—The problem of removing the waste or barren rock from the gold bearing ore has agitated the minds of engineers on the Rand for a considerable time, as evidenced by the papers read and discussions which have taken place before local Societies.[*] It might seem therefore that the subject has been considered from every point of view, making further remarks superfluous. In view of certain developments in the milling or crushing plant in the immediate past, the writer thinks that reconsideration in the light of the altered conditions will pay handsomely for the trouble taken. Members are probably familiar with the now acknowledged fact that the efficiency of the tube mill considerably increases if fed with a comparatively coarse product. Dr. Caldecott[†] referred to this in detail in the reply to the discussion on his paper on the development of the heavy gravitation stamp, aud considerable further light has been thrown on the point by E. H. Johnson[‡] in the published results of a most succesaful crushing experiment at the East Rand Proprietary Mines. Seeing that it is possible to feed the tube mill with anything smaller than $\frac{3}{8}$ in., it occurred to the writer that a convenient place to obtain same was the sorting and breaking plant. The rock brought from below ground contains a considerable amount of fine particles which are not generally eliminated by screening the total rock into a $+2$ in. and -2 in product. The latter product, however, also contains a good deal of $-\frac{3}{8}$ in., and gradings obtained by the writer from two mines showed following result :—

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[*] Robert Dives, Ore Sorting and Sorting Appliances, *Journal of South African Association of Engineers*, Vol. III., p. 138. Wager Bradford, Some Limitations of Ore Sorting, *Journal of South African Association of Engineers*, Vol. IX., p. 61. H S. Denny, Observations on the Metallurgical Practice of the Witwatersrand, this *Journal*, Vol. IV., p. 116. H. S. Denny, Economic Problems of Metallurgy, Report of S.A.A.A.S., 1904, p. 109. G. A. Denny, Valedictory Address, *Journal of South African Association of Engineers*, Vol. XI., p. 371. W. Calder, *Journal of Transvaal Institute of Mechanical Engineers*, Vol. VII., p. 241. C. O. Schmitt, Sorting, an Economic Problem on the Rand, *Journal of South African Association of Engineers*, Vol. XV., p. 151.

[†] Trans. Institution of Mining and Metallurgy, Vol. XIX., p. 57.

[‡] This *Journal*, Vol. XI., p. 168.

	Mine A.		Mine B.	
$+\frac{3}{8}$ in. ...	58·3%		45·38%	
$-\frac{3}{8}$ in. $+90$ ..	38·3	41·7%	46·92	54·62%
-90 ...	3·4		7·7	

In these cases the average of $-\frac{3}{8}$ in. may be taken at 50% of the total -2 in. product, in spite of the imperfect screening now taking place over stationary grizzlies, and notwithstanding the fact that the rock was not washed. As the -2 in. product represents from 35% to 50% of the total rock hoisted, the $-\frac{3}{8}$ in. therefore equals from 17·5% to 25% of the total tonnage, the whole of which can be sent directly to the tube mill plant. If coarse breakers are in use, they naturally produce a further percentage of $-\frac{3}{8}$ in., and so will the fine breakers, the total percentage so obtained probably not falling far short of that produced underground by the explosives. G. O. Smart* found in his battery feed not less than 33% of $-\frac{1}{2}$ in., corresponding to probably 28% of $-\frac{3}{8}$ in., while W. R. Dowling found at the Knights Deep, Ltd., 26% of $-\frac{1}{2}$ in. and probably 20% of $-\frac{3}{8}$ in. This shows that in no case need less than 20% or even 25% of $-\frac{3}{8}$ in. be expected, especially seeing that we have already from 17·5% to 25% in the under-size product from the grizzlies. The present arrangements could, however, be so modified by adding more breakers that even more than 25% of the rock hoisted could be reduced to a size not exceeding $\frac{3}{8}$ in., thus still further increasing the quantity available for feeding direct to the tube mill plant.

The rock hoisted also contains a quantity passing a 90-mesh screen, and this product can be sent direct to the cyanide plant. In the two gradings previously given the average is over 5%, while G. O. Smart, at the Simmer and Jack Proprietary Mines, found 4·38% in the mill feed, and W. R. Dowling, at the Knights Deep, Ltd., 3·44%. Naturally this product is not wanted in either stamp-mill or tube mill plant.

Before describing the proposed modifications of the sorting and breaking plant, it will be as well to shortly state the present practice. The rock mined is sent to the surface after passing through grizzlies spaced about 8 in. On arrival in the head frame, it is automatically dumped upon a grizzley, the bars of which are generally spaced $1\frac{3}{4}$ in., where it is divided into a coarse product ($+1\frac{3}{4}$ in.) and a fine product ($-1\frac{3}{4}$ in.), called "fines." These "fines" are by no means cubical or spherical in shape, as, owing to the design of the screening device, long flat pieces, the bulk of which considerably exceeds a $1\frac{3}{4}$ in. cube, are passed through. The "fines" so produced are invariably taken direct to the battery, it having been taken for granted that it is impossible to remove any waste rock therefrom. The quantity of "fines" so produced varies considerably in different mines—partly due to the different spacing of the grizzley bars, but also due to different methods of mining—and may be as high as 50% and as low as 30% of the total rock hoisted.

The coarse rock remaining on the grizzley bars, after having been washed in a more or less summary manner, is spread out upon a sorting belt or table, with the object of facilitating a close scrutiny for detecting and discarding the waste rock. The waste so discarded varies from under 10% to over 30% of the total hoisted, the average for the Rand being about 16%. The rock or ore remaining on the sorting belt is then broken to a size not exceeding 2 in., and is sent to the battery. The complete operations involved are shown on the volume and flow diagram Fig. 2, while a schematic illustration of a typical plant is given in Fig. 3.

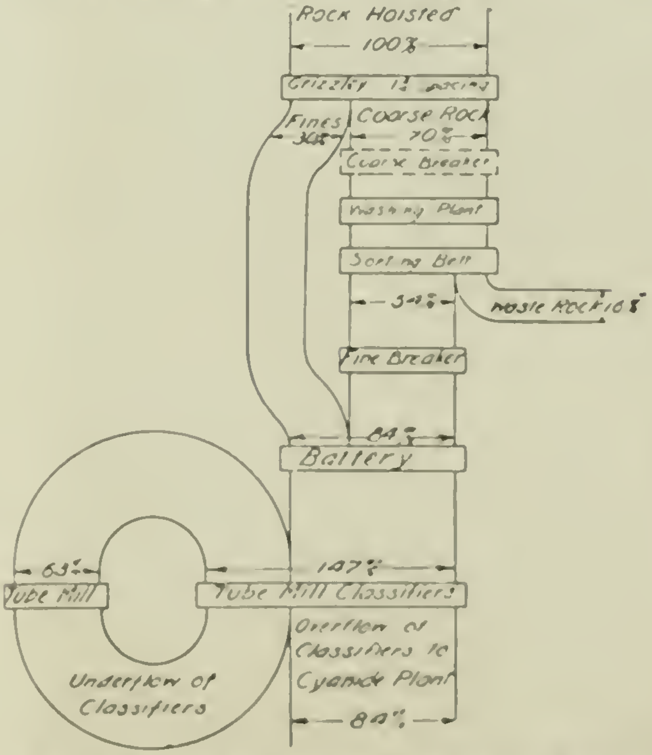

Fig. 2.—Flow and Volume Diagram for Sorting and Breaking Plant, Present Day Practice.

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* This *Journal*, Vol. XI., p. 325.

In some plants coarse breakers are used prior to sorting and the "fines" so created are generally added to the undersize product from the grizzleys. This type of plant is shewn in Fig. 4.

A small quantity of ore is sent direct to the tube mill plant in the shape of pebbles and these are picked on the sorting appliance, but as the quantity is very small it is ignored in the volume diagram.

As may be seen the arrangements are very simple in character, and although in isolated

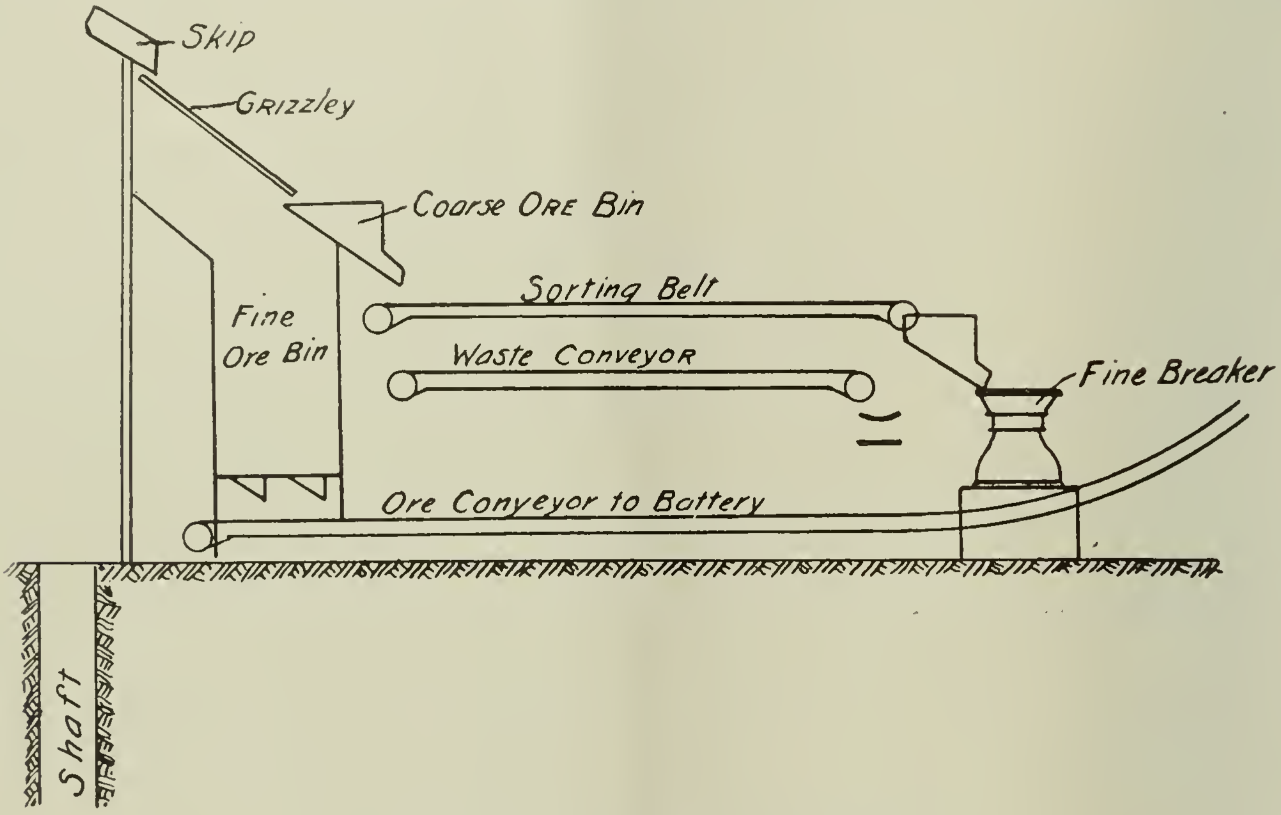

Fig. 3.—Present Type of Sorting and Breaking Plant.

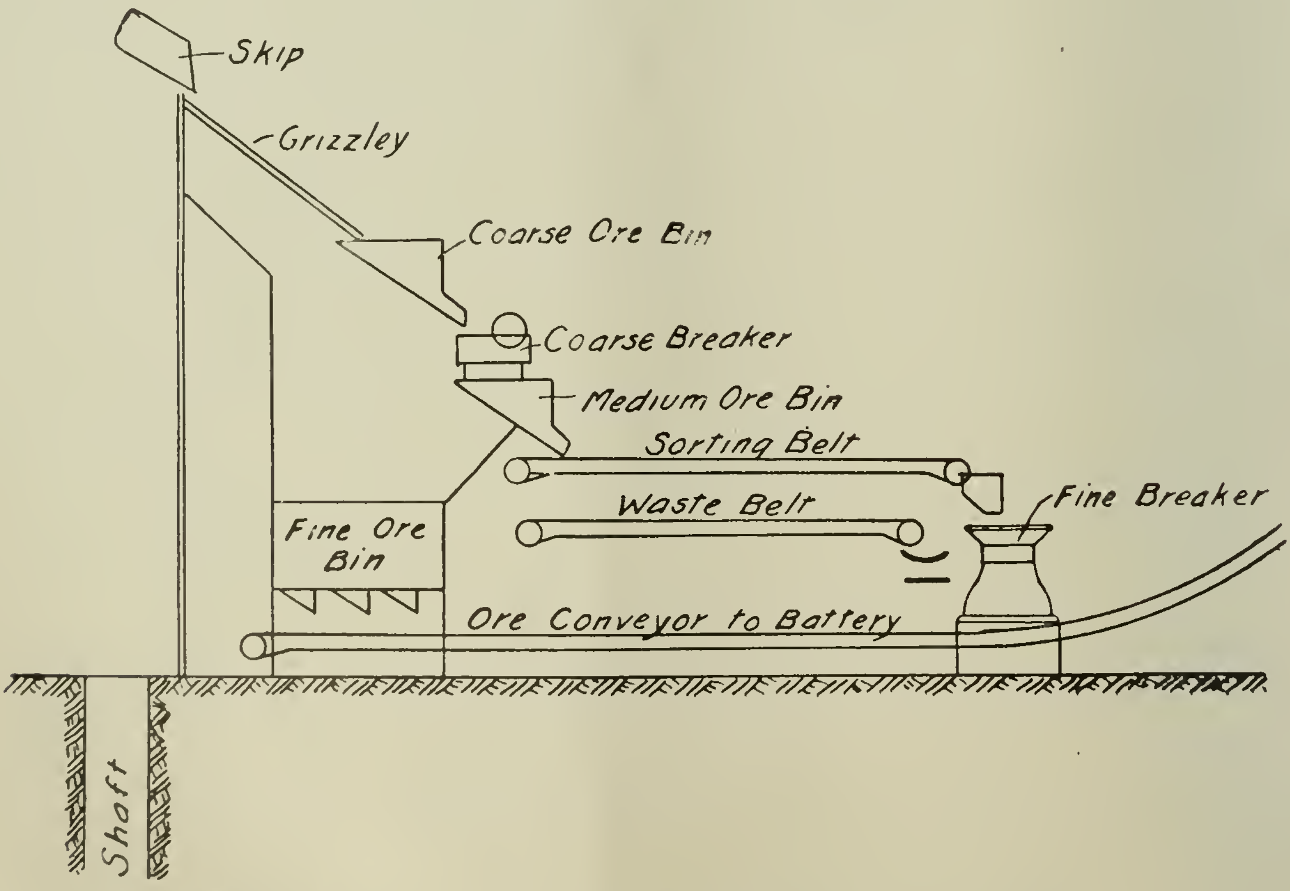

Fig. 4.—Present Type of Sorting and Breaking with Coarse Breaker.

cases more elaborate installations are found, the benefit which might be derived therefrom is altogether negatived by inefficient labour. It is notorious that only those natives who are physically unfit for other work are used for sorting purposes, although it would appear that, in view of the importance of the work and its subsequent influence upon the economy of the whole reduction plant, the work should be done by competent men. Further, the task is made still more difficult by too much crowding of the rock on sorting belt or table and by insufficient cleaning. Yet, under these distinctly unfavourable conditions an average of 16% of the rock hoisted is sorted out, and as no protest is made regarding the value of the waste dump, it is to be presumed that only waste rock is discarded. Now, provided that better labour is employed, that the rock is properly spread out and, above all, thoroughly cleaned, it does not seem impossible to raise the percentage of waste rock discarded to 20% or only 4% more than the average, always presuming that the quantity of waste rock mined and hoisted remains as before. The total waste in the rock hoisted may be estimated at from 35% to 40% on the average, but in many cases, when very small reefs are dealt with, it is much greater.

The rock hoisted is separated into a coarse and fine product, as has been stated, by stationary grizzleys when flowing at a great velocity down an incline of 38°. That the screening under these conditions is and must be imperfect is obvious and can at any time be seen by inspecting the "coarse" product upon the sorting belt or table. To improve this it would seem necessary to adopt a better screening device which may be either a shaking screen or a revolving screen, called trommel, the main object being to allow sufficient time to effect the separation and, incidentally, to permit of thoroughly cleaning the rock from all impurities and finely crushed particles. The time factor has so far been entirely ignored, except in rare cases, seeing that the time occupied by the rock in flowing over the grizzley never exceeds two seconds, while actually from 1$\frac{1}{2}$ to two minutes are available between the arrival of two succeeding skip loads. It would seem possible, therefore, to obtain a better separation into two products, say a coarse or $+2$ in. and a fine or -2 in. product and yielding, each, say 50% of the total rock hoisted, especially seeing that with the primitive arrangements now in use this is already done in many cases.

Having thus obtained the two products, the coarse or $+2$ in can be sent to the sorting appliance where, as already stated, 20% of the total hoisted will be removed as waste rock, the balance of 30% to be broken ready for the reduction plant.

The fine or -2 in. product, which contains proportionately as much waste rock as the coarse product must now be prepared to allow of eliminating as much as possible of the waste. To do this it is proposed to immediately screen out all particles too small to allow of conveniently detecting and discarding waste rock. It will therefore, be necessary to provide another screening trommel or other equally efficient device which will remove all particles $\frac{3}{8}$ in. and smaller, thus giving a fine product ready for the tube mill plant. In view of the gradings given on page 9 it seems that the $-\frac{3}{8}$ in. product will equal fully 15% of the total rock hoisted, thus leaving 35% of (-2 in. $+\frac{3}{8}$ in.) The coarser product (-2 in. $+\frac{3}{8}$ in.) can now be again washed and then subjected to a close scrutiny for detecting and eliminating the waste rock. Seeing the much greater facility of removing waste rock from this material without also discarding ore than is the case when dealing with coarse rock, it would seem possible that waste rock equal to 10% of the total quantity hoisted can be so removed, thus making the total waste discarded equal to 30% of the rock hoisted. With these modifications the flow and volume diagram would now be as shewn in Fig. 5.

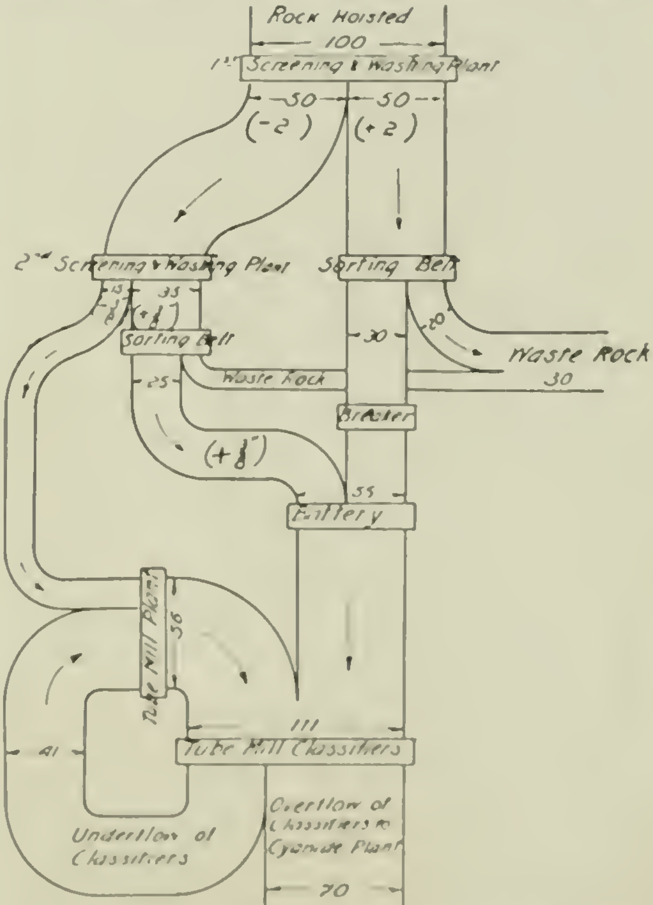

Fig. 5.—Flow and Volume Diagram for Modified Sorting and Breaking Plant on Basis of 30% Waste Rock.

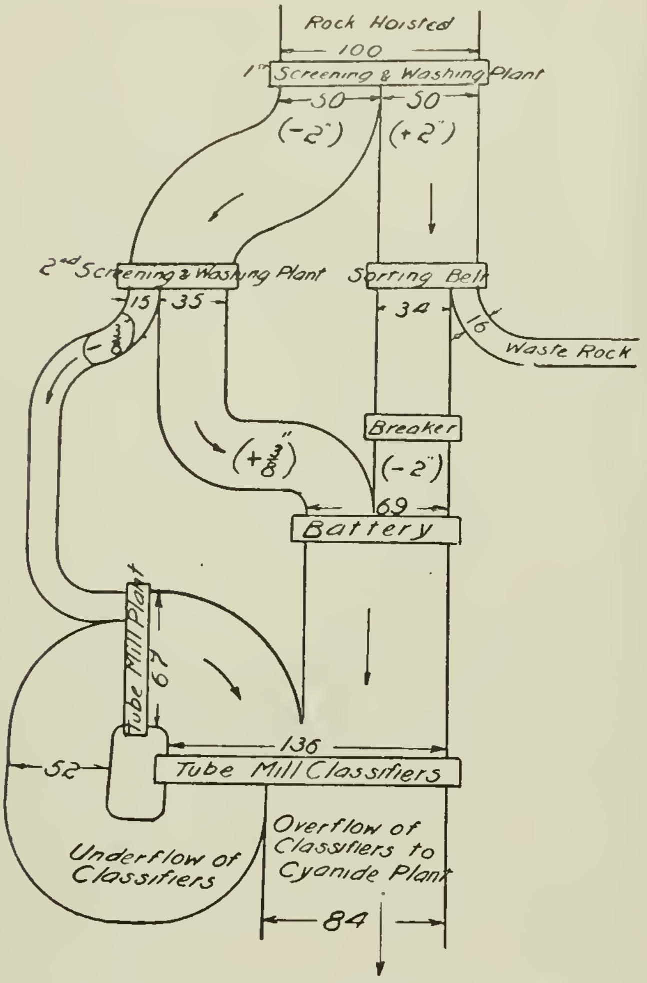

FIG. 5A.—Flow and Volume Diagram for Modified Sorting and Breaking Plant on Basis of 16% Waste Rock.

Comparing Fig. 5 with Fig. 2 it will at once be seen that of the total rock hoisted the battery now has to deal with only 55% as against 84% previously and that the tube mill plant only deals with 56% as against 63%, while the cyanide plant need only treat 70% as against 84%,

Although the battery has now been considerably relieved, yet a further reduction in its duty can be made by eliminating all the $-\frac{3}{8}$ in. product resulting from the operation of the fine breaker on the coarse rock, after sorting. The $-\frac{3}{8}$ in. product so obtained, and which may be estimated at 3% of the total hoisted would also be sent direct to the tube mill plant as shewn in the diagram Fig. 6. The work of the battery is now limited to 46% of the total rock hoisted and that of the tube mill plant rises only by 2·5% to 58·5%,

If it should be thought that the (-2 in. $+\frac{3}{8}$ in.) product from the fine breaker could with advantage be again subjected to sorting it could easily be added to the similar product from the first screening plant and the plant resulting would be represented by the flow diagram shown in Fig. 7. Compared with Fig. 6 there will be two screening plants, thus making a simpler arrangement, and while all the (-2 in. $+\frac{3}{8}$ in.) product is sorted in one place the product resulting from the 30% passed through the fine breaker is sorted twice.

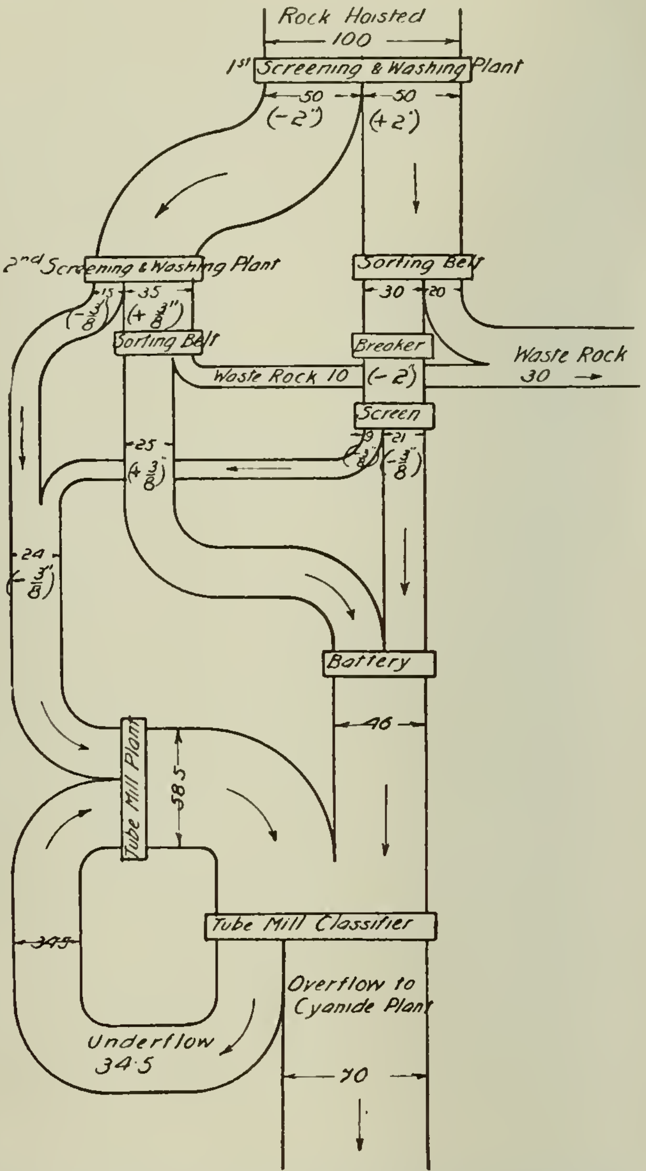

FIG. 6.—Flow and Volume Diagram for Modified Sorting and Breaking Plant with Additional Screening after Fine Breaking, on Basis of 30% Waste Rock.

If, as in some plants now in operation, it should be thought advisable to use a coarse breaker it becomes necessary to eliminate the fine product so created and to particularly see that the $-\frac{3}{8}$ in. product is sent direct to the tube mill plant. The diagrams Figs. 5, 6 and 7 would then have to be altered as shewn in Figs. 8, 9 and 10, the assumption having been made that the coarse breaker produces 10% of the total rock hoisted in a size smaller than 2 in. of which 4% again would be $-\frac{3}{8}$ in.

The diagrams shew all operations up to and including stamp milling and tube milling thus clearly expressing where the work is reduced as compared with present day practice. In explanation of the duty allotted to the tube mill plant other than crushing the $-\frac{3}{8}$ in. product obtained direct from the sorting plant, it may be stated

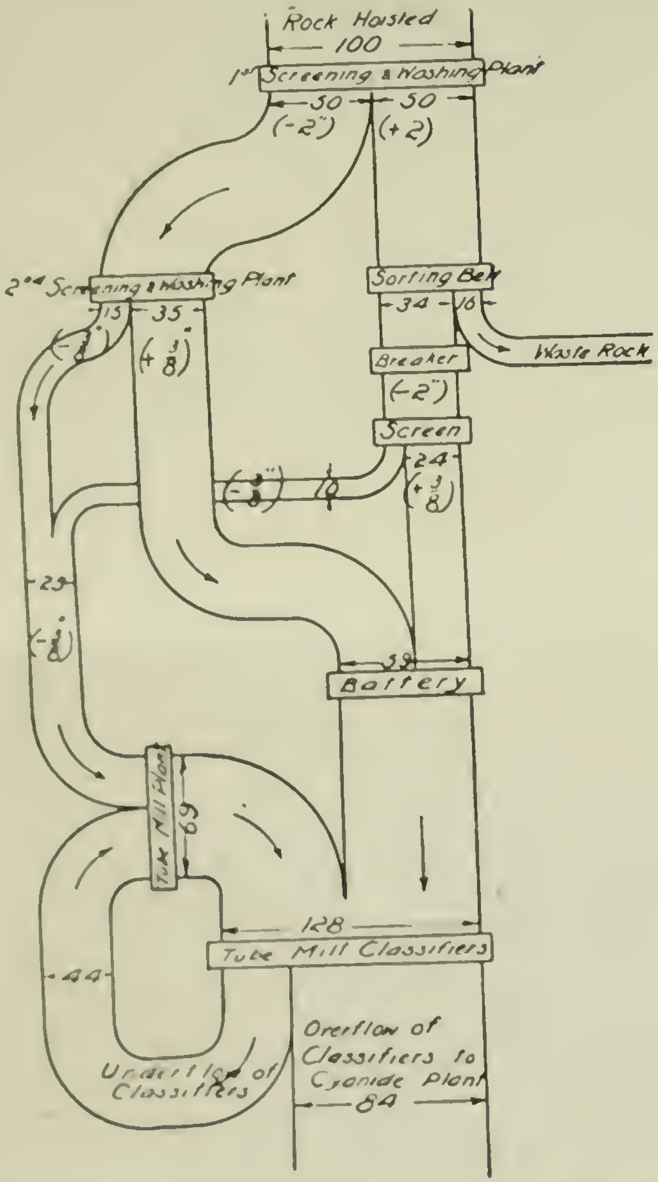

FIG. 6A.—Flow and Volume Diagram for Modified Sorting and Breaking Plant with Additional Screening after Fine Breaking on Basis of 16% Waste Rock.

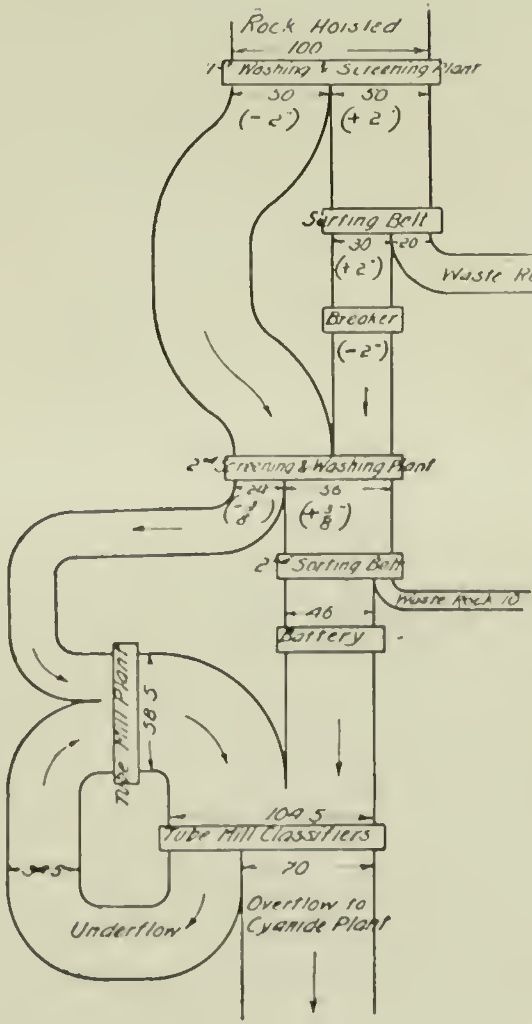

FIG. 7.—Flow and Volume Diagram for Modified Sorting and Breaking Plant with Double Screening and Double Sorting, on Basis of 30% Waste Rock

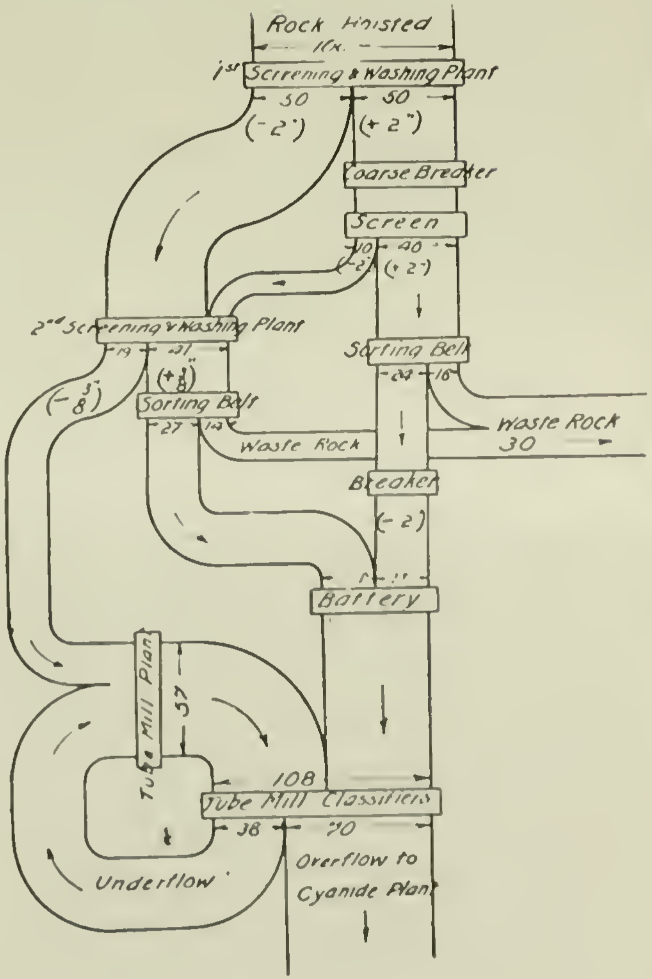

FIG. 8.—Flow and Volume Diagram for Modified Sorting and Breaking Plant as in Fig. V, on Basis of 30% Waste Rock but with Coarse Breaker

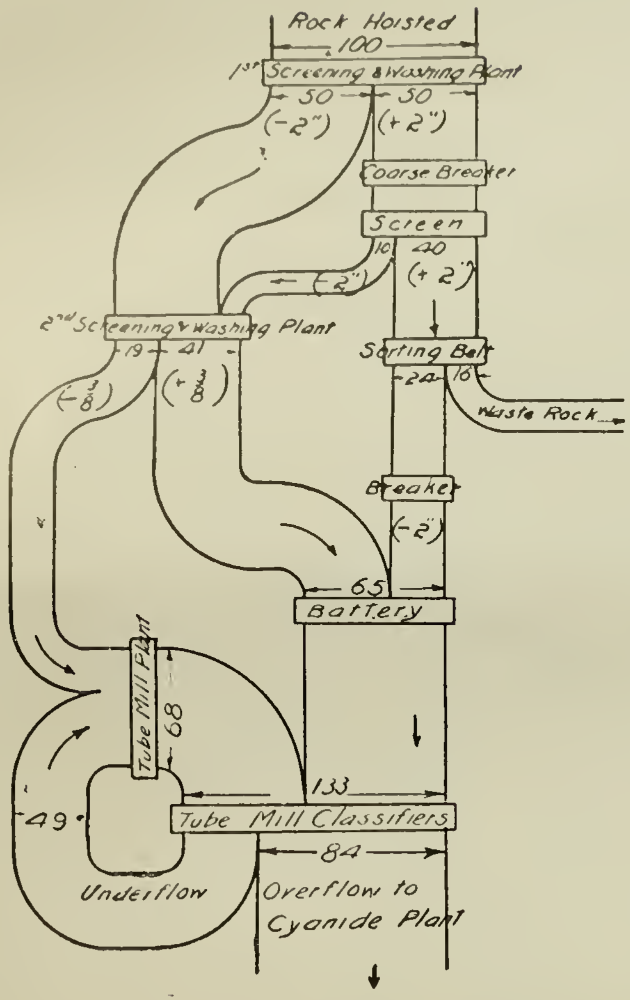

FIG. 8A.—Flow and Volume Diagram for Modified Sorting and Breaking Plant, on Basis of 16% Waste Rock, as in Fig. VA., but with Coarse Breaker.

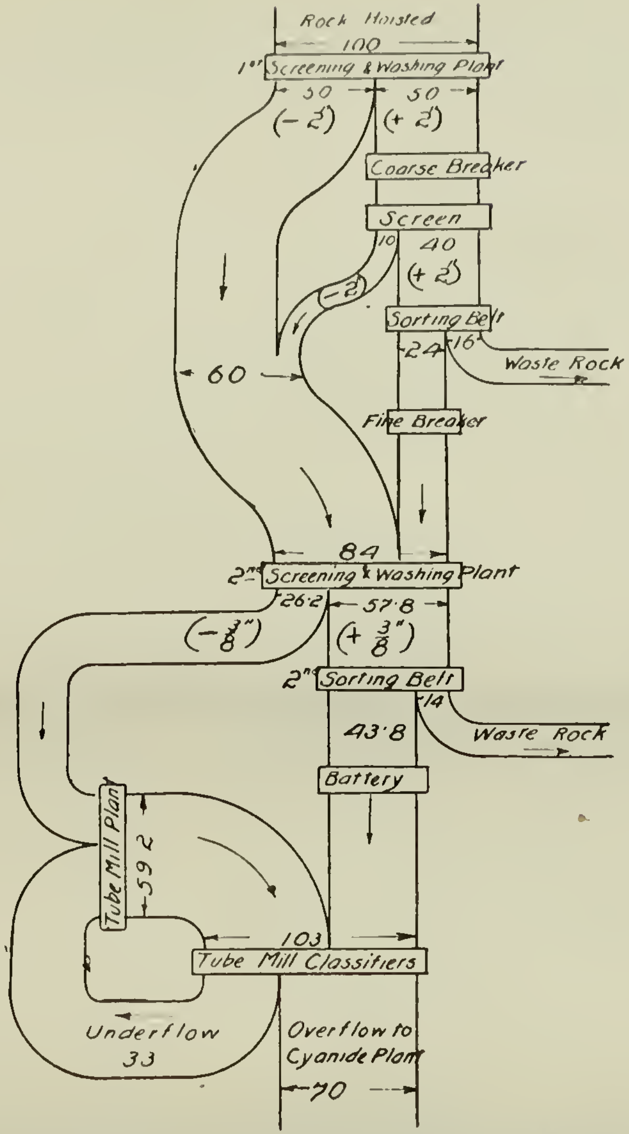

FIG. 9.—Flow and Volume Diagram for Modified Sorting and Breaking Plant, as in Fig. 6., on Basis of 30% Waste Rock, but with Coarse Breaker.

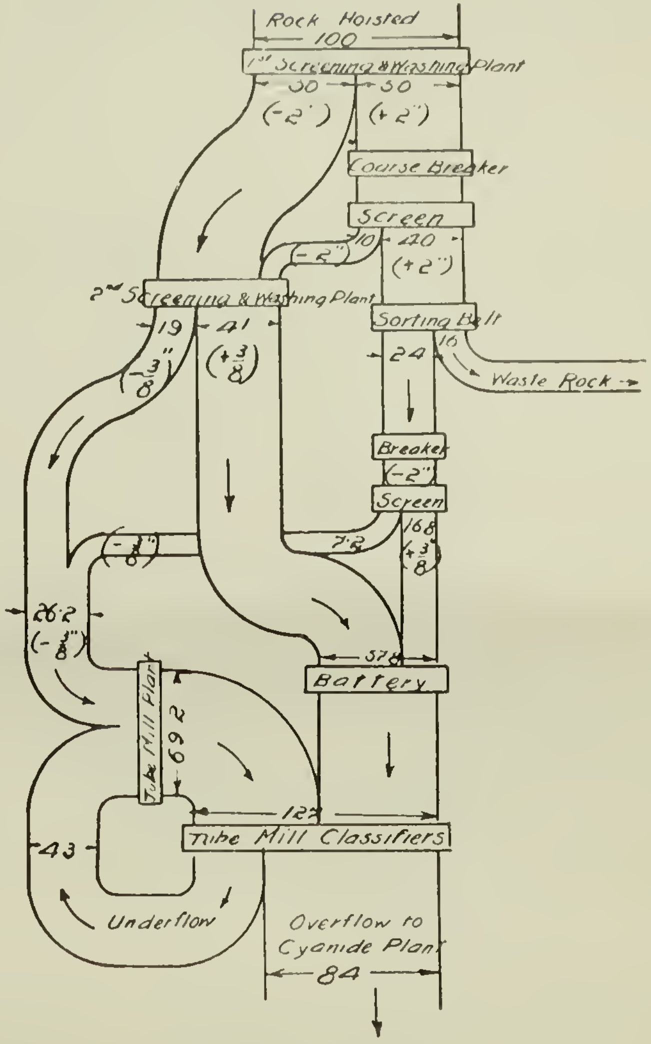

FIG. 9A.—Flow and Volume Diagram for Modified Sorting and Breaking Plant on Basis of 16% Waste Rock as in Fig. 6A., but with Coarse Breaker.

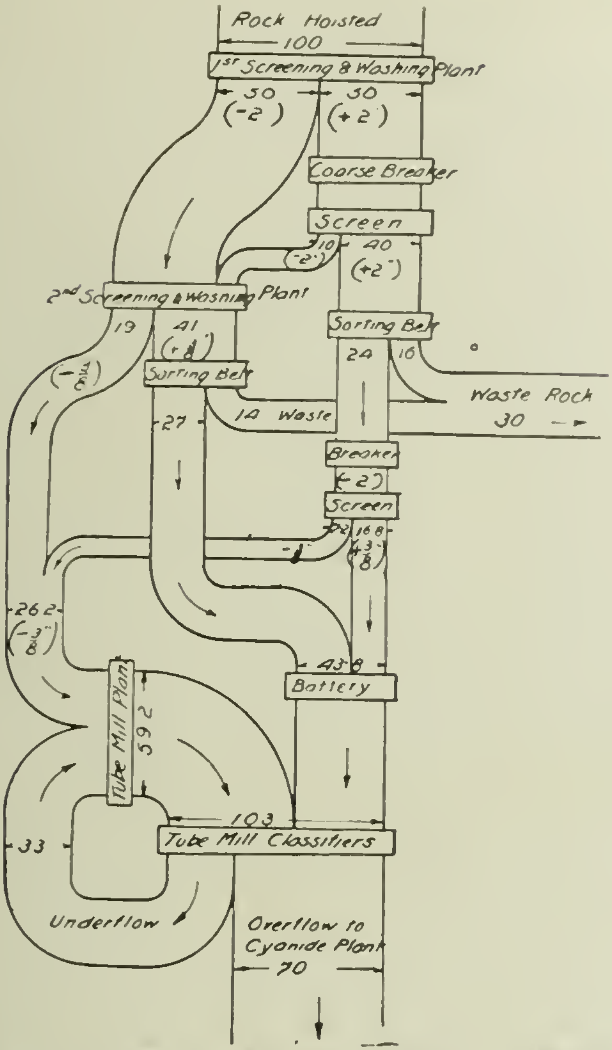

Fig. 10.—Flow and Volume Diagram for Modified Sorting and Breaking Plant as in Fig. 7, but with Coarse Breaker.

that the percentage of the ore crushed in the battery and requiring further crushing in the tube mill plant is taken at 75% which corresponds closely with actual practice in many of the later plants now at work.

For the purpose of permitting a more ready comparison between the present type of sorting plant with any of the six different forms proposed in the foregoing remarks certain data expressive of the work done have been collected in the Table III. As will be seen the basis of comparison is the quantity of rock dealt with in the various operations and expressed as a percentage of the tonnage hoisted. This obviously eliminates the usual basis for comparison—the tonnage milled as this naturally does not readily express the work done anywhere but in the stamp mill.

It appears from Table III that while the duty of the various portions of the sorting plant is increased with the exception of that of the fine breaker, the work of which is considerably reduced, the duty of the stamp battery is in the most favourable case almost halved, while that of the cyanide and tube mill plants is also reduced. With the aid of the table it will now be possible to make an estimate of the probable increase and decrease in both capital expenditure and operating costs brought about by the proposed modifications of the sorting and breaking plant.

Table III.–Comparative statement of duties of various operations with existing and proposed methods

Type of Plant.	Existing Method.	Proposed Methods on basis of 30 per cent. waste rock.					
	Fig. 2.	Fig. 5.	Fig. 8.	Fig. 6.	Fig. 9.	Fig. 7.	Fig. 10.
Number of Column.	1	2	3	4	5	6	7
1. Rock Hoisted	100	100	100	100	100	100	100
2. Waste Rock sorted out ...	16	30	30	30	30	30	30
3. Ore Milled and Treated in Cyanide Plant	84	70	70	70	70	70	70
4. Ore passed through Stamp Battery...	84	55	51	46	43·8	46	43·8
5. Ore passed through Tube Mills ...	63	56	57	58·5	59·2	58·5	59·2
6. Ore direct from Sorting Plant to Tube Mills	—	15	19	24	26·2	24	26·2
7. Rock Sorted	70	85	81	85	81	106	97·8
8. Rock Screened	100	150	210	180	234	180	234
9. Duty of Coarse Breaker ...	70	—	50	—	50	—	50
10. Duty of Fine Breaker	54	30	24	30	24	30	24
11. Ore passed through Tube Mill Classifiers	147	111	108	104·5	103	104·5	103
12. Ore passed through Tube Mill Classifiers if (—3/8) product from sorting plant is also passed through Tube Mill Classifiers		126	137	128·5	129·2	128·5	129·2

Table IIIa.

Comparative Statement of duties of various Operations with existing and proposed Methods.

Type of Plant.	Existing Methods.	Proposed Methods on basis of 16% Waste Rock.					
	Fig. 2.	Fig. 5a.	Fig. 8a.	Fig. 6a.	Fig. 9a.	Fig. 7a.	Fig. 10a.
Number of Column.	1	2	3	4	5	6	7
1 Rock hoisted	100	100	100	100	100	100	100
2 Waste Rock sorted out	16	16	16	16	16	16	16
3 Ore milled and treated in cyanide plant	84	84	84	84	84	84	84
4 Ore passed through stamp battery ...	84	69	65	59	57·8	59	57·8
5 Ore passed through tube mills ...	63	67	68	69	69·2	69	69·2
6 Ore direct from sorting plant to tube mills	—	15	19	25	26·2	25	26·2
7 Rock sorted	70	50	40	50	40	50	40
8 Rock screened	100	150	210	184	234	184	234
9 Duty of coarse breaker	70	—	50	—	50	—	50
10 Duty of fine breaker	54	34	24	34	24	34	24
11 Ore passed through tube mill classifiers	147	136	133	128	127	128	127
12 Ore passed through tube mill classifiers if ($-\frac{3}{8}$) product from sorting plant is also passed through tube mill classifiers	—	151	152	153	153·2	153	153·2

Ross E. Browne* in the paper already mentioned furnishes the following data regarding the average plant of the Rand Mines, Ltd. :—

	Tons.	
1. Rock hoisted per month ...	17,779	
2. Waste discarded per month ...	2,125	= 12%
3. Milling ore per month ...	15,654	
	s.	
4. Cost of sorting per ton of waste discarded ...	1·433	
5. Cost of conveying per ton of waste discarded ...	·334	
6. Total cost per ton of waste discarded	1·767	
7. Total cost per ton milled ...	·264	
8. Total cost per ton hoisted ...	·211	
9. Cost of breaking per ton broken	·339	
10. Cost of breaking per ton milled	·279	
11. Cost of breaking per ton hoisted	·246	
12. Cost of sorting and breaking per ton milled ...	·543	
13. Cost of sorting and breaking per ton hoisted ...	·457	

It may be gathered from these figures that sorting and breaking each represented practically half of the total cost in the type of plant then in use by the Rand Mines, Ltd. Since then there has been little change, if any, in the design of the sorting and breaking plants so that it may be assumed that at present the ratio is the same. With increased tonnages the cost per ton has, however, been considerably reduced and varies in different plants from 2·5d. to 5d. per ton milled. If we assume the comparatively high figure of 3 75d. per ton milled for the cost of sorting and breaking, the cost of sorting alone will then be 1·875d. per ton milled, which on the basis of the present percentage of waste discarded would be 2·24d. per ton hoisted (16% of waste rock) or 14d. per ton of waste removed, say 1s. 2d.

The cost of sorting also includes the cost of screening, however as the actual cost of screening has not been separately kept it will be necessary to assume a figure. Considering the very small attention paid to screening in the past the only clue we have the cost of operating a trommel, including maintenance and power, and water for washing purposes. If we assume this to be 2d. per ton sorted we have the following :—

	s.	d.
Screening per ton sorted ...	0	2
Sorting per ton sorted ...	1	0

As this is not a good basis we have to find the cost per ton screened, and as in the existing plants all the rock hoisted is screened, we have on the basis of 30% being "fines":

Cost of screening per ton screened $\frac{2\times16}{70} = \cdot5$d.

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* Working Costs of the Mines of the Witwatersrand, *Journal of the South African Association of Engineers*, Vol. XII., p. 289.

As the cost of breaking the rock is a little more than half the total sorting and breaking costs, we have 2d. as the cost of breaking per ton milled, or on the basis of 30% "fines" and 16% waste discarded:—

$$\frac{2 \cdot 0 \times 84}{54} = 3 \cdot 11\text{d. per ton broken or say 4d.}$$

which is practically the figure given by Ross E. Browne. The cost of coarse breaking is considerably less and may be taken at 2d. per ton broken. We now have the following operating costs:

	s.	d.
Sorting per ton of waste discarded	1	0
Screening per ton screened ...	0	0·5
Fine breaking per ton broken ...	0	4·0
Coarse breaking per ton broken ...	0	2·0

To complete the comparison we also need the cost of stamp milling, tube milling and cyaniding. These vary considerably in various plants, and although the figures published by E. J. Way* and used on page 466, show the average reduction costs at 5s. per ton milled, it is proposed to use the lower figures actually obtaining in a number of large plants, especially as by so doing the result obtained by the modifications suggested is adversely affected. The following will therefore be taken as a basis:

	s.	d.
Crushing in stamp batteries per ton milled	1	6
Cyaniding per ton treated .	1	9
Tube milling per ton milled ...	0	6·75
Tube milling per ton passed through tube mill ...	0	9

or a total of 3s. 9·75d. to per ton milled. It will be admitted that these figures are low, and therefore distinctly unfavourable to the point to be brought out, and it is hoped that this will be taken into account if any of the statements made should appear too favourable. With the aid of tabulated statement of duties (table III) operating costs of the modified plant as well as of the present type of plant can be calculated. The result is shewn in table IV. It appears from table IV. that there will be a saving in working costs varying from 8·16d. to 10·32d. per ton hoisted, or per unit of gold recovered, it being assumed that the mining is carefully and efficiently done in both cases.

Naturally a comparison on the basis of the tonnage milled is not possible, as owing to the reduced quantity of milling ore, due to the increased percentage of waste rock discarded, such a comparison would yield misleading results.

It now remains to be seen what the saving in capital expenditure will be under the altered conditions. However, before comparisons can be

TABLE IV.—Comparative Statement of Operating Costs in Existing Plant and Proposed New Plants.

Type of Plant	Rate per Ton.	Existing Plant Fig. 2		Proposed Plants on Basis of 30 per cent. Waste Rock.											
				Fig. 5.		Fig. 8.		Fig. 6.		Fig. 9.		Fig. 7.		Fig. 10.	
Number of Columns		1		2		3		4		5		6		7	
	Shillings	Tons	Shillings	Tons	Shillings	Tons	Shillings	Tons	Shillings	Tons	Shillings	Tons	Shillings	Tons	Shillings
1 Milling	1·5	84	126	55	82·5	51	76·5	46	69	43·8	65·75	46	69	43·8	65·75
2 Tube milling ...	·75	63	47·25	56	42	57	42·75	58 5	43·75	59·2	44·25	58·5	43·75	59·2	44·25
3 Cyaniding ...	1·75	84	147	70	122·5	70	122·5	70	122·5	70	122·5	70	122·5	70	122·5
4 Screening... ...	·0416	100	4·16	150	6·24	210	8 74	180	7·49	234	9·73	180	7 49	231	9·73
5 Sorting	1·0	70	70	85	85	81	81	85	85	81	81	106	106	97·8	97·83
6 Coarse breaking ...	·167	70	11·67	—	—	50	8·33	—	—	50	8·33	—	—	50	8·33
7 Fine breaking ...	·334	54	18	30	10	24	8	30	10	24	8	30	10	24	8
8 Total costs ...			424·08		348·24		347·77		337·74		339·56		358·74		356·36
9 Costs per ton hoisted			4·24		3·48		3·48		3·38		3·40		3·59		3·56
10 Saving per ton hoisted					·76		·76		·86		·84		·65		·68
11 Saving per ton hoisted					9·12d.		9·12d		10·32d.		10 08d.		7·8d.		8 16d.
12 Saving in per cent. ...					17·9%		17·9%		20·2%		19·8%		15·3%		16·0%

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* Trans. Inst. M. and M., Vol. xix., p. 12.

TABLE IVA.

Comparative Statement of Operating Costs in Existing Plant and Proposed New Plants.

Type of Plant.	Rate per ton.	Existing Plant.		Proposed Plants on Basis of 16 per cent. Waste Rock.							
		Fig. 2.		Fig. 5a.		Fig. 8a.		Fig. 6a.		Fig. 9a.	
Number of Columns.		1		2		3		4		5	
	Shillings	Tons	Shillings	Tons	Shillings	Tons	Shillings	Tons	Shillings	Tons	Shillings
1. Milling	1·5	84	126·00	69	103·50	65	97·5	59	88·50	57·2	85·8
2. Tube Milling ...	·75	63	47·25	67	50·25	68	51·0	69	51·75	69·2	51·9
3. Cyaniding ...	1·75	84	147·00	84	147·00	84	147·00	84	147·00	84	147·00
4. Screening ...	·0416	100	4·16	150	6·24	210	8·74	184	7·65	234	9·73
5. Sorting ...	1·0	70	70·00	50	50·00	40	40·00	50	50·00	40	40·00
6. Coarse Breaking ...	·167	70	11·67	—	—	50	8·33	—	—	50	8·33
7. Fine Breaking ...	·334	54	18·00	34	11·33	24	8·00	34	11·33	24	8·00
8. Total Costs ...			424·08		368·32		360·57		356·23		350·76
9. Costs per ton hoisted			4·24		3·68		3·61		3·56		3·51
10. Saving p. ton hoisted					·56		·63		·68		·73
11. Saving p. ton hoisted					6·72d.		7·56d.		8·16d.		8·76d.
12. Saving in per cent....					15·8%		17·8%		19·2%		20·6%

made, it will be necessary to shortly describe how far the modified method of working will affect the design of the plant.

The present type of plant may shortly be described as follows:—In most mines there are one or more storage bins underground varying considerably in size, there are also one or more storage bins in the sorting and breaking plant and finally there is a large bin attached to the stamp battery. The last mentioned has so far been of a capacity to hold a three days supply of ore. However, with the great stamp duties now being reached it will be impossible without unduly increasing the cost of the mill bin to comply with the rule. The combined capacity of the storage bins in the sorting plant may be taken as equivalent at from 12 to 24 hours supply to the battery. In view of the fact that hoisting and surface transport is not permitted on Sundays, the storage underground and in the sorting plant is of no avail for supplying the battery on that day. This has already been taken into account in some plants where storage bins in the sorting plant have been omitted, arrangements having been made for continually transferring the ore from the sorting plant to the battery. Therefore, except in the case of intermittent transport of the ore to the battery, these bins are of no use and for our purpose can be omitted. We have then only the mill bin to consider and it is proposed, in the new plant, to allow for a bin giving three full days storage capacity. Hence, the battery will remain as before, except as regards the number of stamps, and the cost per stamp will, other things being equal, remain the same.

The tube mill plant also will remain the same, except in the number of mills required, but the cost per mill will not change.

The cyanide plant will not alter in design, but the capacity will be decreased in conformity with the decreased tonnage to be handled, due to sorting out more waste rock.

The sorting plant, however, will be considerably altered from the present design. It has already been stated that in order to obtain more efficient screening the stationary grizzleys should be discarded and replaced by a more efficient device. This will also necessitate a storage bin below the tipping gear in the head frame to hold sufficient rock for equalising the flow through the first screening plant. This bin will not be very expensive, in fact some plants already have it, although it is in the wrong place, being placed after the screening device instead of before it. The chief point of difference between the old and new sorting plant will be a storage bin for the $-\frac{3}{8}$ in. product which, in future, is to be sent direct to the tube mill plant. To secure proper running and regular work this bin also should have a capacity equal to a three days supply. Perhaps it had better be erected adjoining the tube mill plant, in any case its cost must be allowed for. The sorting arrangements proper will naturally have to be augmented to deal with the increased quantity of rock handled as shewn in table III. and this also applies to the breaking plant, except that in this case it will be a decrease.

According to the writer's investigations the cost of the various parts of the reduction plant can be conveniently stated on the basis of the tonnage milled per day.

Taking a plant capable of dealing with 2,000 tons of ore per day, the cost per ton milled per day and per ton hoisted per day, on the basis of 16% waste discarded, may be taken as follows :—

	Per ton milled	Per ton hoisted.
Sorting and breaking plant	£13	£10·92
Stamp battery	25	21·00
Tube mill plant	12	10·18
Tube mill plant per ton passed through tube mills	16	
Cyanide plant	59·1	49·644
Power distribution service	5·4	4·536
Total cost per ton hoisted		£96·280

The milling or crushing plant would consist of 240 stamps of 1,500 lbs. falling weight or 200 stamps of 1,500 lbs. falling weight, augmented by 8 tube mills 22 ft. x 5 ft. 6 in. The pulp elevating plant which will also be favourably influenced by the proposed modifications has not been included as the difference is so small as not to affect the total seriously.

Again using the tabulated statement (page 475) the capital expenditure for the modified as well as the existing plant, not considering the sorting plant, has been calculated and is as shown in Table V.

In the table the figures for power distribution have been obtained as follows :—The power consumption of the reduction plant is approximately at the rate of ·8 K.W. per ton milled per day, or a total of 1,600 K.W. for the 2,000 ton plant. The plant consists of :—

200 stamps @ 4 h.p. = 800 h.p.
= 600 K.W. = 37·5%
8 tube mills @ 100 h.p. = 800 h.p.
= 600 K.W. = 37·5%
Cyanide plant, 400 K.W. = 25%

As the total cost of the power distribution service is £5 4s. per 84 tons milled, the total cost of power plant is therefore 84 × 5·4 = £453 6s. Apportioning this in the proportion of 37·5%, 37 5%, and 25%, we have :—

Mill... ... £170 or £2·0 per ton on 84 tons
Tube Mill ... £170 ,, £2·7 ,, ,, ,, 63 ,,
Cyanide Plant £113·6 ,, £1·35 ,, ,, ,, 84 ,,

which are the rates used in lines 5, 6, and 7 of the table. The table shows that as far as stamp mill, tube mill plant, cyanide plant, and power distribution service are concerned a saving of from 20·6% to 23·4%, as compared with the present type of plant, can be obtained by the modifications already described.

Table V.—Comparative Statement of Capital Expenditure on Part of Reduction Plant with Present and Proposed Methods of Working.

Type of Plant	Rate per Ton per Day.	Existing Plant.		Proposed Plants on Basis of 30 per cent. Waste Rock.											
		Fig. 2.		Fig. 5.		Fig. 8.		Fig. 6.		Fig. 9.		Fig. 7.		Fig. 10.	
Number of Column		1		2		3		4		5		6		7	
	£	Tons	£	Tons	£	Tons	£	Tons	£	Tons	£	Tons	£	Tons	£
1 Stamp mill	25	84	2100	55	1375	51	1275	46	1150	43·8	1095	46	1150	43·8	1095
2 Tube mill plant ...	16	63	1008	56	896	57	912	58·5	936	59·2	917 2	58·5	936	59·2	917·2
3 Cyanide plant ...	59·1	84	1964·4	70	4137	70	4137	70	4137	70	4137	70	4137	70	4137
4 Total (1 + 2 + 3) ...			8072·4		6408		6324		6223		6179·2		6223		6179·2
5 Power distribution service: Stamp mill	2·0	84	168·0	55	110	51	102	46	92	43·8	87·6	46	92	43·8	87·6
6 Tube mills...	2·7	63	170·1	56	151·2	57	153 9	58·5	157·7	59·2	159·8	58·5	157·7	59·2	159·8
7 Cyanide plant	1·35	84	113·4	70	94·5	70	94·5	70	94·5	70	94·5	70	94·5	70	94·5
8 Total (4 + 5 + 6 + 7) ...			8523·9		6763·7		6674·4		6567·2		6521·1		6567·2		6521·1
9 Saving due to modified plant					1760·2		1849·5		1956·7		2002·7		1956·7		2002·8
10 Saving per ton hoisted per day					17·602		18·495		19·567		20·028		19·567		20·028
11 Saving in per cent. ...					20·6%		21·6%		22·8%		23·4%		22·8%		23·4%

TABLE VA.—Comparative Statement of Capital Expenditure on Part of Reduction Plant with Present and Proposed Methods of Working.

Type of Plant.	Rate per ton per day.	Existing Plant.		Proposed Plants on basis of 16 per cent. Waste Rock.							
		Fig. 2.		Fig. 5A.		Fig. 8A.		Fig. 6A.		Fig. 9A.	
Number of Columns.		1		2		3		4		5	
	£	Tons	£	Tons	£	Tons	£	Tons	£	Tons	£
1 Stamp mill	25	84	2100	69	1725·0	65	1625·0	59	1475·0	57·2	1430·0
2 Tube mill plant ...	16	63	1008	67	1072·0	68	1088·0	69	1104·0	69·2	1107·2
3 Cyanide plant ...	59·1	84	4964·4	84	4964·4	84	4964·4	84	4964·4	84	4964·4
4 Total (1 + 2 + 3) ..			8072·4		7761·4		7677·4		7543·4		7501·6
5 Power dis- Stamp mill	2·0	84	168·0	69	138·0	65	130·0	59	118·0	57·2	114·4
6 tribution Tube mills	2·7	63	170·1	67	180·9	68	183·6	69	186·3	69·2	186·8
7 service Cyanide plant	1·35	84	113·4	84	113·4	84	113·4	84	113·4	84	113·4
8 Total (4 + 5 + 6 + 7) ...			8523·9		8193·7		8104·4		7961·1		7916·2
9 Saving due to modified plant					330·2		419·5		562·8		607·7
10 Saving per ton hoisted per day					3·30		4·2		5·63		6·08
11 Saving in per cent. ...					3·86%		4·9%		6·6%		7·1%

With regard to the sorting and breaking plant, the chief item of additional expenditure will be the cost of the storage bin for the $-\frac{3}{8}$ in. product plus the cost of a means of transport for conveying the same to the tube mill plant. For the remaining portion of the sorting plant the expected increase of cost will have to be determined in accordance with the duty, as shown in the tabulated statement on p. 475. Dealing with the new storage bin first, it is proposed to allow for a three-days' supply, as suggested on p. 478, as being in accordance with the present practice in relation to the ore bin in the stamp battery, and which is ample to tide over the period of forced idleness on Sunday. As the greatest tonnage sent from the sorting plant direct to the tube mill does not exceed 60% of the tonnage hoisted, the storage required for the 100-ton plant would be about 180 tons. Presuming that the cheapest form of storage bin is a circular steel vat set on the ground, and as the product in question can be conveniently and cheaply conveyed by pumping, it is proposed to allow for a vat with a conical bottom, having the discharge hole at the apex of the cone. This discharge hole will be connected to the suction branch of a centrifugal pump, which will deliver a steady stream to the launder entering the tube mill plant. If the $-\frac{3}{8}$ in. product is conveyed to the storage vat by gravity with the aid of the washing water, the water may be promptly returned for re-use by providing the vat with a peripheral overflow launder, as now used on the slime collecting vats.

For a plant dealing with 2,000 tons of rock hoisted per day, to take a convenient size, the capacity of the storage vat would require to be 3,600 tons, or, at 20 cub. ft. per ton, 72,000 cub. ft. This would require two vats of 50 ft. diameter by 20 ft. deep, which, together with the necessary pumping plant for ore and return water, would not cost more than £3,000, or £1 10s. per ton hoisted per day.

In order to arrive at a reliable figure regarding the cost of the various parts of the sorting plant under the altered conditions, the amount (given on p. 479) of £13 per ton milled per day, or £10·92 (say £11) per ton hoisted per day, must be sub-divided. As these figures do not allow for coarse breaking, an amount of £1 per ton hoisted per day is provided for this purpose. On the basis of the actual cost of a number of sorting and breaking plants now in successful use, the writer arrived at the figures shown on Table VI., which are also rectified to give the cost per ton of rock or ore actually dealt with in various portions of the plant. With the aid of Table III., the capital cost for the proposed new plants, as compared with the present type, can now be calculated. The result is given in Table VII. It appears from the table that the proposed modifications of the sorting and breaking plant cause an increase in the capital cost varying from 11·2% to 39%.

Now, in order to arrive at the nett saving in capital expenditure, if any, for the whole plant, it will be necessary to combine the totals of Tables V. and VII. as shown in Table VIII.

The saving on capital account due to the modifications in the total plant, *i.e.*, sorting and breaking plant, stamp mill, tube mill plant, cyanide plant, and power distribution service,

TABLE VI.

Statement of Capital Cost per ton of Rock hoisted or dealt with in various portions of Sorting and Breaking Plant.

TYPE OF PLANT.	Cost per ton Hoisted.	Tons of Ore dealt with.	Cost per ton of Ore actually dealt with.
1. Screening Plant	£ ·9	100	£ ·9 per ton screened.
2. Coarse Breaking Plant	£1·0	70	£1·425 per ton broken.
3. Washing Plant	£1·1	70	£1·57 per ton washed.
4. Sorting Plant	£2·0	70	£2·85 per ton sorted.
5. Fine Breaking Plant	£2·1	54	£3·9 per ton broken.
6. Waste Rock Disposal...	£1·0	16	£6·25 per ton of waste.
7. Building and Foundations	£2·0	70	£2·85 per ton sorted.
8. Storage Bins	£1·6	84	£1·9 per ton milled.
9. Repair Shop, Lifting Tackle, etc. ...	£ ·3	100	£ ·3 per ton hoisted.
Total	£12·0		

TABLE VII.

Comparative Statement of Capital Expenditure on Sorting and Breaking Plants, with present and proposed methods of working.

Type of Plant.	Rate per ton.	Existing Plant.		Proposed Plant on basis of 30 per cent. Waste Rock.											
		Fig. 2.		Fig. 5.		Fig. 8.		Fig. 6.		Fig. 9.		Fig. 7.		Fig.	
Number of Column.		1		2		3		4		5		6		7	
	£	Tons	£	Tons	£	Tons	£	Tons	£	Tons	£	Tons	£	Tons	
1 Screening ..	·9	100	90	150	135	210	189	180	162	234	210·6	180	162	234	
2 Coarse breaking	1·425	70	100	—	—	50	71·25	—	—	50	71·25	—	—	50	
3 Washing ..	1·57	70	110	150	235·5	210	329·7	180	282·6	234	367·38	180	282·6	234	
4 Sorting ...	2·85	70	200	85	242·35	81	230·8	85	242·25	81	230·8	106	302·1	97·8	
5 Fine breaking...	3·9	54	210	30	117	24	93·6	30	117	24	93·6	30	117	24	
6 Waste disposal	6·25	16	100	30	187·5	30	187·5	30	187·5	30	187·5	30	187·5	30	
7 Building ..	2·85	70	200	85	242·25	81	230·8	85	242·25	81	230·8	106	302·1	97·8	
8 Storage bins ...	1·9	84	160	—	—	—		—	—	—	—	—	—	—	
9 Repair shop	·3	100	30	100	30	100	30	100	30	100	30	100	30	100	
10 Storage for $\frac{3}{8}''$	1·5	—	—	100	150	100	150	100	150	100	150	100	150	100	
11 Total ...			1200		1339·6		1512·65		1413·6		1571·93		1533·3		
12 Increase due to modified plant					133·95		312·65		241·36		371·93		333·3		
13 Increase per ton hoisted per day					1·34		3·13		2·41		3·72		3·33		
14 Increase in per cent. ...					11·2%		26%		20·1%		31%		27·75%		

TABLE VIIA.—Comparative Statement of Capital Expenditure on Sorting and Breaking Plants, with Present and Proposed Methods of Working.

Type of Plant.	Rate per ton.	Existing Plant.		Proposed Plants on basis of 16 per cent. Waste Rock.							
		Fig. 2.		Fig. 5a.		Fig. 8a.		Fig. 6a.		Fig. 9a.	
Number of Column.		1.		2		3		4		5	
	£	Tons	£	Tons	£	Tons	£	Tons	£	Tons	£
1. Screening ...	·9	100	90	150	135·00	210	189·00	184	165·60	234	210·60
2. Coarse Breaking ...	1·425	70	100	—	—	50	71·25	—	—	50	71·25
3. Washing	1·57	70	110	150	235·50	210	329·70	184	288·88	234	367·38
4. Sorting	2·85	70	200	50	142·50	40	114·00	50	142·50	40	114·00
5. Fine Breaking	3·9	54	210	34	132·60	24	93·60	34	132·60	24	93·60
6. Waste Disposal ...	6·25	16	100	16	100·00	16	100	16	100	16	100
7. Building ...	2·85	70	200	50	142·50	40	114·00	50	142·50	40	114·00
8. Storage Bins ...	1·9	84	160	—	—	—	—	—	—	—	—
9. Repair Shop ...	·3	100	30	100	30·00	100	30·00	100	30·00	100	30·00
10. Storage for – ⅜ in. ...	1·5	—	—	100	150·00	100	150·00	100	150·00	100	150 00
11. Total			1200		1068·10		1191·55		1152·08		1250·83
12. Increase or decrease					– 131·9		– 08·45		– 47·92		+ 50·83
13. Increase or decrease per ton hoisted ...					—1·32		—·085		—·48		+ ·51

TABLE VIII.

Comparative Statement of Capital Expenditure on Part of Reduction Plant and Sorting and Breaking Plant with Present and Proposed Methods of Working, being Total of Tables V. and VII.

Type of Plant	Existing Plant.	Proposed Plants on Basis of 30 per cent. Waste Rock.					
	Fig. 2	Fig. 5	Fig. 8	Fig. 6	Fig. 9	Fig. 7	Fig. 10
Number of Column.	1	2	3	4	5	6	7
	£	£	£	£	£	£	£
1 Cost of stamp mill, tube mill and cyanide plant and power supply	8523·9	6763·7	6674·4	6567·2	6521·1	6567·2	6521·1
2 Cost of sorting and breaking plant	1200·0	1339·6	1512·65	1413·6	1571·93	1533·3	1667·79
3 Total cost (1 + 2)	9723·9	8103·3	8187·05	7980·8	8093·03	8100·5	8188·89
4 Saving due to modified plant ...	—	1620·6	1536·85	1743·1	1630·87	1623·4	1535·01
5 Saving per ton hoisted per day ...	—	16·2	15·37	17·43	16·3	16·23	15·35
6 Saving in per cent.	—	16·6%	15·8%	17·9%	16·8%	16·7%	15·8%

TABLE VIIIA.—Comparative Statement of Capital Expenditure on Part of Reduction Plant and Sorting and Breaking Plant with Present and Proposed Methods of Working being Total of Tables VA. and VIIA.

Type of Plant.	Existing Plant.	Proposed Plants on basis of 16 per cent. Waste Rock.			
	Fig. 2.	Fig. 5A.	Fig. 8A.	Fig. 6A.	Fig. 9A.
Number of Column.	1	2	3	4	5
	£	£	£	£	£
1 Cost of stamp mill, tube mill and cyanide plant and power supply. VA. ...	8523·9	8193·7	8104·4	7961·1	7916·2
2 Cost of sorting and breaking plant. VIIA.	1200·0	1068·1	1191·55	1152·08	1250·83
3 Total cost (1 + 2)	9723·90	9261·8	9295·95	9113·18	9167·03
4 Saving due to modified plant		462·1	427·95	610 72	556·87
5 Saving per ton hoisted per day		4·62	4·28	6·11	5·57
6 Saving in per cent.		4·75%	4·5%	6·25	5·7%

therefore varies from 15·8% to 17·9%, thus showing that notwithstanding an increase in the cost of the sorting plant, amounting to nearly 40%, a very substantial saving results in the cost of the whole plant.

Assuming that the remainder of the reduction plant, viz., amalgamating plant, mill clean-up plant, pulp elevating plant, return water service, ore transport on surface, water supply, offices, workshops, and stores cost £22 per ton milled per day, or £18 10s. per ton hoisted per day, the total cost will then be as follows, taking only the best and worst cases:—(see Table IX.)

The saving in capital expenditure, as evidenced by Table VIII., will reduce capital charges to the same extent. Taking 12% per annum, made up as follows:—

Interest at	6%
Depreciation at	4%
Redemption at	2%

as the basis, the saving in capital charges due to the modified plant will be shown in Table X. Add to this the saving in operating costs, as per Table IV., the total saving will be as per Table XI.

TABLE IX.

Column (referring to previous tables).	Existing Plant. 1.	Proposed Plant. 4.	7.
	Per ton	hoisted per	day.
1. Cost of Sorting Plant, Stamp Battery, Tube Mill Plant, Cyanide Plant, Power Distribution ...	£ 97·239	£ 79·808	£ 81·8889
2. Cost of remainder ...	18·5	18·5	18·5
3. Cost of entire reduction plant	115·739	98·308	100·389
4. Saving as compared with existing plant ...		17·431	15·350
5. Saving in per cent.		15·1%	13·25%

TABLE X.

Statement of Saving in Capital Charges by adopting the proposed Methods of Working.

	Proposed Plants on basis of 20 per cent. waste rock.					
	Fig. 5.	Fig. 8.	Fig. 6.	Fig. 9.	Fig. 7.	Fig. 10.
Number of Column	2	3	4	5	6	7
Saving in Capital (Table VIII.)	£1,620	1,537	1,743	1,631	1,623	1,535
Saving in Capital Charges, per month ...	£16·2	15 37	17 43	16 31	16·23	15·35
Saving per ton hoisted (28·5 days per month)	1·37d	1·29d	1·47d	1·37d	1·36d.	1·29d.

Table Xa.—Statement of Saving in Capital Charges by Adopting the Proposed Methods of Working.

	Proposed Plants on Basis of 16 per cent. Waste Rock.			
	Fig. 5A	Fig. 8A	Fig. 6A	Fig. 9A
Number of column ...	2	3	4	5
Saving in capital (Table VIIIA.	£462·1	427·9	610·7	556·8
Saving in capital charges per month ...	£4·62	4·28	6·11	5·57
Saving per ton hoisted (28·5 days per month) d.	·39d.	·36d.	·514d	·47d.

Table XI.—Statement of total saving in Working Costs (Operating Costs and Capital Charges) by adopting the proposed methods.

	Proposed Plants on basis of 30 per cent. waste rock.					
	Fig. 5.	Fig. 8.	Fig. 6.	Fig. 9.	Fig. 7.	Fig. 10.
Number of Column	2	3	4	5	6	7
Saving in Operating Costs (Table IV.) per ton hoisted. Pence...	9·12	9·12	10·32	10·08	7·80	8·16
Saving in capital charges (Table X.) per ton hoisted	1·37	1·29	1·47	1·37	1·36	1·29
Total Saving. Pence	10·49	10·41	11·79	11·45	9·16	9·45

Table XIa. — Statement of Total Saving in Working Costs (Operating Costs and Capital Charges) by Adopting the Proposed Methods.

	Proposed Plants on Basis of 16 per cent. Waste Rock.			
	Fig. 5A.	Fig. 8A.	Fig. 6A.	Fig. 9A.
Number of column ...	2	3	4	5
Saving in operating costs (Table IVA.) per ton hoisted, pence ...	6·72	7·56	8·16	8 76
Saving in capital charges (Table XA.) per ton hoisted, pence ...	·39	·36	·51	·47
Total saving, pence ...	7·11	7·92	8·67	9·23

Thus the total saving per ton hoisted varies from 9·45d. to 11·79d., as compared with present-day practice.

As the saving in working costs and capital charges shown in Table XI. is referred to the tonnage hoisted, it will be necessary to rectify the figures so as to express the saving per ton milled on the old basis of sorting out 16% of waste rock. Taking the average saving from Table XI., viz., 10·46d. per ton hoisted, the saving per ton milled will be 12·6d., or say 1s.

Reconstructing Table II. and diagram, Fig. 1, on the basis of the reduced working costs, we obtain Table XII. and diagram, Fig. 11.

The gain, while not very great if referred to mines treating high-grade ore, which is supposed to exist at shallow depths, is yet considerable for low-grade mines, or those mining at greater depths than 3,000 ft. The advantage gained by working in the manner proposed by the writer, increases the profit derivable from the 22s. 6d. ore by 21%, and that obtainable from 20s ore by 57%.

Table XII.

1. Average Depth of Mine.	1,000 feet.	2,000. feet.	3,000 feet.	4,000 feet.	5,000 feet.
	s.	s.	s.	s.	s.
2. Mining Costs.	8·0	8·5	9·0	9·5	10·0
3. Reduction Costs.	4·0	4·0	4·0	4·0	4·0
4. General Exp's.	1·25	1·25	1·25	1·25	1·25
5. Development Redemption	2·0	2·0	2·0	2·0	2·0
6. Total Costs.	15·25	15·75	16·25	16·75	17·25
7. Yield.	40·0	30·0	25·0	22·5	20·0
8. Profit.	24·75	14·25	8·75	5·75	2·75
9. Profit Tax.	2·475	1·425	·875	·575	·275
10. Company's Profit.	22·275	12·825	7·875	5·175	2·475

Now that the monthly tonnage milled is reaching nearly 1,800,000 tons, an extra gross profit of 1s. would represent a monthly gain of £90,000, of which the profit tax accounts for £9,000, leaving £81,000 to shareholders. Of course, existing mines with reduction plants in full swing could not benefit to the same extent as mines which have not yet reached the milling stage. Yet, in cases where an increase in the capacity is planned, considerable benefit would also fall to those mines adopting the proposed modifications. Even assuming, which the writer does not admit, that the reduction in working costs as calculated in these notes will not materalise, it is hoped that a saving of 15% in

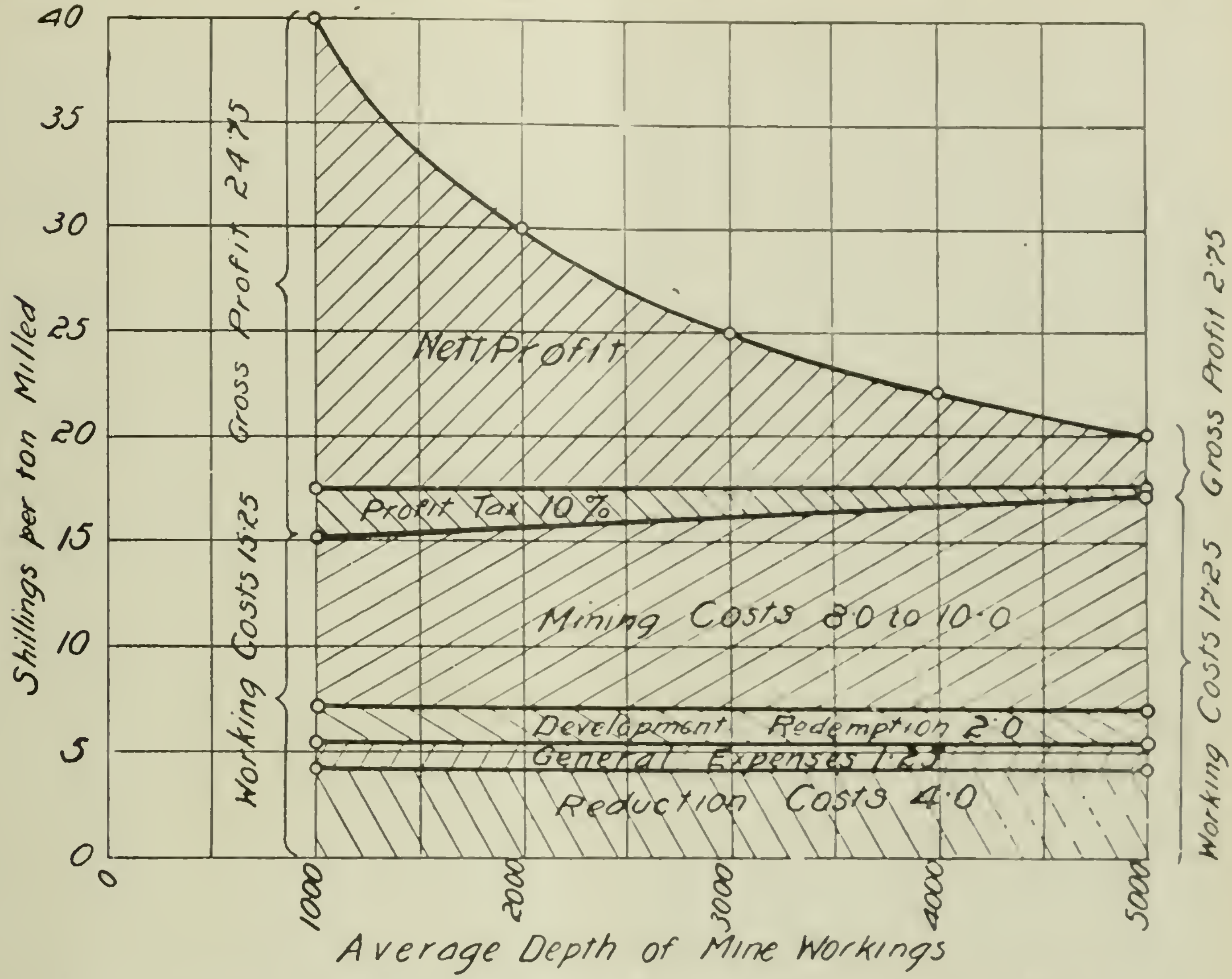

FIG. 11.—Diagram showing Screen Value of Ore, Working Costs and Profit per Ton Milled under Present Day Conditions.

the capital cost of the reduction plant is sufficiently attractive to deserve some attention, especially now when the flow of capital to the Rand is by no means as free as it used to be. In any case, it is hoped that renewed discussion, and possibly a practical demonstration of the vexed question of sorting "fines," especially the (2 in. + $\frac{3}{8}$ in.) product mentioned by the writer, will lead to some tangible result, beneficial to the mining industry and the country as a whole.

After writing the foregoing, it occurred to the author that members might object to some of the assumptions made, particularly with regard to the proposed percentage of waste rock which has to be removed in order to obtain the calculated results. Although it is thought that this percentage (30) has been justified in the reasoning already made, yet, in order to meet these objections, we will now assume that only the usual 16% of waste rock can be sorted out, and on this basis will investigate what the probable benefits may be expected to be.

From the diagrams, Figs. 5a, 6a, 8a and 9a and table III.a, which have been reconstructed on the basis of 16% of waste rock, can be seen how and where the proposed plant would be adversely affected if compared with the diagrams Figs. 5 to 10 and table III. New diagrams to correspond with Figs. 7 and 10 have not been made for reasons of economy, as it was found that these would be identical in the result with 6a and 9a.

It appears from the new diagrams and table III.a that the work to be done by the stamp battery, while considerably less than in present day plants, is more than would be the case if 30% waste were removed. Still, there is a reduction in the proportion of 81 to 57·2. The work to be done by the tube mills increases in the proportion of 63 to 69·2, and that to be done in the cyanide plant will be the same as in the present day plant. The work of the screening plant is found to be practically the same as before, and the same applies to the coarse and fine breakers. However, owing to the better separation in the first screening plant, the proportion of rock handled by the sorting belt or table decreases from 70 to 50 or 40. More work is also thrown on the tube mill classifiers.

Remodelling tables IV., V., VII., VIII. X. and XI., we obtain the tables IV.a to XI.a, and the final result (table XI.a) shows a saving in operating costs as compared with present day practice of from 7·11d. to 9·23d. per ton hoisted. This, compares with a saving varying from 9·45d. to 11·79d. previously found. It is interesting to note that the plant represented by Fig. 9a actually shows a saving greater than that obtained with the plant shown in Fig. 7, and closely approaches the plant Fig. 10.

The average saving from table XI.a, taking also into account the plants corresponding to Figs. 7 and 10, but not reproduced, is 8·47d. per ton hoisted, and on the basis of 16% sorted out, 10d. per ton milled thus decreasing reduction costs by that amount. Allowing for this in table XII. we obtain table XII.a as under :—

TABLE XII.a.

1. Average Depth.	1,000 feet.	2,000 feet.	3,000 feet.	4,000 feet.	5,000 feet.
	s.	s.	s.	s.	s.
2. Mining Costs	8·0	8·5	9·0	9·5	10·0
3. Reduction Costs ...	4·2	4·2	4·2	4·2	4·2
4. General Expenses ...	1.25	1·55	1·25	1·25	1·25
5. Development Redemption.	2·0	2·0	2·0	2·0	2·0
6. Total Costs...	15·45	15·95	16·45	16·95	17·45
7. Yield ...	40·0	30.0	25·0	22·5	20·0
8. Profit ...	24·55	14·05	8·55	5·55	2·55
9. Profit Tax ...	2·45	1·40	·85	·55	·25
10. Company's Profit ...	22·10	12·65	7·70	5·00	2·30

It will be admitted that even on the basis of present day practice regarding the percentage of waste rock sorted out, a considerable saving in working costs can be obtained by modifying the sorting and breaking plant on the lines proposed in these notes.

II.—*The Milling or Crushing Plant.*—From the foregoing remarks in respect of possible economies resulting from a modification of the sorting and screening plant, it appears that the greater portion of the saving effected is due to a redistribution of the work in the milling or crushing plant. As the suggested modifications of the sorting plant may or may not be adopted in practice, it remains to be seen how and where economies can be effected in the milling or crushing plant if regarded as an independent unit of the reduction plant.

Reviewing the work of the immediate past, even the casual observer cannot help being impressed by the change in the design of the milling or crushing plant At the end of 1904, about 800,000 tons were crushed per month by 6,500 stamps, while at the end of 1910 about 1,750,000 tons were crushed per month by 9,500 stamps, assisted by 200 tube-mills, which are equivalent to nearly 6,000 stamps. The average falling weight per stamp in 1904 was probably under 1,200 lbs., while in 1910 it must have been nearly 1,400 lbs., or on the basis of the nominal crushing unit introduced by Dr. Caldecott, there were in use :—

1904— 6,250 nominal crushing units.

1910—10,600 nominal crushing units in stamps ...
6,000 nominal crushing units in Tube Mills ...
} Or a total of 16,600 nominal crushing units.

The duty per crushing unit in 1904 was 127·5 tons per month, and in 1910 105 tons per month. The apparent decrease in the duty is explained by the finer crushing. The grading of the average final pulp in 1904 showed probably :—

20% of ... ... +60.
20% of ... ... −60+90.
60% of ... ... −90.

thus producing a total of 480,000 tons of −90, and the average pulp in 1910 showed probably the following grading :—

10% of ... ... +60.
15% of ... ... −60+90.
75% of ... ... −90.

an advance of 15% in the percentage of −90. product, or a total of 1,312,500 tons. Further, by referring to Table I. and Fig. 1, giving the probable grade for various depths of mining, it will be seen that while in 1904 the average depth of mining was about 1,600 ft., corresponding to a grade of 34·7s., in 1910 the average depth was 2,400 ft., corresponding with a grade of 28s. The rock from the greater depth is naturally harder, requiring a greater effort to reduce it to the required fineness. Ignoring this fact, however, it is found that the tonnage of −90 per nominal crushing unit per month in 1904 was :

77 tons, or 2·7 tons per day,

and in 1910 :

80 tons, or 2·8 tons per day,

on the basis of 28·5 working days per month. The increase in duty, ignoring the greater hardness of the rock is, therefore, not very marked, and this is undoubtedly due to the fact that many plants on the Rand are even to-day run in a very indifferent manner. This is obvious, if it is considered that in many plants the −90 produced per nominal crushing unit is 3·5 tons, or 25% more than the calculated average for the whole Rand, and is further confirmed by the results* of the crushing trials at the New Kleinfontein, where in one case (1,680 lb. stamps) 3·27 tons, and in the other (1,400 lb. stamps) 3·26 tons of −90 per nominal crushing unit were produced.

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* Trans. Inst. M. & M., Bulletin No. 75, p. 42. This *Journal*, Vol. XI., p. 248.

Although from the crushing point of view the improvement is not very great, the net result is nevertheless considerable because, not only have operating costs being reduced, but the extra extraction due to finer crushing is equal to at least 3% or 10d. per ton milled on 28s. ore. In 1904 the crushing plant consisted of stamps only the cost of which per nominal crushing unit installed would to day be £240, including mill bin, as against £180 per nominal crushing unit of the double stage crushing plants in use in 1910, made up as follows :

106 nominal crushing units in stamps at £240 = ...	£25,440
60 nominal crushing units in tube mills at £80	4,800
Total for 166 N.C.U. ...	£30,240

or £180 per nominal crushing unit. This represents a saving of 25%.

While the foregoing figures do not lay claim to accuracy, which is obviously impossible to attain, they do shew that better work is now done with the aid of the tube mill than was the case in 1904 besides being interesting from a historical point of view.

Possible further economies in the cost of the milling and crushing plant have been alluded to by several writers, notably Dr. Caldecott,* while the beneficial effects on the capital expenditure resulting from using a proportionately greater tube mill equipment, than is now the case, have been pointed out by the writer in 1909.† Since then a considerable advance in the right direction has been made, and it is hoped that, in the light of the following notes, some of the critics of the two stage crushing plant will repent.

While the fight between the advocates of single and double stage crushing has been waxing fast and furious, abuse masking the want of data and argument on the part of the single stage men, E. H. Johnson made some crushing trials at the East Rand Proprietary Mines the result of which must certainly be considered as sufficient to eliminate all doubt from the minds of unbiased observers. Mr. Johnson and his corporation deserve the greatest credit for their enterprise in doing such signal service for the benefit of the whole mining industry. The result of these trials, as published‡, was as follows :—

10 stamps of 1,634 lbs. falling weight, and one tube mill 22 ft. x 5 ft. 6 in., with preliminary screening into − ⅜in. and + ⅜in. prior to entering the mortar box, crushed 261·34 tons per 24 hours to the following grading :—

+ 90	−	·5%
− 90 + 200	−	15·5
− 200	−	84·0%

that is 99·5% or 260 tons −90 product or 6 tons of −90 per nominal crushing unit, the total equipment being made up as follows :—

10 stamps = 13 nominal crushing units ... ...
1 tube mill = 30 nominal crushing units ...
} a total of 43 N.C.U.

The power consumption was 150 h.p. (approximately) so that the production of one ton of −90 required $\frac{150 \times 24}{260}$ = 13·8 h.p.-hours. Such a result has, of course, never before been reached on similar ore, but whether this is the ultimate capacity of the double stage crushing plant or whether better results may yet be expected must remain a matter for further investigation. One thing, however, is certain namely that a very considerable reduction in capital expenditure on future crushing plants coupled with a substantial reduction in operating costs can now be depended upon. The possible saving can best be realised by comparing the work done in the foregoing crushing test with similar tests made, by using stamps only, at the Kleinfontein G. M. Co.* This is done in Table XIII. with regard to power charges.

It will be seen that on the basis of the −90 product the power consumption of the double stage crushing plant is not much more than half of that required in the single stage crushing plant, while if compared on the basis of tonnage milled, the cost of power in the double stage plant is only 75% of that for the single stage plant or a saving of 4·32d. per ton of −90 and 1·87d. per ton of ore milled.

Comparing the capital expenditure of the two types of plant on the basis of these experiments the following results :—

Double Stage Crushing Plant : E.R.P.M. experiment :

1.	10 stamps at £260 .. .	£2,600
2.	1 tube mill at £2,400 ...	2,400
3.	Portion of pulp elevating plant (£1 per ton)	260
4.	Extra for trebling mill ore bin ..	600
5.	Total	£5,860

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* "The introduction of the heavy gravitation stamp," Trans. Inst. M. & M., Vol. XIX., p. 57.

† *Journal of South African Association of Engineers*, Vol. XV., p. 113.

‡ This *Journal*, Vol. XI., p. 168.

* Trans. Inst. M. and M., [illegible] p. 42; also this *Journal*, vol. XI., p. 218.

TABLE XIII.

Test.	Ore milled per day.	– 90 product.		Power consumption. h.p.	– 90 per h.p.	Percentage.	H.P. hours. per ton of – 90.	Cost per ton of – 90 at ·4d. per h.p. hr.	Cost per ton milled.
		%.	Tons.						
East Rand Prop. Mines ...	261·34	99·5	260	150	1·734	100	13·84	5·535d.	5·5d·
New Kleinfontein I.	200·0	74·8	149·6	153	·98	56·5	24·64	9·856d	7·37d.
New Kleinfontein II.	1320	60·7	801·54	814	·98	56·5	24·64	9·856d.	5·98d.

6.	Cost per ton milled (261 tons)	£22·45
7.	Cost per ton of – 90 product (99·5%)	£22·54
8.	Capital charges at 12% per annum, per ton milled amount to ..	1·89d.

Single Stage Crushing Plant ; New Kleinfontein experiment No. 1 :

1.	44 stamps at £260	£11,440
2.	Cost per ton milled (261 tons) ..	£43·8
3.	Cost per ton of – 90 product (74·8%)	58·6
4.	Capital charges at 12 % per annum per ton milled amount to ..	3 69d.

Saving through Double Stage Crushing £5,580

It appears from the above that the capital cost of the double stage plant, after allowing for extra mill ore bin capacity and for extra pulp elevating plant is only half of that of the single stage crushing plant, quite apart from the fact that the pulp produced by the double stage plant contains 99·5% of – 90.

The saving in capital charges at 12% per annum would be £55·80 per month, or on the basis of 28·5 working days giving a monthly duty of 7438·5 tons, $\frac{55\cdot8 \text{ x } 240}{7438\cdot5} = 1\cdot8$d. per ton milled. A comparison on the basis of the tonnage of – 90 produced is naturally still more favourable to the double stage crushing plant.

In order to be fair to both types of plant when making a comparison under the heading operating or working costs it will be necessary to determine what to include. In view of the fact that costs are kept in different ways on different mines, it is impossible to take collective figures as published in the periodical returns. However, as individual items can always be obtained and easily verified, it is proposed to build up the total operating expenses in this manner. Power costs have already been dealt with and amalgamating costs will be excluded as not being part of crushing expenses. Hence, there remain :—

1. Labour charges.
2. Spare parts and stores.
3. Maintenance and repairs.
4. Sundries.

The operating costs of stamp mills per stamp per day work out as follows (Appendix I.) :—

	Pence.
1. Labour	16·44
2. Shoes and Dies	9·51
3. Screens	3·00
4. Maintenance and General Stores...	25·07
5. Water and Pumping ...	9·80
6. Assaying and Sampling ...	1·21

say 5·5s., or 5s. 6d. per stamp per day.

For tube mills, operating costs per tube mill per day will be as follows (Appendix II.) :—

	s.	d.
1. Labour	22	0
2. Liners	30	0
3. Pebbles	1	6†
4. Maintenance and General Stores	24	0
5. Assaying and Sampling...	1	0
6. Pulp Elevation	9	6
	88	0 = £4 8s. 0d.

Comparing the plants we have :—

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†This allows only for the transport of the pebbles from sorting plant to tube mill plant (see Appendix II.) If the cost of picking pebbles is also to be allowed for, the cost of 2 natives at 3s. per day must be allowed, which adds 6s. to the daily running expense of the tube mill, or a total of 94s. against 88s., and the saving due to the double stage plant will be reduced from 8·218d. to 7·942d.

Comparative Statement of Operating Costs for Single and Double Stage Crushing Plants, excluding Power and Amalgamating Costs.

	Single Stage.	Double Stage.
Cost of operating stamps	44 @ 5·5 = 242·0 sh.	10 @ 5·5 = 55 0 sh.
Cost of operating tube mills... ...	—	1 @ 88/- = 88 sh.
Total cost per day...	£12 2s.	£7 3s. 0d.
Cost per ton milled (261 tons) ...	0·927 sh. = 11·124d.	0·548 sh. = 6·576d.
Cost per ton of – 90	1·241 sh. = 14·892d. (195 tons)	0·55 sh. = 6·6d (260 tons)
Saving per ton milled		0·379 sh. = 4·548d.

Collecting the results so obtained, we have the following statement :—

	Single Stage Crushing.	Double Stage Crushing.	Saving due to Double Stage Crushing.
	Cost per Ton Milled.		
	Pence.	Pence.	Pence.
1. Working Costs	11·124	6 576	4·548
2. Power Costs ...	7·37	5·5	1·87
3. Capital Charges	3·69	1·89	1·80
4. Total Costs ..	22·184	13·966	8·218

Thus a saving of about 8·2d. per ton milled results.

The above saving of 8·2d. per ton milled by no means represents the total benefits to be derived from double stage crushing, as the chief advantage lies in the better extraction which can naturally be obtained from a pulp containing 99·5% of – 90 as against 74·8% of – 90. Of course, it is quite impossible to economically produce 99·5% of — 90 when crushing with stamps only, so that if such a fine product is required the single stage crushing plant need really not be considered at all. However, to show to the full the benefits resulting from double stage crushing by a combination of stamps and tube mills, a value will be placed on the extra work done in crushing finer corresponding with the gain in extraction thus rendered possible. In order to be quite safe in his estimate, the writer submitted the following query to a number of well-known metallurgists :

"What percentage of attraction would you expect in your cyanide plant when treating ore, crushed to the three different stages of fineness shown by the gradings I. to III. hereunder :

Pulp.	I.	II.	III.
	per cent.	per cent.	per cent.
+ 60 product ...	2·5	9	17
—60 + 90 product ...	9·5	16	22
—90 product ...	88·0	75	61

It will be seen that the gradings here given represent the final pulp resulting from the crushing tests at the East Rand Proprietary Mines and at the New Kleinfontein, already referred to.*

Members will probably remember that when these results were published a statement was made to the effect that the gain in extraction due to the finer crushing by the double stage plant "*amounts to about* 1%," and may well ask how this does compare with actual practice. In reply thereto the replies received as a result of the foregoing query are given herewith :—

Name of Mine.	Estimated Percentage of Extraction. Pulp No. I.	II.	III.
Mine A, West Rand (Roodepoort)	93·9	93 0	91·8
Mine B, " " "	95·9	95·5	94·8
Mine C, Middle East Rand (Germiston)	95 0	91·7	89 2
Mine D, Middle East Rand (Germiston)	95·0	92 0	90 0
Mine E, Far East Rand, Boksburg)	95·0	93·0	89·2
Average	94·96	93·4	91·0

The gain in extraction between I. and II. is therefore ......1·56%, and between I. and III. is 3·96%, say 4%.

As we are comparing crushing results which have yielded 75% and 99·5% of —90, or a difference of 24·5%, it is safe to estimate on a gain in extraction of 3·6% when treating the finer pulp, and for a 28s. ore, which is now the average of the Rand, the gain is exactly *one shilling*.

Adding to this the saving in working costs, capital, and power charges, of 8·2d., the total gain per ton milled due to crushing by a double stage crushing plant of 10 stamps per tube mill, as against crushing by stamps only, will be 1s. 8d. per ton milled, which represents a very considerable increase in the total profit.

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* Trans. Inst. Mining and Metallurgy, Bulletin [illegible], p 42.

As in the foregoing remarks, so much depends upon a correct estimate of the probable extraction, it will be as well to support the statement by a few additional data obtained from actual practice. The diagram, Fig. XII., represents the result of the ordinary work at one of the mines from 1st April, 1909, to 31st December, 1910, or 21 months, and clearly shows that an increase in the percentage of —90 product is followed by an increased extraction. In the light of these data it is hoped that the assumption of an increase of 3·6% for an increase of the —90 from 75% to 99·5% will not be seriously questioned.

That fine crushing is essential to obtain a good extraction is obvious, hence the necessity of doing this efficiently and cheaply. It must be admitted that the results obtained by E. H. Johnson will be hard to equal by any other crushing machine, except perhaps a still more effective combination of stamps and tube mills. What the ultimate possibility of this combination is going to be the writer does not care to predict, but it is hardly likely that the point now reached will be the limit.

Having established the probable saving in working costs and capital expenditure in the milling and crushing plant of the double stage type as compared with a plant using stamps only it may be as well to compare the result with that obtained by modifying the sorting and breaking plant.

It will be remembered that in dealing with the influence of the modified sorting and breaking plant upon the whole reduction plant, the type of crushing plant used as a basis consisted of a multiple of 25 stamps and one tube mill, in fact 200 stamps and eight tube mills. It was then found that in the most favourable case the stamp mill was reduced in the proportion of 84 to 43·82, or to 52% of the former capacity, while the tube mill plant remained practically as before, changing only from 63 to 59·2 a reduction to 94% of the former capacity. Hence, the crushing plant will now be constituted as follows:

$25 \times \cdot52 \times \cdot94 = 12\cdot22$ stamps per tube mill.

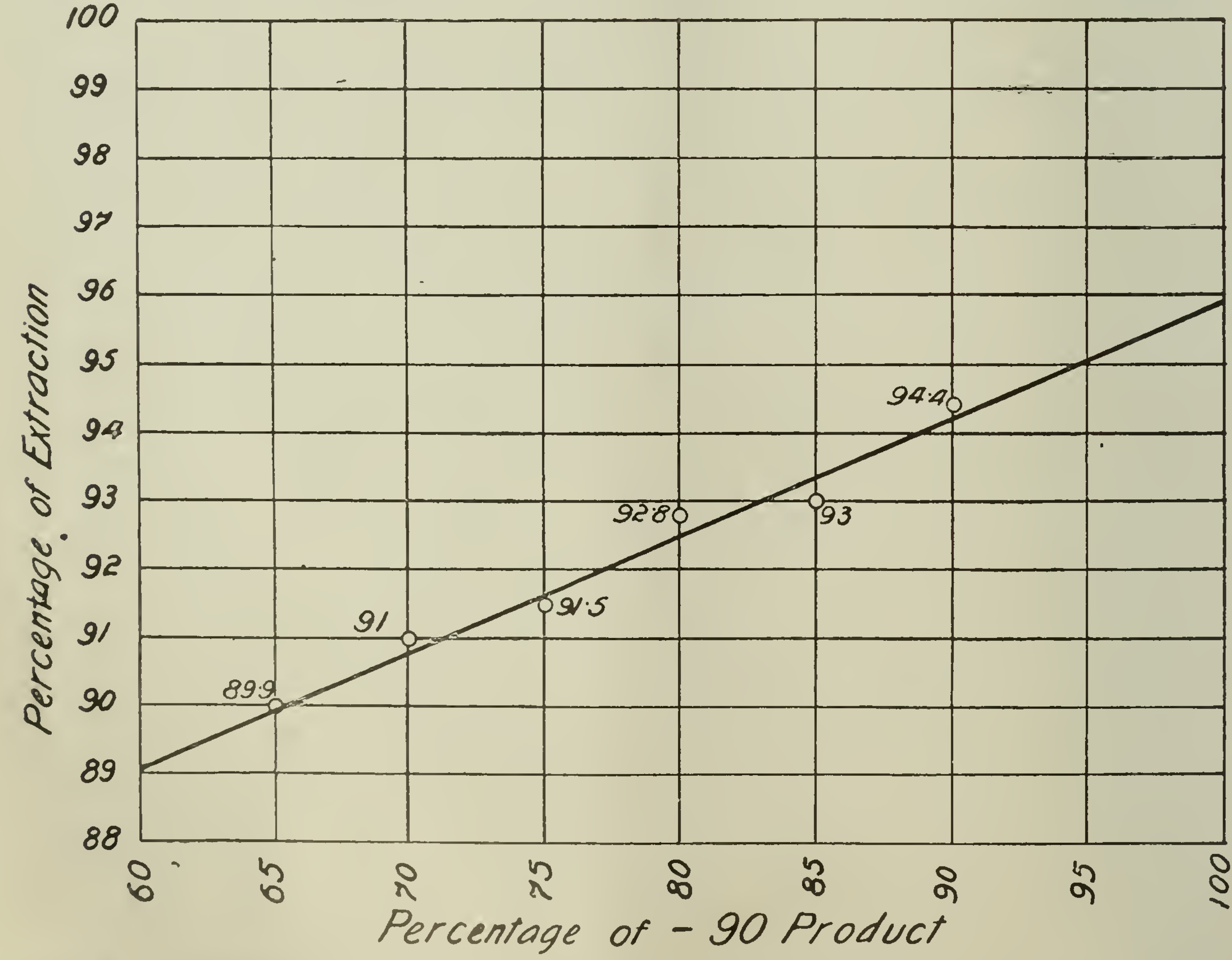

FIG. 12.—Diagram showing Probable Percentage of Extraction for Varying Percentages of −90 in Final Pulp.

It is clear, therefore, that although the milling plant on this basis is a considerable improvement on the average present day plant, it does not yet reach the standard set by the experiment at the East Rand Proprietary Mines, and used in the foregoing as the basis for comparing milling and crushing plants.

Although the saving resulting from a better design of the milling or crushing plant as such, cannot be merely added to that previously obtained by changing the design of the sorting and breaking plant, either result can be improved by a judicious combination of the two. Since the difference between the experimental plant of 10 stamps and one tube mill and the calculated plant of 12·2 stamps and one tube mill is not very great and will not change capital costs or operating expenses to any large extent, we need only consider the better extraction.

No allowance has previously, when dealing with the sorting plant, been made for the possible better extration due to finer crushing on the part of the modified plant, but in view of the result obtained by E. H. Johnson, it seems reasonable to assume that the finer crushing will result in an increase of 10% in the −90 product, thus assuring a better extraction of probably 2%. This, on a 28s. ore is equivalent to ·56s. or 6·72d. per ton milled. Adding this to the saving of 1s. per ton milled (page 22, Table XII.) the total gain, compared with present day practice, is found to be 1·56s. per ton milled. If a comparison with a plant doing the milling in a single stage were possible, the saving would naturally appear even greater.

Considerable changes have recently been made in the amalgamating plant, as shewn in the paper by W. R. Dowling,* and by which a saving in both capital expenditure and working costs will be possible. No doubt, further information on the actual saving will be forthcoming so that there is no need to enter into a discussion of details.

Due to the efforts of Dr. Caldecott† the capital expenditure on the Sand plant can now be considerably reduced, while the introduction of the Butters Filter for the treatment of slime seems to guarantee a better extraction, if not a reduction in capital expenditure and operating costs.

The foregoing has been written not with any idea of dogmatising but in the hope that some useful discussions may arise which will help towards a solution of the one great problem in front of us—the reduction of working cost in order to increase the life of the mines on the Witwatersrand. That our old friends the stamp mill and the tube-mill have shewn such very much greater capacities when used in a more suitable manner, after being with us for a number of years, shows conclusively the enormous inertia in custom and thought on the Rand which is opposed to any change. One feels inclined to ask what would have happened if, on the advice of our self appointed critics with mysterious formulæ, tube-mills had been discarded and crushing by stamps only had been reverted to. Does it not appear likely that, far from the gravity stamp being the surviving part of the present crushing plant, it will have to go and make room for a machine capable of producing a $-\frac{3}{8}$ product more efficiently so as to still further reduce the cost of crushing. That further improvements in all departments are urgently needed a glance at Table I. will prove.

Concluding the writer wishes to thank you for the patient hearing and all those who have directly and indirectly assisted with the data used in these notes, a reference to the source of information having been made in each case.

Appendix No. 1.

Operating Costs of Stamp Mills (excluding power)

Labour.—As a basis has been taken the labour force in an existing battery of 300 stamps, after eliminating the men doing purely amalgamating work. The number of men employed and their pay is as follows :—

6—White @ 18/6 per day,	111/-	
6—White @ 15/ ,,	90/-	
6—White @ 8/6 ,,	51/-	
28—Natives @ 3/ per day	84/-	
Per month		£501 0s. 0d.
1—Fitter (2/3 time) @ £30 per month		20 0 0
1—Carpenter (whole time) @ £30 per month		30 0 0
1—Head Boss (half-time) @ £35 per month		17 10 0
3—Shift Bosses (1/2 time) @ £30 per month		45 0 0
Total labour per month ...		£616 0s. 0d.
Total labour per day ...		£20 11s. 0d.

Cost of labour per stamp per day 1·37s. 16·44d.

Shoes and Dies.—The average cost per stamp per day of six mines, crushing over 250,000 tons, with 1,280 stamps, worked out at 9·51d.

Screens.—The average cost of this item under the same conditions, worked out at 3d. per stamp per day.

Maintenance and General Stores.—For the same six mines the average cost of this item per stamp per day was 25·07d.

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[*] This *Journal*, Vol. XI., p. 111

[†] The Continuous Collection of Sand, this *Journal*, Vol. [illegible] p. 13.

Water, including pumping.—This item worked out at 9·80d. per stamp per day.

Assaying and Sampling.—This item came to 1·21d. pence per stamp per day.

Appendix No. II.

Operating Costs of Tube Mills (excluding power).

Labour.—As a basis has been taken the labour force in an existing plant of 10 tube mills, after eliminating the men doing purely amalgamating work, but allowing for those engaged upon supplying pebbles of banket ore. The number of men and their pay was as follows:—

3—Whites @ 18/6 per day,	55/6			
3—Whites @ 8/6 ,,	25/6			
30—Natives @ 3/- ,,	90/-			
Total for 36 men per day	171/-			
Per month		£256	10	0
1—Fitter (1/3 time) @ £30 per month		10	0	0
1—Head Boss (1/2 time) @ £35 per month...		17	10	0
3—Shift Bosses (1/2 time) @ £30 per month...		45	0	0
Total labour per month ...		£329	0	4
Total labour per day ...		£10	19	0

Cost of labour per tube mill per day, 22/-

Liners.—A set of silex liners has a life of 70 days, and costs installed £105, hence the cost per tube mill per day is 30/-

Pebbles.—As pebbles ore picked in the sorting plant; only the cost of transport need be allowed for which on the basis of 6 tons of pebbles per tube mill per day and at 3d. per ton for transport works out at 1s. 6d. The cost of feeding pebbles has already been allowed for under labour.

Maintenance and General Stores.—The average cost of this item for a number of plants has been found at 24/- per tube mill per day.

Assaying and Sampling.—Averaged under 1/- per tube mill per day.

Pulp Elevation.—The extra power required for elevating the pulp in the tube mill circuit on the basis of 100 tons of solids per tube mill per day, plus 3 to 1 of water, amounts to 120 K.W. hours, and on the basis of ·6d. per unit, this amounts to 6/- per tube mill per day. Maintenance of the pumping plant is estimated at an additional 3/6 per day, thus making the whole item 9/6 per tube mill per day.

The President: This is a great paper in more senses than one, and I am sure you will realise its importance, dealing as it does with the very life of the Rand.

Mr. E. J. Laschinger (*Member of Council*): I think Mr. Schmitt deserves the thanks of every member here and of those who will read this paper afterwards. It is a good thing to have a paper so full of references for actual use if one wishes to go into figures and criticise a paper of this kind. It is naturally a very lengthy paper, but I do not think Mr. Schmitt has drawn it out too long. It deals with a subject which has been considered by the engineers of the Rand ever since we first started to extract the gold from the ore and it will still continue to exercise our minds. I do not wish, of course, to discuss the paper to-night; that is quite impossible simply from hearing it read. It is a paper which lays itself open to a great deal of criticism, and I hope it will get the criticism it deserves. It will give everyone an opportunity of coming forward and giving us their opinions. I have much pleasure in moving a vote of thanks to Mr. Schmitt.

NATIVE FOOD SUPPLIES AND THEIR QUALITY.

(Read at January Meeting, 1911.)

By F. W. Watson, B.Sc. (Lond.), F.I.C., F.C.S. (Member).

DISCUSSION.

Mr. Jas. Gray, F.I.C. (*Member*): This being the first paper as far as I am aware, dealing with foodstuffs, which has been brought before the Society, I consider it incumbent on Mr. Watson's analytical brethren to come forward and show their appreciation of his efforts by contributing their quota to the discussion.

It is, I think, generally recognised that "the gentle art of sophistication" is practised on the Rand to as great an extent as elsewhere and the knowledge that an analytical control is being kept on the food supplies, as well as on other commodities supplied to the mines, will tend to check the progress of this objectionable custom. The many instances of adulteration given by Mr. Watson show that some kind of control must be exercised if a pure and unadulterated article is to be obtained, and it is satisfactory to learn from Dr. McCrae that the authorities are fully aware of the importance of the matter.

The figures given by Mr. Watson as representing the average analyses of mealies are typical, and I desire to contradict Mr. Pooler's assertion regarding the deficiency in nitrogenous principles of South African maize. If Mr. Pooler had studied the subject elsewhere and not relied solely, on Lyster's "Hygiene Advanced," his statement would never have been made.

In the *Transvaal Agricultural Journal*, Jan. 1906, pp. 359/60, H. Ingle gives the results of analyses of 10 samples of mealies grown at Potchefstroom and brings forward for comparison the mean of 154 analyses of American maize. The averages are :

	Potchefstroom Mealies. Per cent.	American Mealies. Per cent.
Moisture ...	7·18	10·95
Ash	1·43	1·45
Proteids ...	9·82	10·40
Fat	4·74	5·00
Fibre	1·87	1·95
Carbohydrates...	74·96	70·25

The proteids in the Potchefstroom samples varied from 9·00 to 11·62%.

The figures which Mr. Pooler produces for comparison with Mr. Watson's are the results of analysis of raw cocoa after the removal of the husk and cannot be compared with the author's which are representative of the commercial product.

In proof of this I will quote Allen's Commercial Organic Analysis Vol. III. Part II. p. 561 : "The large proportion of fat in cocoa (averaging 50%) renders it impossible to manufacture a permanent preparation in the form of powder, without either removing a portion of the fat or diluting the material with some non-fatty matter, such as sugar, starch, or farina. Hence, there are two distinct types of 'cocoa' known in commerce, namely :

1. Preparations commonly called 'cocoa-essence" or 'cocoa-extract,' consisting of ground cocoa-nibs, from which a part of the fat has been removed by heat or pressure.

2. Preparations to which sugar and, generally, some starchy material have been added."

He further states p. 562 : "In addition to the mechanical difficulty of manipulating undiluted cocoa containing all its natural fat, it is stated, with some probability, that the excessive proportion of fat renders the cocoa difficult of digestion. Hence the removal of a portion of the fat, and consequent concentration of the non-fatty constituents of the cocoa, appears to be distinctly advantageous." The brands of cocoa on the local market are similar in quality to those found on the home markets as a reference to any standard text book on 'Food Analysis' will show Mr. Pooler. Instead of the apparent deficiency of fat in the analysis given by the author being an example of the old saying 'Anything is good enough for the Colonies,' the deduction made by Mr. Pooler is a reflection on his own knowledge.

Notwithstanding the exception I have taken to certain of Mr. Pooler's remarks, I would like to congratulate him on a well written criticism, which deserves careful attention.

As stated by Dr. McCrae the Ordinance does not lay down the mode of expressing the percentage of alcohol in Kafir beer, whether it is to be by weight or by volume, and many of you will doubtless recall cases where the prosecution has fallen to the ground owing to the percentage of alcohol being over the limit by volume but under by weight. Personally, I am of the opinion that the practice in England of stating percentages of alcohol by weight should be adopted in this country.

The few Kafir beers I have examined at different times contained the following percentages of alcohol, and it will be noticed that only in one case would a prosecution have been successful :

	A.	B.	C.	D.	E.	F.
Alcohol, by weight ..	2·51	3·32	2·22	2·28	2·45	2·39
Do. by volume ...	3·15	4·15	2·78	2·85	3·07	2·99

It would, therefore, appear that my experience has, like Mr. Watson's, been more fortunate than Dr. McCrae's.

In closing, I would like to draw attention to the unjust manner in which the Public Health work is carried out in the Transvaal to the prejudice of the private analyst. As most of you are aware this work is carried out at the Government Laboratories and although no complaint can be made as to the manner in which the duties are performed, yet it is an instance of Government competition, and as such constitutes a serious hardship for the private analyst. The necessity for this branch of work being in the Government's hands has disappeared, as capable men can be easily obtained now ; and in the second place it interferes with the satisfactory carrying out of the Food and Drugs' Act, as in the event of a private analyst appearing for the defence submitting that no adulteration has been found, the prosecution cannot proceed, whereas in the United Kingdom this is guarded against by the Government Laboratory acting as referee in cases of dispute. Here no such arrangement can be made as long as things go on as they are at present, and the third sample which is taken (supposedly for the referee) is a mere matter of form and a waste of sealing wax. To my mind this is a matter which the Council of the Society should take up, it being the only body in the Transvaal which represents analytical chemists.

I would again thank Mr. Watson for the excellence of his paper and apologise for having wandered into side issues into which, nevertheless, I have been tempted by the wide scope of his paper.

THE MINE DUST PROBLEM.

(Read at January Meeting, 1911.)

By Dr. J. L. Aymard (Associate).

DISCUSSION.

Mr. A. Purser *(Associate)*: There are a few points where the use of my "Dust Arrestor" affects the ventilation question and I would like to hear some discussion on the matter.

I will first proceed to describe it in the form in which it has come most prominently before mining men, namely, its application to drilling dry holes. In order to suspend this device from the working face in such a manner that all the dust coming from the hole must pass through it and go through treatment which separates the dust from the air, a tapered hollow cylinder, or cone, is driven into the collar of the hole. The portion of the arrestor in which the dust is treated is a downward branch leading from the hollow cone and has an opening which throws a spray of air and water within its upper end in a downward direction. This induces a current within the downward conductor and completely wets the dust before it reaches the atmosphere at the foot wall. The same water is used over and over again, thus reducing the quantity of water required.

The hollow cone is sufficiently large to allow the air to be drawn from the upper portion of the working place (as well as the dust from the hole) continually drawing fresh air into the point where the machine operators are breathing, and discharging it at the foot wall, thus assisting the natural circulation.

I may mention that catching and handling dust in its dry state will never get over the dust trouble. In any attempt to get rid of dust there are four points that ought to be aimed at:

(1). Keep the breathing line free from dust and moisture.

(2). Allow the machine man to see the hole he is drilling.

(3). Avoid a device that will compel the miner to waste time rigging it up and removing it when changing jumpers.

(4). Keep the working parts of the drill machine free from grit.

I would like to draw special attention to the latter. The life of a rock drill machine depends very largely on the care that is taken of the front head bearing, and it is a well demonstrated fact that machines working in a rise or other dusty place require reboring and new pistons more often than those drilling water holes, especially when a spray is continually used, therefore an effective dust arrestor will pay for itself in the economic maintenance of the rock drill machine.

A great deal has been said on the question of collaring holes, and my arrestor can be used for this purpose with very little rigging, but a machine will do more work with water than without it, so that as long as water can be used conveniently and without clogging the hole there does not seem to be any demand for a collaring device. But water cannot be used in a back hole after the first few inches.

Although the use of my automatic air saving cock will save a good deal of the compressed air produced on a mine, the question of economy of air is still one of the greatest importance and every mine manager looks for the greatest amount of work from the quantity of air at his disposal. My arrestor can be operated by exhaust or live air. Personally, I like the live air, because it continues to act when the machine is stopped and the jumpers are being changed, and the quantity of air used is probably less per foot drilled than in those dust catching devices which close round the jumper such as the author's and some of my own experiments. This can be illustrated by turning a small quantity of air on to the machine, and it will be found that a very slight pressure of the hand will stop the machine. Not only does the packing round the drill retard the working of the machine, but it makes it impossible for the miner to see that his drill is in line with the hole he is drilling. This leads to wearing away the sides of the drill and further retards the action of the drill, causing extra wear on the cylinder and piston.

The use of exhaust air has some advantages. There is a slight saving of air, the only drawback being that it is not continuous.

I may also mention that a test has been made with a new Butterfly valve Ingersoll drill without an arrestor and an old "Waugh" drill with a "Purser Arrestor." In the latter the air for both the machine and arrestor was taken from a $\frac{3}{8}$ in. nipple close to the machine. The Waugh finished in as short a time as the Ingersoll. I think this will prove that the amount of air used in these arrestors is hardly worth considering.

I find some difficulty in discussing the author's remarks on my device, because (contrary to arrangements made with the author) his remarks do not appear in his paper, or in any other publication, therefore I am only writing from memory and stand open to correction. He says that I told him I got my idea through seeing his collaring iron. I am sorry if any thing I may have said has led him to a conclusion which is so far removed from actual fact, inasmuch as my scheme from first to last is based on artificial circulation.

In my conversation with the author I congratulated him on his determination to press forward the advantage of allaying the dust at its source. But I had been making some little experiments in this direction long before I heard of the author, and six months before I heard of his device I made one on the same principle as his for a Mr. Perkins on the East Rand Proprietary Mines. I remember distinctly the conversation to which he referred. We talked more of construction than anything else, and I told him that I intended to make my arrestors of boiler tube expanded at one end. This scheme I have since abandoned as being too expensive, and I expect that within the next few months, with the extended use of these arrestors, we shall be able to fit up a plant which will turn them out at a price which will leave nothing to be desired.

In my device for allaying dust the draught is made to flow from the top of the drive down through a pipe until it comes out at the bottom, thus encouraging the natural tendency for the lighter and purer air to come into the working place close to the hanging wall and to return close to the foot wall. The author in his paper supports the theory, but omits to remark that this tendency is so mild that (without employing artificial means) it may easily be upset by the traffic in the working place or drive. I think it is quite reasonable to assume that a strong current of air passing through the working places of a mine will collect dust and other impurities, and will throw off those impurities wherever it slows down. In this manner a strong current will throw off its impurities at either side of its course into cross cuts where miners make up their charges, into dead ends of drives and stope benches. The main body of the air current carries very little dust with it, all the dust being in the fringe of the current. Therefore it is the duty of my device not only to separate the dust from the air at the dead ends, but also to draw a fresh supply of air from (as nearly as possible) the centre of the current so as to dilute the gas laden air in the working place.

I have read with interest Mr. Price Griffiths' discussion on the author's paper, and I was particularly interested in his remarks on spraying. A few years ago I tried some experiments for solidifying dust, using a solution of one part of unrefined molasses to about 99% water, and I find that it will go a long way towards meeting Mr.

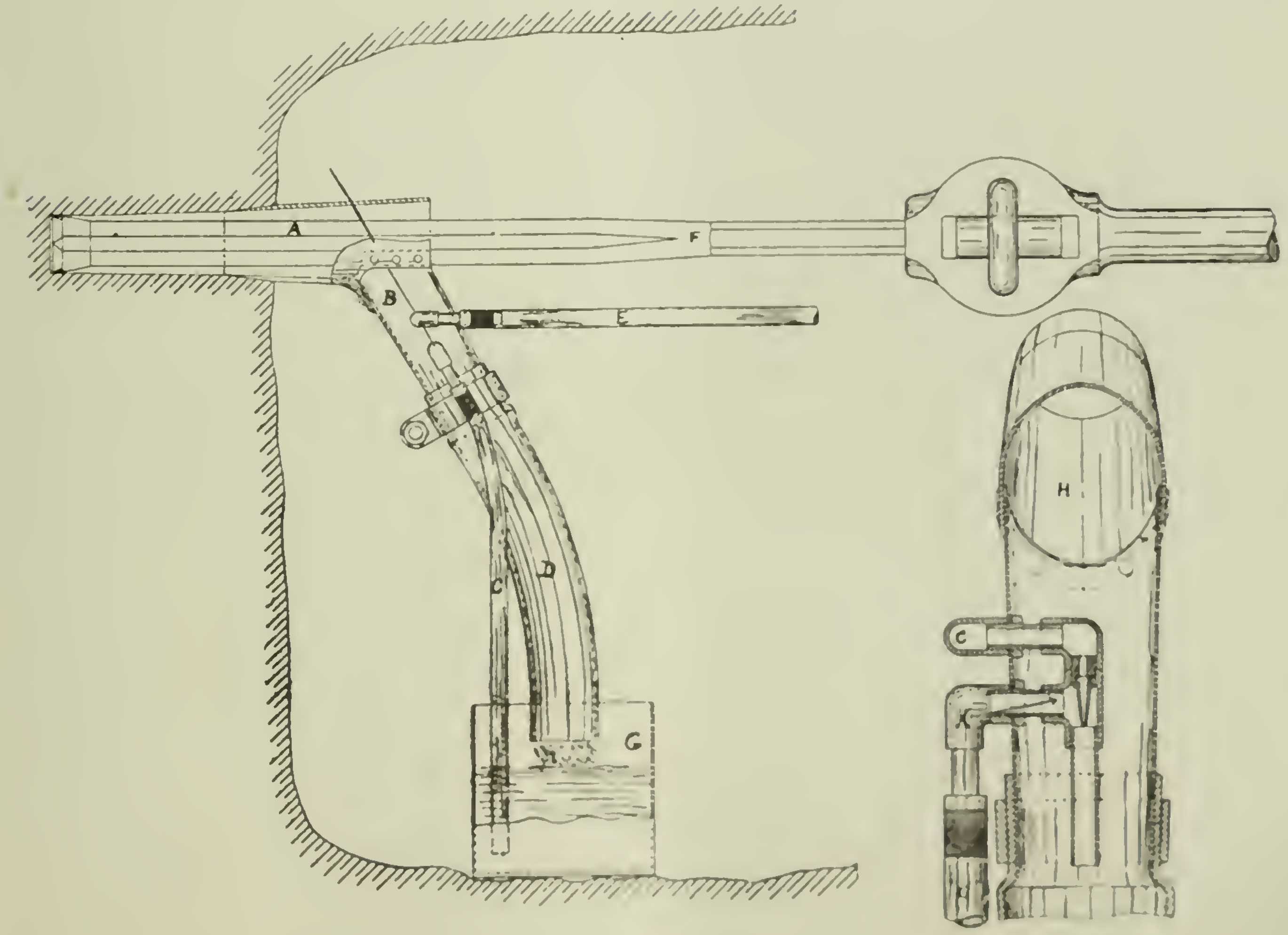


Fig 1

Fig 2

Price Griffiths' suggestion. I do not know what effect this mixture would have on the plates in the mill, but using it in such small quantity I do not think any bad effect would be noticeable. The mixture could be made in the tanks, or sumps, from which the water supply for the drives is taken and can be used with the "Hewitt spray" and also in connection with my dust arrestor.

Before closing, I must thank the management and officials of the East Rand Proprietary Mines not only for the opportunity they afforded for experimenting but also for their assistance in selecting the most suitable combination from the various alternatives.

The President: I am sure we must all thank Mr. Purser for coming here and giving us this very interesting demonstration of his device. Now that the Rand has two such simple devices for catching the dust we ought to hear no more about it if we can only compel the miner to use them. What we want is strict regulations.

Prof. J. A. Wilkinson (*Member of Council*): It was not my intention to say anything on the subject of this paper, but as this is the time set down for final discussion it may be better to make a few remarks now than postpone them to the period immediately preceding the author's reply. It will be remembered by many of you, that this subject, around which so much fierce discussion has raged, was first introduced to this Society by a paper read by Mr. Cullen in February 1903. Since that time very few slides of the actual dust given off by machines drilling into the banket have been shown. In October 1903 Mr. Heymann and Dr. Pakes showed specimens, the former from the actual lung of a miner who had died from the disease, the organic matter having been destroyed and the latter dust taken from a drive, the larger particles being removed by sieving. At the meeting at which the author read his paper, when it was unfortunately impossible for me to be present, your President showed you another slide of the dust taken from the author's dust captor. The slides, which I have prepared, were taken some time ago by the following method. A sample of pure glycerine, a substance which is used largely in microscopic work, was treated in such a manner as to be free from dust particles, being finally tested by the microscope to make certain that this was the case. This was enclosed in a bottle fitted with a tight rubber cork through which passed a glass rod dipping into the glycerine. Glass slides were cleaned carefully and whilst the drills were working in the ordinary manner, one was taken, a small drop of glycerine placed on it and an exposure to the dust made for a certain time. A clean glass coverslip was then pressed over the glycerine and the slide then brought into the laboratory for examination. Photographs were taken in the ordinary manner. I am sorry that we have not the lantern here this evening to show you the actual results obtained. This I hope to do at the next meeting. I may say, however, that these show the highly dangerous character of the dust, such as one would expect from a rock mass of the nature we have to deal with here, and from which any man, who has the least concern for his own life, should only be too ready to protect himself. I am indebted to Mr. J. Whitehouse, Manager of the Village Deep G. M. Co., for procuring these slides in the manner above mentioned, and also to my colleague, Professor Young, for the use of his micro-photographic apparatus.

There are one or two points furth r in the paper, to which I should like to refer. The first is the question of ventilation, and in this respect I think that air currents should not travel at too great a velocity, otherwise serious dangers are liable to arise from this cause alone, even assuming that such air is perfectly free from all dust. If not free from dust then it is obvious that the dangers, however great, would be increased due to the inhalation of dust with the air in quantities greater than is ordinarily the case. The contamination of such air in other ways is also a source of danger. Secondly the author states that "the medical aspect of the dust problem remains to-day very much as it did ten years ago." I am sorry to say that I agree with this statement and my sorrow is the more because we had hoped that some progress would have been made in this direction during the years since this question came prominently before us. Only once before have I discussed this question in this Society, and at that time I stated that there was a lack of medical research here on this matter. As far as I am aware very few histological investigations have been carried out, and hence we are not in a position to state what exactly happens to the dust when inhaled and passed below the bronchioles. How does it pass from thence to the infundibula, to the lymphatics and to the blood stream? Where is its final lodgment and how does it cause fibrosis? What is the part played by poisonous gases such as CO, NO_2, etc.? All these questions remain still a matter of conjecture apart from the conclusions which may be drawn from clinical evidence or mortality statistics. In short we may agree with the author when he states that it is quite clear that the pathology of the disease is still a matter for research, a question upon which, as a member of the medical profession, he is eminently qualified to speak. It is interesting in this connexion to observe the recent debates, which have taken place in the House of Assembly during the last few days on the Miners' Phthisis Bill now before Parliament. If memory serves aright, the Minister of Mines, in replying to the debate, stated that it was his intention to appoint medical officers for this particular work. Such a step will be heartily welcomed and will, we hope, serve to bridge the gaps mentioned above. Much of the work must, as the Minister stated, be of a confidential nature, but research into the causes and course of the disease should be open to full discussion, since otherwise the possibility of drawing erroneous conclusions cannot be avoided. It is therefore sincerely to be hoped that, during the next few years, the pathology of this disease will be considerably elucidated. One problem which calls for solution is the number and kinds of bacteria present in mine air, and a bacteriological survey of any mine would yield most valuable information, more particularly as to the prevalence of ordinary phthisis as compared with silicosis, and as to whether the dust formed a means whereby bacteria were absorbed into the body in large numbers.

Mr. A. J. R. Atkin (*Member*): Towards the end of the year, when those plants installing Butters filters are completed and in running order, a quantity of calcium chloride solution will be available for using in sprays and for damping rock before shovelling.

As this is produced as a by-product from cleaning the leaves, it will alike assist the reduction works with a credit balance and the mines with a cheap dust allayer.

PROFIT PER FATHOM.

(*Read at February Meeting, 1911.*)

By R. E. Sawyer, A.R.S.M., A.I.M.M
(Associate).

DISCUSSION.

Mr. A. Richardson (*Member*): The fathomage system was so comprehensively criticised in the local press at the time of its proposed introduction that the author is to be congratulated on being able to present some of its features in a new and attractive light. The position he takes up will be considerably strengthened if he gives us in his reply some favourable data taken from actual practice, as figures compiled merely as examples, though a stimulus to discussion, lead to hypothetical conclusions foreign to the temperament of the practical mining man. The tables he gives might repay study if one had leisure enough to dissect them; but it is extremely doubtful, for the reason mentioned above, if such an analysis would be very helpful. It may be noticed in passing, however, that the compressor charges in Table I are distributed proportionately to fathoms, irrespective of stoping widths, apparently on the supposition that the drilling of a fathom consumes the same quantity of air no matter what its stoping width may be, an obviously unfair assumption. It would also be interesting to know whether the additional cost of shovelling in the narrow stopes is duly reflected in the cost per fathom; and also if 122 in. is not an extraordinarily large average stoping width for ordinary machine stopes, even after the hammer and small machine stopes have been eliminated. In Table III a stope 143 in. wide assaying 2·4 dwt. shows a profit of 24s. per fathom or 1s. 6d. per ton mined; this excellent result seems to merit a special investigation, for, if correct, it means that the potentialities of the Rand as a gold-field are not fully realized.

Regarding the value of waste, the writer once, out of curiosity and in the interests of science, took representative samples of all the partings and of the hanging and foot in every stope in a mine. The average value worked out at about 0·3 dwt. per ton, and there were no payable values shown. The band of micro-quartzite, commonly known as "black bar" or "interbedded dyke," to which the author refers, immediately underlies the Main Reef leader in the mine in question, and though, as a rule, it never carried more than 1 or 2 dwt. it, on one occasion, furnished quite a respectably sized patch which averaged 10 dwt. over a width of 18 in. This of course was stoped and sent to the mill. Occasionally also it paid to get stoping width by breaking into the black bar, so that the unsortable waste fines should carry something; but as an excellent footwall for breaking to was lost by this practice, and as the black bar was tougher than ordinary quartzite, the extra cost of breaking the ground perhaps counterbalanced the very slight increase in grade. The suggestion that the screen assays should be checked by working out the daily grade for the whole mine would, if acted upon, furnish only a rough estimate, useless as a check, as the variations in stope value are not determined by daily, but by monthly sampling: the statistical work involved would also be considerable.

Speaking generally, the square fathom principle may have its uses in certain technical estimates, but as it requires specialized knowledge to grasp its import it will never commend itself to investors, who are commonly laymen in such matters. The system does not seem to offer any advantages over the present system when employed for drawing comparisons between results obtained during different periods in the same mine, more especially if the ton mined is employed as the unit. Its range of applicability is small as it cannot be used for expressing the results obtained from diamond pipes, very wide reefs, irregular impregnations, masses, and so on. It would, however, appear to find its truest application in coal mining where the exceptional regularity of seam should lend itself admirably to the expression of results on the square fathom basis; indeed it seems strange that a principle of such supposed merit should never have been considered by the capable organisers of that old and well conducted industry. In the statement of ore reserves, again, the square fathom cannot be adopted, for, being a variable and not an absolute unit, the number of square fathoms of ore in reserve would afford no indication whatever of the life of the mine and therefore would convey no hint of its commercial possibilities. In all statistical presentation, results must, in any case, continue to be given on the tonnage principle, as a concession both to the unenlightened public and to those enlightened ones who are opposed to the suggested innovation; consequently, the new system, if adopted, would necessitate double statistical work. On the practical side, the tendency would be towards selective mining, as low grade reefs in the hanging or foot, which can be shown to produce a profit under the present system when broken with the rich leaders, would, in many cases, be allowed to remain unbroken so that the profit per fathom might be made as large as stoping conditions would permit; the grade of the ore would be raised, but the life of the mine would be considerably shortened, and this doubtful result could just as easily be brought about under the present system, if thought desirable.

PRACTICAL NOTES ON COAL.

(Read at February Meeting, 1911.)

By Michael Dodd (Member).

DISCUSSION.

Mr. R. Gascoyne (*Member*): It must be admitted that this paper has dealt with the subject of coal from numerous standpoints, but its value has undoubtedly to some extent been sacrificed by the general, rather than the detailed, manner in which the subject has been dealt with.

For instance, the author begins by saying that in his young days it was estimated "that a colliery might be expected to consume in its own boilers about 10% of its output, but in less than a generation an enormous change has come about and the engineer of to-day is never worth his place unless he knows a great deal more about the character, source and cost of his power supplies, than the colleges were in the habit of teaching five and twenty years ago." Now a statement like this might be perfectly true of a few isolated collieries in Northumberland and Durham, where they use antiquated boilers and equipment and perhaps handle in an expensive way fair bodies of water, but outside Northumberland and Durham, to which the author probably refers, no engineer who allows 10% of the output to be used for colliery consumption would be considered worth his place even thirty or forty years ago. But then Northumberland is the cradle of coal mining, and up to a few years ago, at all events, several prehistoric mining equipments were in regular use, which no doubt would tend to a high consumption, whilst likewise, several collieries do considerable pumping still, but to make a statement that say thirty or forty years ago a colliery might be expected to consume 10% of its output ought certainly to be qualified and made to apply in a few instances where unusual conditions prevail. Then again it ought to be pointed out that in the author's young days as to-day, the English collieries only consume under their own boilers the inferior portion of their output which it is impossible to sell, which, of course, will add to the percentage of the output consumed. The fine coal such as that generally consigned to the waste heap in the Transvaal and the refuse from the washery is generally used under the colliery boilers, but even then any decently equipped colliery, even twenty or thirty years ago, would not use anything like 5% of its output, and in most instances the consumption would be in the neighbourhood of 3% where the output was anything like a thousand tons a day and for a greater output the consumption might be less. In the Transvaal the colliery consumption runs from 1% to 5% of the output, the latter covering unusual conditions such as do not ordinarily prevail at local collieries.

With regard to coke it is generally admitted that, whilst nearly all coals will make coke, it requires a coal much less in ash than the best Transvaal coal yet found, to make a really first class coke, and besides, a coal that will coke readily ought to contain about 4% of free hydrogen. Roughly, we may consider that coking a coal doubles the percentage of ash, so that any coal from which it is desired to make a saleable coke ought to have less than 5% of ash before being put into the oven, and it is doubtful whether even after washing the best coals of the Transvaal they can be sufficiently reduced in ash contents so as to make a really first class coke, even if the other necessary conditions were present.

The author's statement on p. 362, that Middelburg coals with volatile contents from 22% to 24% may be expected to give the best results as steam coals, is too vague and loose to be taken as a good definition of a steam coal as there are other constituents besides ash and volatile contents to be considered. There is, for instance, the coal this side of Oogies in the Middelburg district with 22% of volatile matter but so inferior for steam or heating purposes that its evaporative value is only 8 as against 12 for the best steam coal in the Transvaal. To be able to judge the quality of coal for any purpose it is necessary to know the whole of its contents, not ash and volatile only—a coal may carry 15% ash and 23% volatile matter according to the author's standard of what a good Witbank steam coal ought to be and then be so lignitic in quality as to be worthless as a steam coal. There are numerous instances of this even in the Transvaal. For an ideal steam coal in the Transvaal the following rough composition would be better: Fixed carbon 70·%, volatile matter 20·%, ash 10·%.

The author, at the conclusion of his paper, recommends the American system of making contracts for the purchase of coal by which payments are made on the ash factor plus calorific value, and says "that such conditions appear sound and are certainly effective." Now, as a matter of fact, such conditions of making a coal contract are neither sound nor effective, unless they include a minimum clause, a recent instance having occurred where such conditions were embodied in a contract and enabled the seller to dispose of the whole of his dump and waste heap at an excellent price, thanks to the adoption of the American practice, although under the ordinary conditions of contract the coal was unsaleable and had gone to the dump for the last two or three years.

THE AMALGAMATION OF GOLD IN BANKET ORE.

(*Read at March Meeting, 1911.*)

By W. R. Dowling, M.I.M.M. (Vice-President).

DISCUSSION.

Mr. E. J. Laschinger (*Member of Council*): I will open my remarks on this paper by a few carping criticisms, and trust the author will not take them amiss.

Referring to line 1, amalgamation is a method or means of gold recovery and not a "source.

In paragraph 2 the author seems to speak of a minimum of simplification being necessary in plant and operations—he surely does not advocate a maximum of complication.

In paragraph 3, the author in referring to Rand gold ore milling practice before cyaniding came in says that the screens mostly in use were from 800 to 1,000 holes per square inch: my own recollection is that they were more commonly from 1,000 to 1,600 mesh.

Regarding the author's remarks on pulp elevation in relation to the various schemes of working, I am afraid that the author has missed the most important point in regard to this matter. The chief factor generally determining the number of times pulp has to be elevated and the amount of lift at each elevation is the surface contour on the site of the plant. That the process of ore reduction and gold extraction adopted will to some extent influence the pulp elevation is, of course, admitted. All known methods require some kind of scheme of flow, and the simplest sequence of processes will, as a rule, require the least pulp elevation, other things being equal. My point will perhaps be best illustrated by referring to the author's diagrams. It is evident that if in arrangement A the first elevation were higher so that the pulp could flow to the point of delivery of the second elevator its flow diagram would be similar to B. Or if the author had drawn in on B the second pulp elevator to the cyanide plant as it actually exists at the Simmer and Jack, he would have had the same flow system as the Randfontein Central.

Again, if in arrangement C, which represents the City Deep plant, the first elevator were slightly lowered the return from the tube mill plates could gravitate to it, and the second elevation would disappear and be similar to arrangement B (Simmer and Jack). In point of fact this scheme C thus modified represents the arrangement at the Bantjes Consolidated Mines. The arrangement at the City Deep was made different from the Bantjes, because in the former case the ground is very flat whereas in the latter there is a rapid fall. It was solely this difference which influenced the design at the City Deep. If a single elevation had been decided on, the lift would have been so great as to cause most undue wear and tear on the pumps and much trouble all round. As it is, the elevation was purposely split in two, with the lift in the second pump higher than in the first. It is interesting to note as the result of the experience in actual operation, that with the first pump elevating coarser particles at lower lift, and the second handling a finer product at greater lift, the combination is so well balanced that the liners in both pumps last for equal lengths of time.

In the author's proposed arrangement E he has very conveniently placed his stamp mill high up in the air, or else sunk his tube mills and plates deep into the ground, thereby saving one pulp elevation. His scheme, as applied to almost any mine on the Rand, would involve a pulp elevation from the stamps to the tube mills, or else very expensive excavations or foundations.

In considering pulp elevation one must not forget two other things, the elevation of the ore to the mill bins and the re-elevation of return water from the tail to the head of the plant. The whole flow scheme, as far as the engineers' designs for mechanical handling in ore reduction plants are concerned, commences at the pit mouth and ends at the residue dumps. Enough has been said to show that Mr. Dowling has made rather too much of saving in pulp elevation as affected by methods of amalgamation of gold.

From the standpoint of obtaining ideal conditions in ore reduction, which ideal I think could and should be attained, all the schemes outlined by the author are at fault. I refer more particularly to the reclassification of tube mill pulp. We do not reclassify battery tails for the purpose of returning the oversize back to the mill bins to be restamped, and the principle of returning a large percentage of tube mill tails back to the mills is fundamentally wrong. The amount of returns is, I believe, very much greater than is generally realised, and unsatisfactory classification is one of the most important evils which is aggravated thereby.

Another point affecting the elevation of pulp is that in all Rand practice to date a great deal of unnecessary elevation of pulp is done because of considering the stamp mill as one unit and the battery of tube mills as another independent unit. All the pulp is gravitated to one point from 200 and more stamps, and this with steep grades of launders means a very considerable loss of height. The pulp has again to be redistributed from one point to the heads of the tube mills again at grades even steeper than in the mill launders and re-collected at the tails of the tube mill plant with more loss of height together with a return launder again to the stamp mill pulp elevator. In this matter, using the words of the author in another connection, we are also following ideas imbibed from ancient practice when ore reduction plants were small and grades of launders comparatively flat. The alternative is to treat a certain number of stamps and one tube mill as an independent unit, thus saving unnecessary pulp elevation and what is much more important making classification, what it should be, simple and effective. Such an arrangement in combination with crushing and fine grinding in one pass through the combined apparatus forms an ideal scheme, and the amalgamating plates can be placed where most desirable, whether up in the air, on the ground or in an excavation. I trust I have not wandered too far away from the subject matter of the author's paper in discussing the above points.

The author's point that fine free gold in the sand is not recovered in the same proportion as the average extraction by cyanide treatment of the whole sand charge is well taken.

The locking up of gold on amalgamating plates is not an argument in favour of cutting down the total plate area to an absolute minimum : it is only an incidental advantage or on argument in favour of not having excessive plate area. The amount of plate area should solely depend upon the capacity of the surface to catch such gold as cannot be so efficiently recovered by cyaniding.

From the author's remarks on flow over plates, as regards velocity and depth of the pulp stream it would appear that what is known as turbulent flow has as much or more to do with bringing particles of gold into contact with the amalgamating surface than the idea of giving the heavier particle of gold time to sink to the bottom of a more quiet stream. It is well known that most gold collects where there is fall or impact of the stream against the plate.

The author says he "considers that sufficient importance is not attached to launders." I think the subject has received a good deal of attention in recently erected plants. As an example with which I happen to be familiar, I might refer those of an enquiring turn of mind to the pulp launders at the City Deep. Here all main launders in the mill and from the shaking tables are formed in solid concrete, and all main overhead launders in sheet steel concrete-lined. There are a few straight distributing launders of wood, lined with bitumen sheeting. When the author recommends "cement" for lining launders it is presumed that he means fine concrete in Portland cement.

Although the author deprecates amalgam traps, the advisability of having these rests upon two grounds. In a system where there is a tube mill return the trap is not so necessary as in a system where there is no return. The trap in the latter system would of course be advisable to catch coarse amalgam which would have no chance of being caught again. But even when there is a return the so-called amalgam trap is useful in catching particles and pieces of iron and eliminating them from the circuit. This is especially important in view of the economic failure of most magnetic separators that have been tried on the Rand. In fact it is a question if an iron catching trap should not be installed in the mill launders to avoid the trouble frequently experienced owing to pieces of iron having lodged in the delivery to the tailing pumps and also choking classifiers. There is also the problem of black sand which is caught in traps. Does the author also maintain that the gold in this would be as efficiently extracted in the ordinary treatment of the main plant as in the special plant such as is referred to by Mr. Allen in his paper on Air Agitation, or does he consider that all the so-called black sand is caught on the plates when scraped when there are no traps?

With regard to shaking plates *versus* stationary plates it will be interesting to note what the results will be. It is for the metallurgist to say which is the more efficient and economical, and to thresh out the arguments as to what are the real principles underlying shaking plate amalgamation, and under what conditions they could be used with advantage over stationary plates.

Our thanks are due to the author for bringing up for discussion this subject of amalgamation plate area, and for his compilation of data on this subject. It is to be hoped that all Rand metallurgists who have conducted experiments and collected original data on this practical problem will come forward and publish their results.

Mr. Mather Smith (*Member*): Mr. Dowling's paper is just what was wanted, especially by the outside districts, in that it gives comparisons of the different methods of classification adopted by the various Rand groups of mines

On this, the Worcester G. M. Co., in the Barberton district, we require a specially good classification as the gold is silica-encased, and the + 60 in the cyanide residue frequently assays over 2 dwt. Six vats which had an average charge value of 2·9 dwt. gave, after over nine days treatment, the following average values in the residues:

+ 60 = 1·85 dwt.
+ 90 = ·95 „
− 90 = ·38 „

It appears to me that to try and understand the work done in a spitz, or cone, one should try and imagine oneself to be a particle of + 60 struggling to get down through a mass of − 60, or as a fine particle of − 60 trying to squeeze up through many particles of coarser sand. The particles must have time to separate out properly, and this they cannot do in the series of spitz shown in the author's sketch D. or common arrangement.

When one of the nozzles on this Eiffel-Tower-like spitz chokes up, the first person to notice it is frequently the boy in the sand collector, and by the time the nozzle is cleared there is often quite an appreciable quantity of coarse sand in the collector. With Mr. Dowling's proposed arrangement the shiftsman will be able to see at a glance by simply examining the pulp issuing from the safety-cone, exactly what is going to the cyanide plant. At present he has to climb up a precipitous ladder, a feat which he is not likely to perform any oftener than he can help, and further, a cursory examination of the whole stream is not likely to give one much idea of the grading of the sand pulp.

One objection I have heard to the idea is this: That the + 60 caught in the safety cone would accumulate in the circuit and overflow to the cyanide plant before the tube mill shiftsman had time to rectify things. To this I reply that this + 60 would continue, for quite an appreciable time, to replace − 60 which otherwise would unnecessarily be going through the tube.

The dewatering cone in the common arrangement I could never, with my limited experience, quite see the necessity of. It has always appeared to me that a 12 in. pipe throttled at the bottom would have done equally well. I notice that two diaphragms are now being put in cones. I would be inclined to prefer an idea of Mr. Chas. Dowsett of the Knights Central. He, I am informed, intends trying the efficiency of a plate, with a hole in the centre, in the position of the second diaphragm, and this, I should think, would make the route of the pulp more tortuous than two diaphragms would do

I have never noticed any paper or remarks on another problem, namely, at what size of screening the amalgamating plates in front of the mortar boxes cease to give efficient results, and think some such paper would be of great use to the outside mines.

AIR LIFT AGITATION OF SLIME PULP.

(*Read at March Meeting, 1911.*)

By Robert Allen, M.A., B.Sc., M.I.M.M.
(Member of Council).

DISCUSSION.

Mr. E. J. Laschinger (*Member of Council*): Mr. Allen's paper is an interesting compilation of the history of and present practice in air agitation of pulp as applied to metallurgical processes.

In criticising this paper there are various points of greater or less importance that may be touched upon, but in the most important question raised—air lift versus mechanical agitation—the author has missed the real point. He has failed to differentiate between agitation as the simple aim in keeping ore particles in suspension in a fluid and agitation as a means of intimately bringing into contact ore, chemical reagents and air so as to thoroughly facilitate certain chemical reactions.

If the problem on the one hand is simply to bring particles of ore into thorough contact with cyanide solution, or to keep ore from settling in water or solution, or on the other hand to also bring air for the purpose of supplying free oxygen to this intimate mixing process, the two cases are not on a par when considering simply the most economical means of agitation. All the above cases arise in practice. If air be necessary as a chemical adjunct, it is of course self evident that subsidiary mechanical stirring is superfluous if the air itself by its method of introduction can perform the mixing at a reasonable cost without other aids.

The author has not proved his case in favour of air agitation in a satisfactory manner, because he has not even attempted to differentiate between the various cases above mentioned. Even his figures on power required for air agitation are not reliable, or perhaps one should rather say are not consistent. This I will endeavour to prove.

It requires a certain amount of power to compress air, and the amount of this power can be accurately determined. The theoretically minimum work required to compress air from atmospheric pressure to any other pressure is by isothermal compression, *i.e.*, without rise of temperature during the process. The work thus required to compress a cubic foot of air is :—

$$(1)\ \text{Work in ft. lb.} = 144\ P_a\ \log_e \frac{P_1}{P_a}$$

Where P_a is the absolute atmospheric pressure in lb. per sq. in., P_1 is the absolute pressure of compression, *i.e*, gauge pressure plus atmospheric pressure, say, $P_g + P_a$.

The horse power required per cub. ft. of air at atmospheric pressure compressed per minute is therefore :—

$$(2)\quad (\text{H.P.}) = \frac{144\ P_a}{33{,}000} \log_e \frac{P_1}{P_a}$$

For conditions at sea level with a barometric pressure equal to 14·7 lb. sq. in., we have :—

$$(3)\ (\text{H.P.})_S = 0{\cdot}0642 \log_e \frac{P_g + 14{\cdot}7}{14{\cdot}7}$$

For conditions on the Rand with average barometric pressure equal to 12·1 lb. sq. in. we have :—

$$(4)\ \text{H.P.})_R = 0{\cdot}053 \log_e \frac{P_g + 12{\cdot}1}{12{\cdot}1}$$

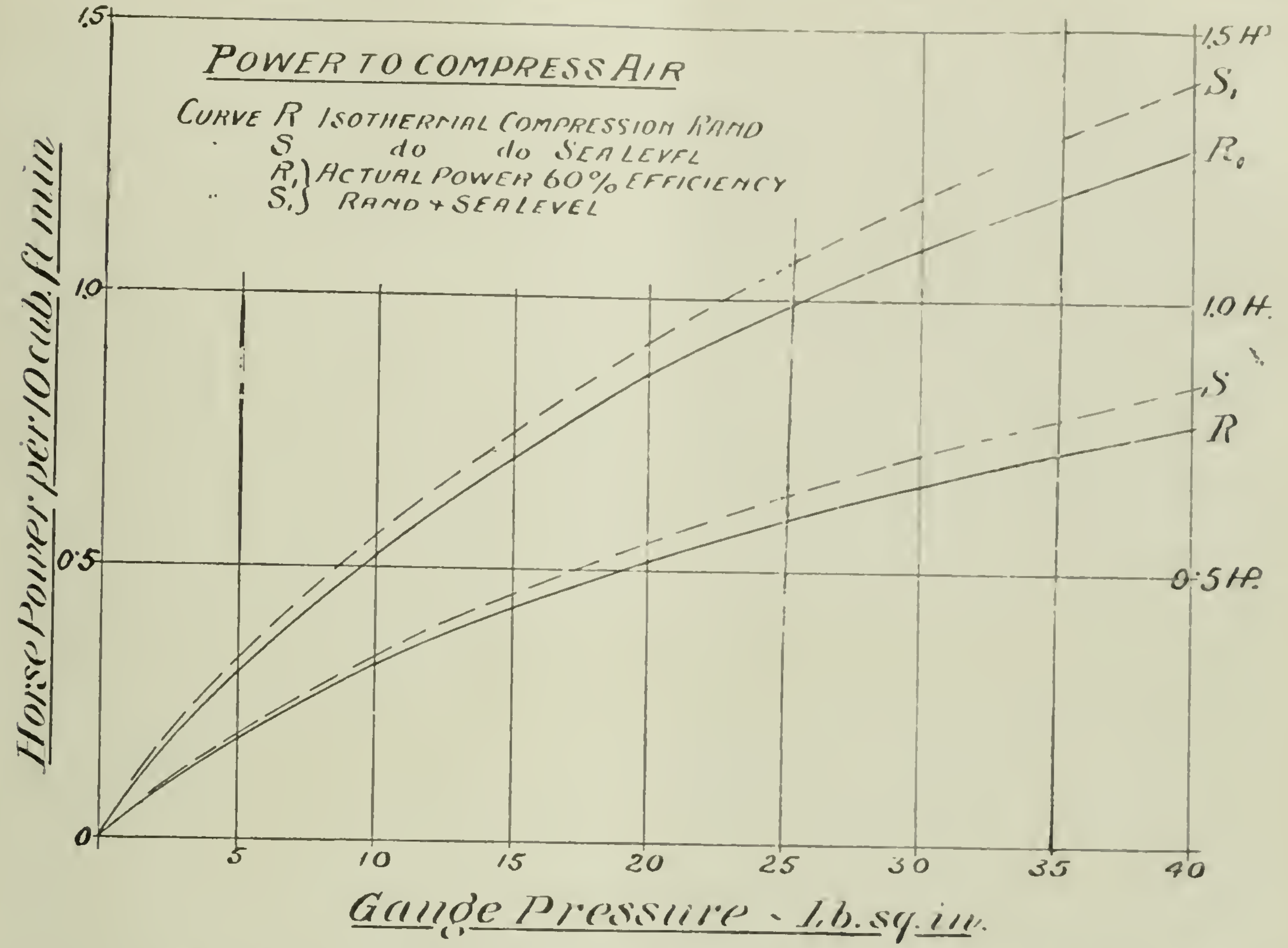
The above give the theoretically minimum figures of power required. In practice there are power losses in compression, friction losses in the motor and compressor, heat losses in compression, and leakage losses past valves and pistons. The over-all efficiency of small compressors is probably not more than 50%; even in large compressors the total efficiency will not be much greater than 65%. Assuming the favourable figure of an all-round efficiency of 60%, the actual power required will be $1\frac{2}{3}$ times the figures given by formulæ (2), (3), and (4). In the diagram herewith I have shown curves from which the power required to compress 10 cub. ft. of air per minute from atmospheric pressure to gauge pressure up to 40 lb. per sq. in. Curve S shows the theoretical horsepower required at sea level, and curve R the same for average Rand conditions; curves S_1 and R_1 the corresponding power with an over-all efficiency of 60%.

Before using this diagram to criticise certain figures quoted by Mr. Allen I crave indulgence for a digression in order to point out a few characteristics of air agitation. A glance at the diagram shows that it requires less work to compress a cubic foot of air at atmospheric pressure (generally called "free air") to any given gauge pressure on the Rand than at sea level. If we assume that the amount of work got back from the compressed air in stirring the pulp is the same percentage of the work put into it in the two cases, the conclusion is that for the same amount of agitation more volume of free air would have to be used on the Rand than at sea level to produce the same mechanical effect. The amount of air used on the Rand would have to be increased in inverse proportion as the work per cubic foot is greater at sea level than on the Rand.

If a certain amount of air be required to satisfy chemical reactions, then the air required must be measured by weight and not by volume. Since air at sea level averages 13·2 cub. ft. per lb. and on the Rand 16 cub. ft. per lb., it follows that in order to deliver the same weight of air $21\frac{1}{2}$% more volume of free air must be compressed on the Rand to furnish the weight of air that would be delivered at sea level. For this case, therefore, the amount of power required would also be greater on the Rand than at sea level. Curves of power required for equal weights of air delivered would on the diagram show the R curve above the S curve. In general the above reasoning is true for different altitudes.

It may also be of interest to discuss the point as to how much of the power put into the air is really effective as mechanical work in agitation. Air agitation in a Brown vat with the central pipe is the reverse of hydraulic air compression. In tests made on this form of compression the the over-all efficiency ranges between 70% and 80%. It may be assumed that the same ratio holds good in air agitation. The over-all efficiency of air agitation from work in the motor to agitation in the vat is therefore, say,

$$0{\cdot}60 \times 0{\cdot}75 = 0{\cdot}45 \text{ or } 45\%$$



Taking now the author's first table of average results of experiments by F. C. Brown as given, and assume for argument sea-level conditions, we have the following comparisons, taking the quantity of air used as correct :—

1st case h.p. per table	$\frac{3}{4}$ – 1	from diagram	1·1 – 1·4
2nd " " "	$\frac{1}{3}$ – $\frac{1}{2}$	" "	0·4 – 0·6
3rd " " "	1$\frac{1}{2}$ – 2	" "	1·6 – 2·2
4th " " "	1$\frac{3}{4}$	" "	3·9

In the fourth case the theoretical horse-power required works out at 2·4, therefore the amount of power quoted, namely, 1$\frac{3}{4}$, may be dismissed in certain familiar words of our old friend Euclid.

In actual practical design of plant I would make even a liberal allowance on the power required as found from the diagram.

With regard to the author's question of the power required by the Pachuca system (Reference No. 8), I find on looking up this article that there is simply the bald statement that "pulp containing 100 tons of dry slime to 150 of solution can be treated with 1$\frac{1}{2}$ h.p."

Figuring out the probable power consumption for the air lift agitation in the Luipaard's Vlei plant as quoted by the author, say, 85 cub. ft. of free air (presumably per minute) at between 25-30 lb. sq. in. gauge, we have from the diagram 8·3 to to 9·3 h.p., probably 9-10 h.p. in actual practice.

The above discussion proves that either the horse-powers as quoted by the author are wrong, or that the quantities of air as given are not correct. From my own experience in this matter I would judge that the quantities of air are the more correct.

In discussing the author's remarks anent mechanical stirrers, such as "clumsiest constructions," the necessity of "digging out the paddles after a short stoppage," it might be pointed out that mechanical devices need not be clumsy, and paddles need not be dug out if the machinery be designed properly.

Mr. Knitfen, in "Methods of Pulp Agitation" (author's reference No. 8), says that "if the ore only requires that fresh molecules of cyanide be brought continually to the particles of metal, and that the air which is included in the solution be sufficient for the reaction, there is no system at present better than arm agitators, raised a few feet above the bottom of the tank, and driven at 800 linear feet per minute." Presumably he means 800 ft. min. at the outer ends of the arms.

In these remarks I am not advocating mechanical stirring gear as against air agitation, but submit that in the present state of knowledge one may preserve an open mind on these questions, since there is no doubt but that mechanical devices and air agitation have each their sphere of usefulness in metallurgical processes. The author tries to make his case for air agitation so strong that he seems to have closed his eyes to any advantages of mechanical gear. While I do not wish to pose as an advocate for the other side only, I maintain that there is reasonable doubt as to the accuracy of the power figures quoted for air agitation.

Simple circulation or stirring of pulp could be carried out economically as regards power and wear by means of a centrifugal pump specially constructed and used for that purpose only. As an example, it would be possible to turn over the whole of the pulp in the Luipaard's Vlei vat every 50 minutes at a cost of no more than 6 to 8 h.p.

The trouble in the past with most centrifugal pump circulation and agitation has been that the pump has been used for other purposes beside circulation, as, for instance, in transferring charges against fairly large heads. The pump was necessarily speeded up to deal with the maximum head, and is therefore very wasteful of power when circulating. The wear and tear on the pump also increases at a more rapid rate than the increase in speed. The over-all efficiency of a low lift circulating pump, motor and piping system should be about 50%.

I am rather of opinion, however, that the whole question as regards air agitation vs. mechanical agitation will turn more on maintenance and capital cost than on power cost.

With regard to the general results found by F. C. Brown (4th paragraph), we read :—

1. "The higher the tank compared with the diameter the less the power required."

This statement is incomplete, the probable meaning being that this is true only for equal weights of pulp handled. The statement then falls into line with theoretical considerations. For if the height be doubled the same volume of free air is still only required for agitation at slightly less than double the gauge pressure. This means that the absolute pressure is not doubled, and as the work required does not increase at the same rate as increase in pressure, but only as the logarithm of the pressure ratios, considerably less power per ton of pulp treated is required.

In conclusion (2), regarding less cyanide consumption with deep than shallow vats, this result may also be expected to follow partly from the argument as in (1), but also because in a deeper vat, owing to the higher pressure of the air, chemical action between the air, the cyanide, and the pulp is more rapid, and the total time of reaction is thus also reduced. On the law that the intensity of chemical reaction is proportional to the mass density of reagents present, it follows that doubling air pressure doubles its density, and the total time is cut in half. Less total air per weight of cyanide present will be required, since only a very small proportion of the total oxygen supplied is really used up by the cyanide, and so again there will be less loss of cyanide by oil, etc., which is carried by the air supply.

There is, however a very practical point to be considered in fixing the total height of a vat, that is the expense of filling the vat by pumping, because, theoretically even, only half the work done in pumping the vat full can be regained in emptying it, and in practice generally the whole of this power is lost. There is also to be considered the extra cost of building very tall vats strong enough and furnishing suitable foundations to withstand the heavy strains, wind pressure, etc.

It may be mentioned here that a good practical rule of thumb in regard to pressure of air required is to take the figure representing the total depth of the vat in feet and divide this by two, which will give the lb. sq. in. gauge pressure required. For example, overall depth 40 ft. requires gauge pressure 20 lb. per sq. in. If air be used at a pressure much greater than that so calculated there is bound to be a serious waste of air, and probably inefficient circulation.

The maximum amount of cooling of the pulp charge due to the expansion of the air used can be readily calculated. If air be allowed to expand while doing work and no heat is added or abstracted during the process the ratios of the absolute initial and final temperatures is given by the expression.

$$\frac{T_a}{T_1} = \left(\frac{P_1}{P_a}\right)^{\frac{2}{7}} \qquad (5)$$

T_a is the initial absolute temperature (°F + 461) and T_1 the final temperature; P_a and P_1 being the pressures as used in the formulæ for horsepower as given previously. The amount of heat abstracted in British thermal units would be for every pound weight of air used.

$$\text{BTU} = 0{\cdot}24\ (T_0 - T_1)$$

the figure for the specific heat of air being 0·24. The specific heat of water is 1, and of rock generally 0·2. Assuming that the initial air temperature to be the same as that of the pulp charge and knowing the quantity of free air per minute used, the calculation becomes simple. For the Luipardsvlei installation quoted by the author, we have the following data :—

$P_a = 12{\cdot}1$, P_1 average $27{\cdot}5 + 12{\cdot}1 = 39{\cdot}6$, T_a say 70°F = 531°abs..

Quantity of air per minute 85cf = say 5·3 lb., charge of slime 150 tons, water 300 tons.

BTU required to heat pulp 1°F = 660,000.

BTU taken by expansion of air per hour, per °F = 76.

Theoretical difference in temperature Ta—T_1 = 152°.

B.T.U. taken by theoretical expansion of air per hour, equivalent to heat abstracted per hour 76 × 152 = 11552.

Cooling effect on pulp per hour 0·0175°F.

It will thus be seen that this effect of cooling is absolutely negligible, and could not in practice be even measured after 10 hours of circulation. I have mentioned this point because although Mr. Allen has not touched upon it, other writers on the subject seem to have made too much of it without going into the matter thoroughly.

In this connection I might suggest that if warming up of the pulp be necessary to hasten chemical reaction and solution of gold by cyanide it would be advisable to introduce a jet of steam with the air, as the heat would then be applied in the initial stage of contact of air, cyanide, and pulp, and should thus be most effective in shortening the time of treatment.

The destruction of cyanide by carbon dioxide which the author mentions, should also be inappreciable for two reasons. First, the amount of CO_2 added by normal air is very small. Allowing 0·06% CO_2 by weight as usual in air, 100 cubic ft. per minute supplied amounts to only 0·22 lb. per hour, which would represent 0·32 lb. of potassium cyanide; Second, different authorities say that CO_2 is not a cyanicide in the presence of alkali in solution. In Rand practice, alkaline solutions are always used, quick-lime being added for the purpose.

With reference to the air agitation vats at the Geldenhuis Deep, referred to by the author, these were designed three years ago. The central pipe was made in telescopic form, so that experiments could be made on the best length of pipe to work with. The plant was finally worked with the upper telescopic pipe as far down as it would go.

Regarding the author's statement as to the Luipaard's Vlei agitation vat being probably the largest single charge vat in the world, one might be permitted to state that the probability has vanished. There are two air agitation vats installed at the City Deep, Ltd., designed in 1909. Each vat is 32 ft. in diameter, 30 ft. height of cylinder, with 8 ft. depth of cone or 38 ft. high overall, each vat treating from 250 to 300 tons of dry slime at a charge. The central pipes, 16 in. diameter, extended about two thirds of the way up. Air for both vats is supplied by a small compressor belted to a 25 H.P. motor so as to have spare power for emergencies. An emergency air supply pipe is connected to the regular mine air service pipe.

The author has suggested that the question of turbo blowers in series to supply air might be looked into, so as to avoid atomized oil in the air. In the present state of engineering science, small blowers would not be nearly so economical as reciprocating compressors. Oil troubles can easily be avoided by proper attention to the lubrication of the compressor cylinders. Soapy water can be used at intervals for lubrication instead of oil, and the oil can be thrown down by a water spray in a suitable receiver, and the combined oil and water emulsion drained off by a suitable trap before the air is delivered to the vats.

The advantages of the shorter central pipe, which advantages are treated of in an article by A. J. Jager (Mr. Allen's reference No. 12) Oct., 1910, were realized in the City Deep design, more especially the two main points, that it is possible with the shorter pipe to commence operations before the vat is full, and that aeration is more thorough when a thick body of pulp lies over the top of the pipe. In fact, at the City Deep the air can be advantageously turned on when the vat is only one-third full. The air bubbles in the central pipe are comparattively large as they rise, but when they strike the more solid body of pulp over the mouth of the pipe the air is broken up and intimately incorperated with the pulp. With the pipe debouching at or near the surface of the pulp there is a violent disturbance in the centre and comparative quiet round the outer circles, but with the pipe opening deeper down, the whole pulp looks whipped and light, and the general appearance of the surface is like a mushroom-top, with continuous motion of the pulp from the centre to the sides.

Since air agitation is still a young process on the Rand, doubtless it will develop on lines peculiar to our local requirements, and Mr. Allen deserves the thanks of the Society and local mining circles generally for bringing this matter up for discussion at the present time.

The meeting then closed.

Annual Dinner of the Society.

The Annual Dinner of the Society was held at the Grand National Hotel on Saturday, April 29th, the President (Dr. Moir) in the chair. There were also present:

Messrs. W. Abel, A. Aiken (Hon. Auditor), Geo. Albu, R. Allen, L. Aubourg, E. L. Bateman, R. G. Bevington (Past President), G. Blenchard, J. Frank Brown (Member of Executive Committee of Provincial Council), R. A. K. Bruce, E. W. Buxton, W. A. Caldecott (Past-President), W. Calder, F. W. Cindel, L. Colquhoun, J. Cowie, W. M. Coulter, A. F. Crosse (Past-President), W. H. Dawe, W. R. Dowling (Vice-President), W. R. Eales, M G. Elkan, *Evening Chronicle*, J. W. Forster, W. Frecheville (Past-President, Institution of Mining and Metallurgy), K. L. Graham (Member of Council), J. Gray, T. J. Greenwood, B. C. Gullachsen, R. Hamilton (President, Johannesburg Chamber of Commerce), H. L. Harland, Hodgson, J. Irvine, E. G. Izod, E. H. Johnson (Past-President), Tom Johnson (Member of Council), A. McA. Johnston (Past-President), J. A. Jones, R. N. Kotzé, C. Lamont, E. J. Laschinger (Member of Council), G. A. Lawson, H. Lea, J. Lea, Q. J. Leitch, J. Littlejohn (Hon. Treasurer), Dr. D. Macaulay, M.L.A., A. MacDonald (President, Pharmaceutical Society of the Transvaal), H. S. Macgregor, J. McCracken, J. N. Meeser, Dr. E. T. Mellor (President, Geological Society of South Africa), G. Melvill, H. Meyer, P. T. B. Morrisby, S. J. Murphy, Dr. Temple Mursell (President, Transvaal Medical Society), S. Newton, P. N. Nissen, H. G. Nitch, E. A. Osterloh, E. Pam, S. H. Pearce (Past-President), C. Peterson, R. G. Campbell Pitt (President Rand Pioneers), R. A. Porter, H. Potter, J. F. Pyles, Frank Raleigh, *Rand Daily Mail*, J. H. Rider (President, South African Institute of Electrical Engineers), W. H. Roe, Fred. Rowland (Secretary), A. Salkinson, P. Scatterty, Courtenay Shaw, G. M. Smythe, H. Stadler, *Star*, A. N. Stockett (President, Association of Mine Managers), A. N. Stuart, W. A. Tester, J. E. Thomas (Member of Council), Professor G. R. Thompson (Principal, S.A. School of Mines), Dr. Ritchie Thomson, W. E Thorpe, C. Toombs, M. Torrente, *Transvaal Leader*, S. T. Tregaskis, Hon. Senator W. K. Tucker, C.M.G., J. A. Vaughan (President, South African Institute of Engineers), J. F. Walker, John Watson, F. Wartenweiler, H. O'K. Webber (President, Transvaal Chamber of Mines), F. Wells, W. S. Whaley, H. A. White (Member of Council), J. Whitehouse, Professor J. Wilkinson (Member of Council) and T. Wood.

At the conclusion of the repast, the President gave the toasts of "The King" and "The Governor-General," which were received with musical honours.

Thereafter, the President announced that apologies for inability to attend the dinner had been received from the Governor-General, the Administrator, Lord de Villiers, Rear-Admiral Bush, Lord Methuen, Generals Botha, Smuts and Hertzog, the Hons. J. W. Sauer, A. Fischer, H. C. Hull, F. S. Malan, H. Burton, Dr. O'G. Gubbins, Col. Leuchars and Sir David Graaff, (Members of the Cabinet), Sir George Farrar, Bt., M.L.A., Sir Percy Fitzpatrick, K.C.M.G., M.L.A , Messrs. F. D. P. Chaplin M.L.A., P. Duncan, C.M.G., M.L.A., Lionel Phillips, M.L.A., J. G. Hamilton, M.V.O., D. Christopherson, W. Dalrymple, the Mayor of Johannesburg, S. S. Hough (President, Royal Society) and Dr. P. D. Hahn (President, South African Association for Advancement of Science).

The Society.

The Hon. Senator W. K. Tucker, C.M.G. : The toast I have the honour to present to you to-night is that of the Chemical Metallurgical and Mining Society of South Africa. I feel that it is a great pleasure as well as an honour to be asked to propose this toast on the Witwatersrand. I say that because I have been 'in another place' lately, and there I find that small credit is given to Johannesburg for having any intelligence worth speaking about, or that at all events, if it has any intelligence at all, it is of a kind of which we have to speak with bated breath. But those of us who have been a long time in South Africa and a long time on the Witwatersrand and who know what the Rand owes to the gentlemen who form the members of this Society and what South Africa as a whole owes to these gentlemen—such of us, I say, cannot help feeling proud that we have such a body of men on the Witwatersrand as those who form the members of this Society. Now in presenting a toast of this kind one is expected to say something complilimentary—(laughter). Well, it would be extraordinary if one consented to propose a toast which one did not feel one could honestly give in complimentary terms ; and from the fact that I have consented to propose this toast to-night I hope you will believe that whatever remarks I have to offer in commendation of this Society are honestly meant—(Hear, hear). Now those of us who know the condition of the Rand in the early days and the difficulties which were met with in every direction, will realise that the difficulties in the extraction of gold from its ore were such as to leave little if any profit from a large section of our gold-bearing reef, and I think one may fairly say that it is to the energy of these gentlemen our friends of the Chemical Metallurgical and Mining Society that a very large portion of the credit is due for having made it possible to make our low-grade ore pay the handsome profits which lately they have been able to do. One does not forget that there are other Associations of technical men to whom very large credit is also due. I speak more particularly of the South African Institute of Engineers. It is to the indefatigable efforts of the members of these Associations, to their frequent meetings at which defects have been stated and remedies proposed and discussions take place, that a solution of many of the difficulties I have referred to is due. A large proportion of the success attained is no doubt due to the Society which is met here to-night. These are common-places no doubt to men resident on the Rand; they are matters we all know, but we perhaps do not often remember the debt South Africa owes to the efforts of the men who have so thoroughly put their best thought and much time into work of this description. Not only has your Society been concerned in trying to find more economical means of extracting gold from the ore, but it has spent many evenings in discusssing questions which bear on the relief of distressed humanity. I mean that you have spent a large portion of your time in trying to ascertain the causes and nature of dynamite fumes, the effects of these fumes and the remedies to be applied, how to make the mines healthful and in what way to combat the terrible diseases arising from inhaling such fumes. In addition to that you have paid considerable time and attention to what is called silicosis or miner's phthisis. This has also been the subject of debate in "another place." We never talk about Parliament, you know: it is always 'in another place'—(laughter)—and you would be astonished at the number of experts there are there. They are cocksure too, there is no mistake about it. In a scientific Society such as yours I am quite sure you approach a subject such as miner's phthisis knowing there are many things to be considered before you can be positive, but in "another place" you need not trouble very much about that, you can be as dogmatic as you like. My friend Dr. Macaulay will be able to tell you more about that than I can. The point I want to make in referring to this matter is that there is some good on the Witwatersrand, that there are some men who are pursuing good objects, and that there are such institutions as yours and other Associations who are trying to make not only profits better, but to make life itself better and more enjoyable to those participating in the production of gold. For this you get no credit. You have only to go a little way from Johannesburg to find that out. The friends of Johannesburg outside the radius of the Witwatersrand are very few indeed. The other people look upon the Rand as a place where illicit liquor abounds, where nearly every place which is not a canteen or gambling den, is a brothel; where the men are rushing in the hot pursuit of wealth, using means which are not the most decent. They say that some of them may become millionaires and if they do they should be ashamed of themselves; that those who have not succeeded in becoming millionaires are doing their level best to become such, and are not scrupulous as to the means by which they hope to attain their ends. You are told that you are demoralising the innocent young man from the native territories, that you bring him here and half kill him in your mines and then you try to induce him to spend his hard-earned savings in canteens or in the aforesaid brothels. That is the picture (which I assure you I have not overdrawn) of the Rand that is in the minds of many in the political world of South Africa.

It is nice to be able to come back to the Rand and reassure oneself that all these things are not true. One can only be sorry for the introductions which the gentlemen I allude to must have had when they visited the Rand last—(laughter)—and hope that when they come again they will see that their introductions are of a better class—(hear, hear). When you talk of problems surrounding such a subject as miner's phthisis considered as an industrial disease, and what it leads to in the way of compensation—you know the matter has been debated in Parliament—the questions arise: who should compensate the men, how they are to be compensated and whether the men should contribute, and whether the State should contribute—I hope you will agree with the last stage which was arrived at by the Government, namely that Parliament should set aside a sum of money for compensating hard cases which may occur during the next year, and that they should take time to fairly consider the problems which are involved in this great question of miner's phthisis. I think this was a wise step for the Government to take. I think it is generally held by all right-thinking men (whether millionaires or not) that there is a good deal to be said for making provision for persons who fall by the way in connection with any particular industry; but it is idle to think of applying certain principles on the Rand, which it is not intended to apply anywhere outside the Rand. We have just been passing large sums of money for the eradication of East Coast fever. Now if miner's phthisis is an industrial disease peculiar to mining, I should like to know whether East Coast fever is not an industrial disease peculiar to cattle raising, and whether scab is not an industrial disease peculiar to sheep-farming; and I ask if it is sound to force an industry in which miner's phthisis is produced to be taxed on that account, whether it is not just as defensible to tax the man who owns the cattle industry to protect him from East Coast fever, or to tax the man who runs sheep in order to enable him to escape from the ravages of scab (hear, hear). But these matters are not looked at in that light down South. As long as a question is applicable to the Witwatersrand, many politicians are perfectly willing that the principle of compensation should be applied, but they are careful that this principle shall not escape outside the ring-fence round the Rand. It makes anyone who has got to deal with these matters realise that during the time between this session and the next session of Parliament the Government will have much cause for thought, and I hope the question of compensation for industrial diseases will give other people occasion for thought also. We want to do our best for the country ; we wish to help the Government and to give our best assistance if they are willing to accept it. Such institutions as yours have already devoted considerable time and attention to it ; your energies have been given to the causes of the disease and to the remedies which you think may stop it. I hope that, as you have been successful in the past in solving many difficult problems, it will not be beyond your power to find the solution of this one. You may also possibly go a little further and consider the economic part of it. Is it right that only the industry should pay? Should the person liable to the disease also pay and should the Government also contribute? If it is right that the Government should find the money to combat East Coast fever, is it advisable that the Government should contribute towards a fund in connection with miner's phthisis? One does not wish to wander into many fields of political controversy, and I am sure our Chairman has something interesting to say to us, so I ask you to drink success to the Chemical Metallurgical and Mining Society of South Africa — (Applause.)

The President responding, said : I feel it a most distinguished honour indeed to rise and respond to this toast—the toast of Ourselves—so wittily and so kindly proposed by our esteemed fellow-technical and friend Mr. Tucker, on whom the Senatorial dignity sits so well. I fear we have not deserved it. He has given a most kindly presentation of our virtues, such as they are. I thank him very sincerely on your behalf therefore, but when it comes in earnest to replying, I confess I find myself in great difficulty. You know I used to be called a person prejudiced against the mining industry, sometimes quite an enemy of the mining industry, yet here I am wedged in between its leaders, and, gentlemen, the heads of the mining industry are in this case men of such unassuming modesty that really I can only imagine myself to be a lion in a den of Daniels, instead of a Daniel in a den of lions as I expected! However, this is a social occasion and there will be no gobbling up this time even if ever there were any tendency thereto.

The main excuse for these dinners is that they demonstrate that Science is not altogether inhuman and that the votaries of Science are still capable of enjoying these big feasts—although I fancy that dinners of this kind are a relic of prehistoric man and the slain mammoth, and that after-dinner speaking came in as a consequence of Noah's little adventure with the vine : I mean that speech-making was originally merely a physiological effect of diffusible stimulants on a big meal !

Speaking seriously, however, it is very good for the Society to have these feasts, if only to get away for once from technical details and to hear the wider view which we get at these affairs ; and it is especially nice when the serious business of the evening is preceded by our annual pat-on-the-back. If we did not get this encouragement, we should be in danger of losing that 'gude conceit o' oorsels,' which every normal human being—and corporation too—possesses. I see that my distinguished predecessor, Mr. McArthur Johnston, was cynical on this point last year : he said when told that this is the best Society in South Africa that he was beginning to believe it himself. What I would like to say myself is this, that although we have our faults, yet for the present state of South Africa we *are* a good Society and really fulfil a most useful purpose in the State. Our motto, to parody a certain Latin one, should be that we consider nothing technical alien from us. We are a sort of combination of an Institute of Metallurgy, a Debating Society, a Chemical Society, a Surveying Institute and a Mining Journal, to say nothing of our touching on amateur medicine nor of our occasional emulation of the lighter side of the weekly Press—(Laughter). Later on we shall probably get specialised, but in the meantime we are as democratic and almost as all-embracing as Parliament—with our 'push-ons' and 'pas-ops' too. You know we do not admit politics at our ordinary meetings—which is really rather surprising seeing what we *do* admit. But the fact is that politics is ingrained in the human spirit, so we have our fling at these dinners. Perhaps it is because we can only hear, and not discuss, on these occasions, that we like it ; but a better reason is that the gentlemen who come here to talk to us on politics and economics are experts in these arts, and as you see already, they give us something very good indeed, something better than the second-hand article of the local Press—(Laughter).

But to return to the Society itself, I am sorry that I was not here at its foundation. I only came in when it had—so to speak—found itself, so that I can only dimly guess at its vigorous infancy, when its controversies seem to have been conducted with chunks of old red sandstone. Now we are nearly grown up. We have reached the bashful age of 18 and yet we are not bashful, as you see. We have a wonderful supply of papers every meeting and excellent attendances to hear them. Our excursions are very successful and we have had a good many this year, and you can see how highly the generosity of those who invite us is appreciated, by the large numbers who turn up at them: in fact, I can say from Home experience that the Rand easily beats any Home Society in respect of excursions except of course the annual picnic of the British Association. This year also we have twice broken the record as regards the size of our monthly *Journal*, a journal which I may say is highly appreciated by our 1,400 members stationed all over the world.

We meet it is true with a certain amount of criticism, which perhaps is just as well in view of the Scriptural saying about all men speaking well of you. This criticism is mainly directed at the *quality* of the matter provided; but I think these critics do not quite realise that South Africa is not yet a highly specialised country. We like to believe that the Society sets a standard of its own—which is quite in accordance with the South African spirit of independence. I am far from asserting that all South African standards are desirable ones, but I *do* say that so long as our Society gives an opportunity to everything of novelty and originality—in fact, lives up to its motto—we have nothing to fear from criticism.

There is however a fly in the ointment of self-gratulation. We have a very high proportion of silent members. I do not think I exaggerate when I say that our proceedings are carried on by less than 50 out of a membership of 1,400. It is true that these gentlemen have the 'celestial fire of enthusiasm' and amply make up for the deficiencies of the rest, fully maintaining the Society's reputation as the most free-spoken in the world. But that is not everything. It is particularly regrettable that the number of mine managers who grace our proceedings is probably less than half-a-dozen. Why this apathy? Surely the life of the industry is its technical side! At least I hope and believe that the day has gone by when (as one of our critics once said) the mining industry was "merely a side show to a gambling hell." Why then this apathy? We should have been pleased to have the mine managers come forward, even if only to open their hearts to us on the technical side of the Mines Bills recently in Parliament. We could have got up a most excellent controversy on both of these Bills, a controversy which might have done some good outside, because presumably, saving the Parliamentarians who belong to our Society, *we* know more than the Parliamentarians do about these things, and also *we* do not use the closure either.

But when I have said that, I have said all. I have no sympathy whatever with the policy of drift, inertia, "all's right with the world"; nor have I any sympathy with mining on "on the stoep so long" principles. Let us open up the *arcana* of mining for discussion and improvement will surely follow. What is more, if we don't do it, there will be no Rand left for us to discuss in a comparatively short space of time, as Mr. C. O. Schmitt's able paper last week showed us. Although Mr. Schmitt's paper dealt only with metallurgy, there is a big field in the mining work proper for the economies which are to save the Rand. It may seem absurd to talk of saving the Rand at a time when, as at present, it is making a sovereign every second, but we must look to the future. The metallurgical side of gold-winning is much more highly developed than the underground work—witness the wonderful modern plants we have visited this year. I hold the opinion, and I do not think I am alone in it, that no one would tolerate on the surface-works nowadays the lack of supervision, the primitive methods of transport and the incompetent labour, which have to be put up with underground at present. We want *more light* perhaps, physical as well as metaphorical. Secondly, why do we use more explosives per ton than any other country? There has been no improvement in this respect since the war. I might go on with these Why's: there are plenty of controversial subjects in mining. There have of course been plenty of improvements, but there has been plenty of *laissez-faire*, and in this connexion there is one thing that I cannot resist mentioning, namely that I hope that in less than five years we shall no longer hear at these feasts "that blessed word Mesopotamia"—the dearth of native labour. I feel convinced that with the mechanical improvements and the ventilation schemes now in hand and the prevention of accidents this difficult question will solve itself in that time. There are certainly men who have got the impression—and this impression also exists in the kraals—that the native is being exploited and is expected (*Hibernico more*) to "come oot and get killt,"—and it will take some years to remove that impression. You will remember, gentlemen, that our accident death-rate is one of the highest in the world, at all events amongst nations that keep any record of such things. Our attitude in the past towards this question of native labour has been much the same as that of the Backveld towards pests, namely praying instead of spraying—(laughter)—and if I may pursue this vein of elephantine humour, I say the industry should adopt scientific methods, and *attract* the native instead of *contracting* for him. I expect the mining men will say it is 'like my cheek' to talk like this, but if they had not sulked in their tents, it would not have been necessary. I am really putting my views forward with due humility since, as Ben Jowett said "we are none of us infallible, not even the youngest of us." We know that the managers have plenty of worries already with poorness of grade, etc. : and they will find a good deal of sympathy too with their complaint that they are being over regulated and grandmothered now-a-days : but things on the Rand to-day are very different from the easy job they used to be. When I read about the early days, when a proportion of the technical men seem to have been recruited from the ranks of the bricklayers and the drapers, I confess I am surprised at the good results which were undoubtedly achieved, and although I should be the last to say anything against natural abilities, in this case I am irresistibly reminded of the great Dr. Johnson's remark about the suffragettes of his day. I forget what the lady had been trying to do, but he said it reminded him of a dog walking on its hind legs, "it is not well done, but you are surprised it is done at all." The point is, gentlemen, that the Rand cannot stand the strain of rule-of-thumb nowadays. It may have been the case long ago that an ounce of practice was worth a ton of theory, but at the present time practice does not go far without theory.

Anyway, gentlemen, it is due to this Society and the Engineers that the Technics of the Rand are now so advanced as they are. They are so advanced that metallurgy proper is disappearing in favour of engineering, and (if I may be allowed one more little gibe) I wonder when our consulting metallurgists will take the hint and call themselves chemical engineers and metallurgical engineers as they do in other countries.

I want to refer to one thing more on this public occasion, namely the Seymour Library. I consider this library is one of the greatest assets of the Rand and has done inestimable good both to the reading public and to the scientific reputation of this country. In some branches of science it is nearly as good as the great library in South Kensington, and speaking for myself, I can say that more than half of any researches I have been able to do would have been impossible but for this library. The library is still incomplete although over £3,000 has been spent upon it, but the main difficulty is that its annual income is not sufficient to keep it up to date in periodicals, to say nothing of new books, of which there is a list of about £200 worth urgently required. This Society took the initiative and has been successful in obtaining some fresh support for the library from the Chamber of Mines and the Council of Education and the various Technical Societies, which have saved the library from its immediate difficulties ; but if it is to be kept up to date I think that, in addition to these generous donations, we might get a small sum such as £25 a year from the richer of the mining corporations. I do not think they would feel it very much, and four or five such donations would put the matter right. I assure them that no money could be laid out to better advantage ; if the Rand is to survive another 50 or 100 years, it can only be by means of scientific advances and discoveries, and these, I may say, are more likely to originate from library study than from either native or imported genius. The library also has earned the right to be considered by the founders of our Art Gallery, who displaced and disorganised it for five months, but as this is under consideration, I need only say the 'word in season.'

Gentlemen, again I thank you for drinking our healths so heartily, and you, Senator Tucker, for suggesting the idea.

"The Mining Industry."

Dr. D. Macaulay, M.L.A.: I rise with a feeling of a distinct grievance. On arriving late when I got hold of the programme I found one of my pet quotations plagiarised. A second grievance that I have is that my friend Senator Tucker has already traversed the ground which really belongs to me. It is really a great honour to propose the toast of the Mining Industry and to ask you to drink its health. I think it is a somewhat unnecessary toast since, so far as indications go—to a layman at any rate—it never was in a better state of health than it is at the present moment, and while this Society owes its existence to the mining industry of the Witwatersrand, I think as a member of the Society I may also claim that the present healthy state of the industry owes some proportion of its vitality to the efforts of the gentlemen who compose this Society—(Hear, hear)—and it is a matter of very great regret to those of us who have been unfortunate enough—I use the word advisedly—to indulge in the luxuries of public life, to find your efforts and the efforts of the gentlemen who control this industry are so badly understood elsewhere. I would like to corroborate every word Senator Tucker has said in regard to the suspicions with which we are regarded in another place. I do not know whether "suspicions" is the right word, or whether jealousy is the right word or envy is the right word. I think I would leave it to your President's gift for phrasemaking to invent the proper word for the feelings with which we are regarded. Now I would like to impress upon you that it is our duty to dispel that feeling. Johannesburg has been described as a University of Crime. I am not very sure whether in that University you or I must consider ourselves Students or Professors, or whether we are simply lookers-on allowing our pockets to be picked by the robbers from the Stock Exchange. Anyhow we have been deliberately described as a University of Crime, and Johannesburg has been described by a leading Parliamentarian as a gigantic brothel, and I find in the Press last week our white women are described as no better than they ought to be. I really think that the time has come when if we are going to have peace in South Africa we should put an end to these calumnies directed against us. Now, Mr. President, your Society is a non-political one. In politics you are neither acid nor alkaline, but most strictly neutral—(Laughter)—but one cannot help introducing politics on an occasion like this, and I personally want to vindicate the people with whom I have been associated for the last fourteen years. One of our most delightful politicians who is partly a politician and partly a *savant*, Mr. Merriman, has, I can assure you, never got up in his place in Parliament when he has cause to refer to the mining industry without having a sneer and a jeer at the mining industry. I do think his parliamentary life ought to have taught him to view things with a little more judgment and a little more coolness. This is really an important subject. He is the spokesman of a certain section of politicians who do not belong to one particular party. I want to make that quite clear. He was the spokesman of this section of political opinion and he got up to sneer and jeer in righteous indignation about the large mortality in connection with the mines of the Witwatersrand. Now as the result of the active efforts of all those who are intimately concerned in the matter, the mortality of the mines of the Witwatersrand has been brought to within reasonable limits, somewhere about 30 or 32 per 1,000 per annum. Now rummaging through some of the archives of the Cape Parliament I discovered an opportunity of rubbing up my Right Hon. friend Mr. Merriman. I found that the mortality from chest diseases alone among the native population in Mr. Merriman's district of Victoria West amounted to 40 per 1,000 per annum. I pointed this out to my friend across the floor of the House and he was very angry and he has so little recovered from his anger that up to the present moment he has not condescended to reply—(Laughter)—because I suggested to him that before he started to find motes in the eyes of the people here, he should remove the beams and king posts from those in his own constituency. I only want to show the spirit actuating a certain section of the community towards us and to point out to you that it is the duty of every one of us to do what we can to dispel this suspicion, envy or jealousy, or whatever my friends may choose to designate it. To show you the amount of undeserved suspicion which exists, when the Minister for the Interior made an announcement which gladdened my heart, namely that the Mining Industry—it was called by him the W.N.L.A.—had come forward and offered to build here in Johannesburg a Medical Research Laboratory in order to investigate diseases peculiar to humans—animals and plants are well provided for—even that was regarded with suspicion by a certain section in another place—(Laughter). It was called "blood-money." It was called all kind of things. Now, gentlemen, a certain number of us have been advocating this for years, and I assure you gentlemen that if the Devil himself had come along with this money to build a Medical Research Laboratory, we would have had the greatest pleasure in taking it. I hope and expect from this institution the greatest benefits not only to this industry, but to the whole of South Africa and the whole of the world. We have most unique opportunities here for studying human diseases which have been steadily neglected, both by the much abused millionaires and by the elected representatives of the people. We shall have in South Africa very soon, owing to the liberality of the Mining Industry, and I say that I have never known a case where these men have not come up to the scratch —we are going to have an institution which will be second to none in the world. Incidentally, I have not the slightest doubt, the mining industry will considerably benefit—and good luck to them too if they do—and also in conjunction with a similar institution in the north of Africa at Khartoum we are going to be enabled to have an institution which will probably make this country certainly one of the best countries in the world to live in, as many of us think it is at the present moment.

I could point out what the output has been and what we hope it is going to be: what is being done with regard to artificial ventilation on the mines and being undertaken, too, spontaneously by the authorities and under no pressure from the administration: what is being done to induce the workers on the mines to take care of themselves and also that the industry is paying its legitimate share towards the maintenance of those who fall by the way as Mr. Tucker has said. Many other things I could point out to you, I will content myself with asking you to drink to the increased prosperity of the Mining Industry of the Witwatersrand, which means also the increased prosperity of the whole of South Africa.—(Applause.)

Mr. H. O'K. Webber, responding, said: The toast of the Mining Industry has been responded to year in and year out ever since Johannesburg has been a city, and all through, many of those who sit in this room to-night have given all their energies to building up the greatest gold mining industry in the world. First came the miner, then the mechanic and lastly the chemist and metallurgist. The brilliance of America showed its light upon this country later on, and later still we had a glimmer from the light of Asia—(Laughter). But it is to the metallurgist and chemist that this industry owes its real progress. They have shown us how to extract the gold to the last decimal point of a dwt., and what is of much more importance they are showing us how to extract the gold at the lowest possible cost. The time may come when they will perhaps take over the ore from the dump and later on perhaps they will start operations at the face of the rock underground, but we have to-day among the members of this Society the finest talent in the world for the treatment of these gold problems, and although it is fortunate that the treatment of Rand ore does not contain very many difficult chemical problems, yet if it was so I feel sure the gentlemen in this room would be able to deal with any difficult problem that might come before them. Gentlemen, this industry has always been before the eyes of the world, and you have heard to-night that it is also very much before the eyes of our friends the legislators of the Cape. The pastoral portion of South Africa you have heard to-night looks upon us with a good deal of suspicion. We have been called all sorts of names. You have heard to-night that this place has even been called a school of crime, but the gentleman who called it by this name I think really means well. He is really a kind old gentleman—(Laughter). He is the son of a Bishop too—(Laughter)—and I believe if he was here to-night sitting and enjoying our dinner, I believe he would be discoursing to us upon the subject of the possible evils which may attend the life of those who carry on the comparatively guileless occupation of agriculture—(Laughter). We have also been called millionaires. I wish we were. I think perhaps it is this very Society which attracts the envy of our friends, perhaps it is because you call it metallurgical: I expect it must sound very rich in their ears—(Laughter). They also wonder where our gold goes to. Well we all wonder where our gold goes to. But this misunderstanding will wear off. If they had paid us a visit last week they would probably have found that the only idea in our heads was how to grow mealies, cattle, sheep and horses. The times are progressing. At the present time one hears on the one hand people connected with the mining industry discussing in heated argument the subject of scab in sheep, and on the other we hear of farmers discussing the latest mining regulation brought out by my friend Mr. Kotzé: but we must get it into their heads that we have a great sense of our duty towards those who earn their daily bread in the more unhealthy occupations of this industry, be they black or white. The mining industry as you know has been legislated for and they will continue to legislate for us, but many of the laws passed at the last session and regulations that have been promulgated or will be promulgated for the mining industry, have in nearly every case been practised of their own free will by the mining industry for the amelioration of the conditions of mining. But we have lawyers in Parliament, and these lawyers must legislate. They have friends who are not in Parliament and they must bring grist to their mill. Some day when the latter get into Parliament they will return the compliment.

Gentlemen, this mining industry is a huge factory for turning out an article for sale just as any other industry is, with the one exception that the article we produce has only one fixed price in the market. Therefore the question of producing gold at the lowest possible cost and the opening up of new areas containing large quantities of low grade ore holds South Africa in its hand. It spells either constant and continuous prosperity or certain mediocrity for the country we all love and live to build up.—(Applause.)

Mr. J. Frank Brown, M.P.C., in presenting the toast of "South Africa," said: One or two things occur to me to-night. I should like to say that in future the man to be called upon to propose the toast of South Africa is the man who has been in the country about six weeks—(Laughter)—because we all know that when those who visit this country have been here six weeks they know more about our problems than ourselves. I know that when I arrived here about eleven years ago and had been in the country about two months I felt I knew more than I do to-day. Apropos of the remarks of previous speakers, I heard the other day a prominent Dutchman in the Lower House, a member of the Nationalist party, ask a puzzling question. Why are there three parties? The man said "I do not know that there are three parties, who are they?" The other replied "Well, there are the

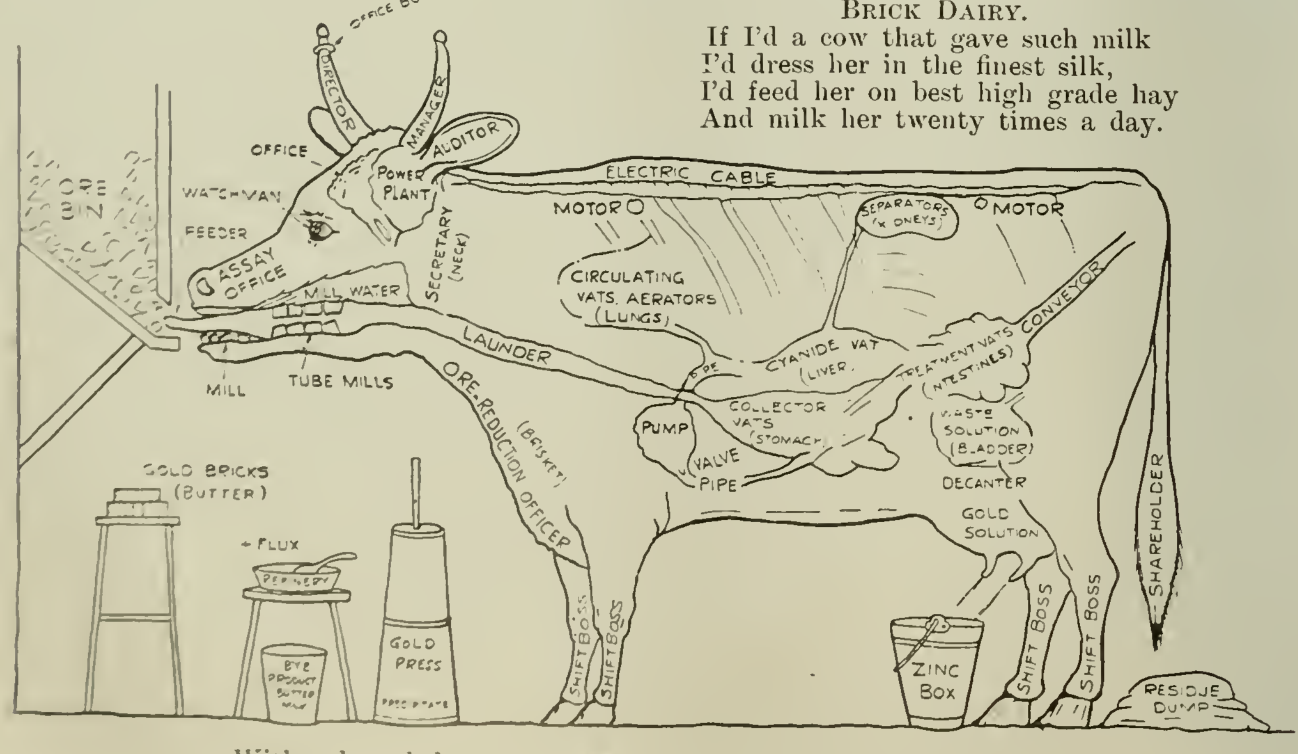
Unionist members who are progressives and the others reactionaries, there are the Nationalist members who are progressives and the others reactionaries, and the Cape Peninsula." It seemed to me that this very readily explained why in a certain place we here on the Rand are looked upon as such scoundrels. I ask you to look upon two pictures, one of ten years ago and one of to-day. Ten years ago we were in the midst of a very bitter war, a war which more or less devastated this country. Look at the picture to-day. What do we see? We see that in these few years we have brought about a union of the Provinces of this country. Any other country in the world would have taken 50 years to have recovered from a war like that and have arrived at the position we are in to-day. And in thinking of that, let us give credit where credit is due. Let us give credit to Lord Milner and those who worked with him for having laid the foundations and laid them truly and well—(Applause). Lord Milner has been criticised for his extravagant ideas, with having laid these foundations on too big a scale, but to-day I believe his most bitter critics of those days are beginning to see that he was right—(Hear, hear). Let us also give credit to the Government which has succeeded his, to those who came with the idea not to destroy but to fulfil, to those who have built up steadily on the foundations laid by Lord Milner. During the ten years that have elapsed we have seen an enormous stride in the mining industry, we have seen attention given to the base metals of the country and are beginning to realise that the country has only been scratched and that possibly the next ten years will see still greater developments. In agriculture you had only to go to the Show last week to see the enormous strides made. To-day agriculture is in a far more flourishing state than before the war. I do not know whether the prize bull of this year is better than the prize bull of last year, but I do know that when one went into the produce shed one could see that enormous strides had been made in various things which have been produced from the land. That is to the credit of Lord Milner's administration and to the present Government. It appears to me that we of the different Provinces are inclined to be a little parochial, a little provincial. The way we are regarded on the Rand from elsewhere shows that there is a good deal of provincialism. We on the Rand are big enough to take a broader view and we here should do what we can first of all for our immediate neighbourhood, but equally for South Africa. Gentlemen, let us end all this petty spirit of jealousy and envy. Let us take the lead with regard to South Africa as a whole. There are many important things we have to deal with in front of us to-day. I put education first. We have all sorts of problems to decide, but I put education first I do not think you can over-estimate its importance, and I do not think you can over-estimate the importance of what you brainy men can do in helping on the education of the youth of this country. Take the recent education question—I may call it the bi-lingual question. Well we have seen that by a compromise a certain result has been arrived at and I think the best I can say of that compromise is that nobody is satisfied with it. Neither side is satisfied. It shows that neither side had all its own way, and yet I believe the outcome will be a fair and practical working scheme, but I will say this—it all depends on the spirit in which the compromise is carried out. I for one would say that we in the Transvaal, both British and Dutch, are fairly satisfied with our Education Act, and I do not think we should be the first to alter our Act until we find that in a certain other Province their Act is altered. There is an interesting experiment in Government going on in South Africa, namely, by means of Provincial Councils. These Councils were elected on party lines, by the same electorate as for the the Union Parliament. There is the Nationalist party and the Unionist party in the Council here. The difference from our ordinary Parliament is this. We have an Executive Committee which corresponds with the Cabinet of Parliament, but whereas the Cabinet is composed of men belonging solely to the Government party, here we have an Executive composed of two men from each side of the House, and I can only say from my own personal experience that everything has gone so far perfectly smoothly. There has been no conflict, which appears to me to indicate that the present political division is a purely artificial one. It is largely racial but ought not to be racial. If two Englishmen and two Dutchmen, representing opposite political parties, can sit round a table and discuss and carry on the public affairs of the Province without friction or ill-feeling, it seems to me to show the present division is artificial and is bound to break down. We are all here, British and Dutch, for good and all and one race cannot blot out the other. So let us try to conduct the affairs of the country on business lines and not have too much party and not too much politics. I will briefly touch on another and a lighter matter—the matter of sport. We have seen our South African boys, British and Dutch, holding their own on the cricket fields and football fields of England, Australia and out here. I for one believe that sport will do as much to break down this absurd division which exists to-day as anything else. I look around and I see men getting old like myself, men who I am sure have taken their part on the cricket and football fields, and I put it to you where have you made the best friendships? In the field of science or on the cricket and football fields? Sport is going to do more than anything else in this country to break down what I call this absurd racial division. This is a land of sunshine, let us try to make it a land of hope and glory. The charm of this country is as the Irishman said, that our future is before us. We are building up—possibly building up a big empire, doing more than we could possibly do in the older countries of Europe. We are building up an empire and each one in his own sphere can do something towards that. Let us determine to do what we can to make this country worthy to stand side by side with the other great dominions of the Empire.—(Applause).

Mr. Geo. Albu, responding, said: I have often attended these little dinner parties given by your Society, but to night the reply to a far more charming toast than usual has been allotted to me. I have to thank you, gentlemen, for drinking prosperity to South Africa. It is a subject which I adore very much, because having lived nearly all my life in this country, I love it as dearly as if it were the land of my birth. What I have been able to achieve I have achieved in South Africa.

South Africa in one word has made me what I am to-day—(Hear, hear). I am so imbued with my gratitude to this country that I have come to love it most dearly as the country which is nearest to my heart of all countries. Would I were a painter, an artist! In what glorious colours I would paint the magnificent scenes and landscapes which so few of us observe when travelling through the country. I have heard men speak of it when travelling through the Karroo, as a lonely veld. These men have no imagination and have never seen its sunrise and its sunsets. South Africa should be able to produce the greatest artists in the world. Have you seen the glorious sunshine in the Karroo, the colour of it, which, I make bold to say, you do not see even in a country like Italy? We do not appreciate sufficiently the beauties of this country and were I an artist I would be able to paint the most glorious pictures. Were I a poet I could sing of the glories of this country, but alas! I must say I am neither. I am sorry to say I am only just a poor mining man—(Laughter)—and a poor mining man of course looks upon these things sometimes with the eye of a materialist. To-night we have heard about the Agricultural Show, about prize bulls and calves and there is on the back of your bill of fare a prize cow munching raw quartz. It does not state how many ounces to the ton, but it goes through the usual process. I have been connected with the mining industry a great many years, and if all the predictions of our scientists, chemists and metallurgists and mining experts were realised, I think we should be working today at a few pence per ton. They have not all been realised and we are still working at a fairly high cost. However, that is a subject which has still to be thrashed out at some other dinner of a similar nature. But (looking again at the menu) here I see the crushed ore, and below the cow's tail I see the poor shareholders pulling at it with-

THE UP-TO-DATE REDUCTION PLANT. THE GOLD BRICK DAIRY.

If I'd a cow that gave such milk
I'd dress her in the finest silk,
I'd feed her on best high grade hay
And milk her twenty times a day.



With acknowledgments to Mr. E. A. Julian (Nevada). E. J. L.

out the slightest gratitude—(Laughter). Of course we are very desirous of paying high dividends and possibly no one is more disappointed if results do not come up to expectation than your humble servant. It is not a question of amassing fortunes as some gentlemen seem to have mentioned in another place. We are not simply bent on making money. We are anxious to give those who have put capital into this Industry a fair return for their money—(Hear, hear). As usual the cow has two horns. Well I have never been impaled on the horns of a cow or bull, but I have often been on the horns of a dilemma as to how to realise the promises I have made at my shareholders' meetings. Much as I enjoy a dinner of this kind, it is a different pleasure to have to eat my own words, a process which has hurt my digestive organs very much.

I again say South Africa is a glorious country with a far more glorious future. It will take years to make this country really proud of its beauty, as some of us may have noticed, when we go through the streets of Johannesburg and see those little black buckets on the wayside which seem to invite you to a quiet *tête-à-tête* or *dos-à-dos*—(Laughter). But the time will come when our Municipality will gradually alter the conditions of this town. You know that Sunday labour has been prohibited in this town; well it is very near Sunday now so you will excuse me if I do not delay you any longer. I thank you on behalf of South Africa.

Mr. W. R. Dowling (*Vice-President*): We have not met here this evening to listen to speeches by members of the Society, so I will not detain you long. At our annual dinner we try to get those who do not attend the regular meetings of members to make the speeches and change the subject from the dry matter we deal with at our meetings.

I wish to welcome on your behalf Principal Thompson, the new Principal of the South African School of Mines and Technology and Professor of Mining. Principal Thompson has undertaken a great work in which we wish him every success. He has made a good start by becoming a member of our Society.

We have also present to-night as one of our guests Mr. William Frecheville, a Past-President of our Sister Society, The Institution of Mining and Metallurgy in London, whom I wish to welcome. We have an arrangement with the Institution for the joint reading and discussion of papers, which arrangement is appreciated out here and we hope is, as well in London.

Gentlemen, I give you the toast of "Our Guests and Kindred Societies," coupled with the names of Principal G. R. Thompson and Mr. William Frecheville.

Principal G. R. Thompson: I believe I am called upon to respond to this toast because of my connection with education, not because of my six weeks sojourn in this land. I am not so precocious as Mr. Brown and do not feel that I can analyse the problems of South Africa, though I feel that I am learning very quickly about them. As a guest you justified my invitation from the fact that I had not paid my subscription to the Society. But, gentlemen, that is rectified. I have already paid it so that next year I shall not be in this position. I noticed that two years ago our President said the guests were jolly good fellows. They were not really guests. They belonged to the Society. I have shown my good sense by joining the Society. In fact I had no alternative, I had to join. I began to be rather distressed on account of some of the remarks I have heard this evening, because I was thinking you had so many speakers in this Society that I should be able to sit in a nice easy chair and remain silent. I did not expect that you wanted every member to speak at your meetings, and I should think from your volumes of proceedings that you have quite enough speakers. Mr. Dowling referred to the guests as being members. I hope in my sojourn here I shall always show my good sense by being in at every good thing. I notice that the President in his address made an appeal for the Seymour Library. I should like to extend what he said as to the importance of such a library, because coming from the Old Country where one has a series of good libraries in London, the want of books is not felt to the same extent as out here. I think any library in this country of a technical nature, should give a member who is working out a problem a chance of getting it solved. When I was in Leeds I did not trouble about our library being incomplete, because I could easily go up to London and get any information I required from the British Museum, the Patent Office, or the South Kensington Library. Now to go to these libraries from here would be a serious business so that we should undoubtedly make our technical library as complete as possible. I am certain the trustees of the library and the committee who are managing it will receive with gratitude any support whatever which will tend to the improvement of the library. As one of the guests of the evening and representing others, I thank you very heartily for the reception you have given us and for the very fine entertainment both of body and mind to which we have been treated—(Applause).

Mr. W. Frecheville, responding, said: It is my duty to express thanks on behalf of the Kindred Societies, and especially for the appreciative remarks about the Institution of Mining and Metallurgy in London. The members of that institution really appreciate the work which has been done and is being done by the Society here in this great centre of gold mining. You have many keen members working on the problems of improving the technical process for extracting the ore and cheapening costs. A very favourable feature of the Rand has always been the absence of any kind of secrecy as regards methods of working—(Hear, hear)—and the Chemical and Metallurgical Society has done excellent work in helping and stimulating further effort. During the last 12 or 15 years I have on three or four occasions had the privilege of coming here and seeing the work that is going on. The visits I have paid have been all too short, but they have all been exceedingly interesting and instructive. Over and above the technical side, there has also been the added interest of seeing the growth of the place and country. The last time I was here the great topic of conversation was the Union of South Africa, and now it it is an accomplished fact. To you who live on the spot that which has been accomplished may seem ordinary, but to us who live at a distance, it appears most wonderful to find that people who only a few years ago were shooting each other, are now pulling together to make a great country of South Africa. I regard the Rand as an excellent place for you, gentlemen, to exercise your special skill. The mines are good and the climate is good, two exceedingly important factors. Of course you know—the point has been spoken of to-night—there are drawbacks. For instance there is the unhealthiness of the mines to which allusion has been made, but I firmly believe that the difficulties can, to a great extent, be got over and in getting over these difficulties I feel confident that not only are you doing good to the worker, but you are doing an enormous amount of good to the industry itself. It is getting late, gentlemen, and I will not detain you with more remarks, but I will thank you for the very cordial way in which you have responded to the toast of the Kindred Societies, and for your very enjoyable entertainment to-night. —(Applause).

Mr. R. N. Kotze: Before we disperse to-night, there is one more toast that remains to us, I have been asked to propose the health of "Our Chairman"—(applause)—not only as representing the whole of your institution and also for the way in which he has fulfilled his duties as Chairman and entertained us to-night.—(Applause).

The President: I thank you very much for your great cordiality in drinking this toast which is a sort of extra turn, though I think it is absurd to say that I am a good chairman, or a jolly good fellow, for I am really rather a cantankerous sort of individual; but there is something to be said for the toast, namely, that it gives you an excuse for a *doch an doruis.* I'll tell you something about that! It was originally a stirrup-cup, a parting glass on the threshold, and it was not charged for. Now just try that on with the waiters—(laughter). I am sorry the Ministers were not able to come this evening, but after all it has been a good affair. Although we have not had any of the Ministers with us, we have had several very eminent politicians and I think our meeting has not been less interesting than usual. I certainly ought not to forget to mention that if there has been any success, the whole of the credit lies with Mr. Rowland who organised the thing. I thank you very much.—(Applause.)

Mr. Fred. Rowland, in response to repeated calls, said: I am not a talker, but I thank you very much. This work falls to my share and I am very pleased to do it. It is now half-past eleven and I think it is time we all went home.

Contributions and Correspondence.

A WORD ON PRECIPITATION.

In Mr. J. Hayward Johnson's contribution to the discussion on Mr. F. D. Phillip's paper* he says: "During the discussion of the paper no mention has been made of any of these bad effects, one of which is to cause the zinc to be covered with an apparently white metallic deposit giving the zinc the appearance of being newly cut or cleaned of all other deposits, whereas on examination it is found to be a coating, and all action ceases."

What is usually done is to stuff the lower compartments with new zinc, and the coated zinc is left lying dormant until it goes into the acid tub.

I take it out and soak it in a weak solution of sulphuric acid for about five minutes. The solution must be sufficiently strong to give a brisk action on the zinc. Then, with, and often without, another wash in clean water, it is returned to the heads of the boxes, and it starts doing its work straight away. I sometimes give it a little extra lead acetate.

On a big plant this could be done by running sufficient water through the box to replace the cyanide solution, and then adding a strong dip of sulphuric acid to the compartments containing the "dead" zinc.

The reverse action Mr. Hayward Johnson speaks of, seems, when it starts, to infect the whole of the zinc in the box. It may be that the adverse conditions which have started it, have also affected in a lesser degree the lower compartments, but it seems to me to be a reverse action, which starting in the first compartment, soon spreads right down through the box.

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* See this *Journal*, Vol. XI., Oct., 1910, p. —.

The following may be of interest :—We have here three slime precipitation boxes ; two are of wood and the other of iron. They run under identically similar conditions, but we get better results from the iron one. On cold mornings the tails of the wooden boxes occasionally show a trace of gold by coloration test, but the iron one has never given any trouble.

MATHER SMITH.

Barberton, April 1911.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

THE WORK OF THE CHEMICAL LABORATORIES OF THE BUREAU OF MINES.—"This work is divided among a number of separate laboratories, each carrying on its own lines of work under the direction of its own chief, the whole forming a group of more or less independent units. The relation of the work of the several chemical laboratories to that of the other departments of the bureau varies with individual cases. In general, however, the problems of the chemists are closely connected with those of the mining and mechanical engineers.

The fuel testing laboratory is occupied mainly with the analysis and calorimetric testing of fuels, including coal, coke, lignite and peat. In addition to analyzing samples of all fuels used in the boiler and gas producer tests of the bureau, ultimate analyses and calorific value determinations are made on mine samples of coal collected at various places. The data on these latter tests are of value in establishing the composition and heating value of the coals in connection with the classification of the coal fields of the United States.

In addition to the laboratory at Pittsburg, there is located in Washington, D. C., a laboratory in which are tested samples of coal, delivered to the various buildings, arsenals, navy yards and military posts within the District of Columbia and in various parts of the country, and of the coal purchased by the Panama Railroad.

Fusibility and Clinkering of Coal Ash.—In the use of coal under steam boilers, the property next in importance to its calorific value is perhaps the fusibility of its ash. Indeed, some coals, which have a high heating value, are worthless for making steam on account of their tendency to clinker and adhere to the grate bars. The relation between the fusibility and clinkering properties of coal ash and its chemical and mineralogical composition is now being investigated.

It is interesting to note that TiO_2 was found in all the clinkers examined in amounts varying from 1% to 3%. Determinations of the fusion point of various ashes give values ranging from 1,150° C. to 1,400° C.

Chemistry of Petroleum Technology.—The bureau is making a study of the commercial bodies contained in the crude petroleums of the United States ; of the methods for their separation and purification and of their economic uses. The California fields, because of their showing at this time the greater promise of a large and continued production, their proximity to naval stations and the peculiar adaptability of their product as a maritime and a locomotive fuel, have been selected for first study. Besides the determination of the properties and uses of the various products of the petroleums of the country, an investigation is being made of the processes of distillation and of the methods of refining.

Combustion Investigations.—The processes of combustion in the boiler furnaces are being investigated in a furnace specially designed for the purpose. By taking simultaneous samples of the combustion gases, the progress of the reactions may be followed and the time or space necessary for the complete combustion of various coals and under varying conditions of operation may be determined.

The process of producer gas formation is being studied from a physico-chemical standpoint, and an attempt will be made to apply on a commercial scale the results of laboratory experiments on the rate of formation of carbon monoxide and water gas.

The Composition of Coal.—Our scientific knowledge of the chemical character of coal is limited almost entirely to its chemical analysis and its adaptation to certain industrial operations. The object of one of the investigations of the bureau is the isolation and identification of some of the constituents of coal. By the use of inert solvents it has been found possible to extract as much as 35% of the original coal. A number of different substances have been isolated and the analysis and molecular weights of some of these substances have been determined. In a few cases it is believed that the materials are practically pure substances.

The Volatile Matter of Coal.—The quantity and composition of the gases evolved from various coals, when heated to temperatures of from 400° to 1,000° C., have been determined. In the experiments which are now in progress particular attention will be given to the influence of the rate of heating on the character of the gases produced : to the initial composition of the gases at the instant of liberation and to the thermal decomposition of these gases during passage over heated surfaces.

Weathering and Deterioration of Coal.—In co-operation with the Navy Department, the Panama Railroad Company and the University of Michigan, the Bureau is conducting an extensive series of tests on the deterioration of various coals in storage both in the open air and when submerged in fresh water and sea water.

The Accumulation of Gas from Coal.—The quantity and rate of formation of inflammable gas from freshly mined coal, at ordinary temperatures, and the rate of absorption of oxygen by the coal have been determined.

The Spontaneous Combustion of Coal is being investigated by the Bureau. Statistical information will be combined with the results obtained in the laboratory.

The Burning of Coal in Mines under a Diminished Supply of Oxygen.—The factors governing the propagation or extinguishing of fires in mines are being investigated. Chief among these are variations in temperature and in the oxygen content of the surrounding atmosphere.

Examination of Mine Gases.—Examination is made of samples from normal mine air, from the after damp following explosions, from stagnant areas and from burning areas during mine fires. Parti cular attention has been given to the detection of small amounts of carbon monoxide. By analyzing samples during the progress of mine fires, the chemist has assisted in combating fires. The effect of variations of barometric pressure on the exudation of methane and the influence of carbon dioxide on the explosibility of mine gases are being investigated.

The Chemistry of Explosives.—Chemical analyses are made of all explosives submitted to the Bureau for test, of the products of combustion of explosives and of electric detonators, blasting caps and fuses. All explosives, blasting caps, electric detonators and fuses purchased by the Isthmian Canal Commission are inspected by representatives of the Bureau, and all shipments of such explosives are sampled and analyzed.

Coal Dust Explosions.—The two greatest sources of danger encountered in mining operations are the explosive gases given off by the coal, and the finely divided coal dust which exists throughout most coal mines. The first danger can be overcome by increasing the ventilation in the mines. Unfortunately, this increases the danger from the coal dust by the removal of its moisture.

A laboratory method has been devised to test the inflammable character of samples of coal dust, and to classify them according to their inflammability. This method is based on determining the amount of combustion which takes place when clouds of dust of the same density are ignited under the same conditions, the amount of combustion being determined by the pressure developed within the explosion vessel. In this way it is possible to obtain results on any one sample of coal dust which agree to 3·5% of the total pressure developed.

One of the proposed means of lessening the inflammable character of coal dust is to add a non-inflammable dust. The laboratory method used to investigate the inflammability of coal dust has been extended to various percentages of coal dust and finely ground shale in order to determine to what extent the combustion is limited by the presence of the inert dust. The experiments indicate that a marked diminution of pressure is not obtained until about 25% of inert dust is added, the pressure then falling off rapidly with a further increase in the amount of shale dust added."—D. J. K. Clement, *Metallurgical and Chemical Engineering*, Vol. 9, No. 2, 90, Feb. 1911. (J. A. W.)

METALLURGY.

Commercial Classification of Fuels.—"Fuels in their natural state may be classed as (1) gaseous, (2) liquid, and (3) solid. Under the third class, coal only will be considered in this paper.

Coal is technically classified as anthracite, semibituminous, bituminous, and lignite.

In the commercial classification of fuels, coals are known as (1) steam, (2) by-product coking, (3) producer gas, (4) illuminating gas, (5) cement, and (6) domestic.

Steam coal constitutes more than 50 per cent. of all coals mined, and covers a wide range in quality. For steam production all kinds of coal are used, with the following range in analytical values :—

Volatile matter, from 15% to 42%.
Ash, from 4% to 18%.
Sulphur, from ·8% to 5%.
British thermal units, from 9,000 to 14,800.

Anthracite is not included and will be eliminated from further consideration, as its use is confined principally to the large cities and densely populated districts along the eastern seaboard of the United States, where local conditions determine almost entirely the fuel to be used.

Steam coals may be subdivided into locomotive, steamship, and stationary power plant fuels.

Locomotive fuel is required to meet rapidly the maximum variations in demand for steam, and a coal which will deliver a considerable portion of its total heating value in the shortest possible time is the most satisfactory. This requirement is best met by gas coal, as it will give up about one-third of its total heat within two minutes after firing, due to the liberation and combustion of the volatile matter. The remaining fixed carbon maintains a constant temperature by its practically uniform rate of combustion.

That the manner of firing and the condition of the locomotive are the ruling factors in 'smoky' or 'smoke-less' operation of locomotives with gas coal has been proved by some of the leading railroads. In such cases three-quarter gas coal is used with very satisfactory results, and almost smokeless combustion.

To avoid excessive smoke, low volatile coals of high heating value are often used, but they do not burn as quickly as gas coals, a shortcoming compensated for by the reduction in smoke.

Aside from regions where low-volatile coal is the most available fuel and localities where strict smoke laws prevail, low volatile coal is used by some railroads on certain passenger runs in order that the quantity of smoke may be more easily kept at a minimum.

While high-ash coal is not especially desirable in locomotive fuel it is used with very satisfactory results, due to the readiness with which the ashes can be removed from the firebox as well as to the increased grate surface with which recent locomotives are provided. For locomotive fuel, sulphur in coal need be given no serious consideration. The belief that locomotive coal must be lumpy or screened is gradually disappearing.

Owing to the strong draught in a locomotive firebox a large proportion of fine coal is objectionable, as considerable of it may be carried through the flues without combustion. The tendency to exaggerate the extent of such loss is due largely to prejudice against any but lumpy coals for locomotive use.

Steamship fuel is required to meet a steady demand for steam, as the boiler load remains practically constant once the vessel is under way. For such service a fuel high in heat units and having a practically uniform rate of delivery of its heat units is best adapted. Low-volatile coals are most desirable for steamship fuel, and nearly all vessels of the trans-Atlantic trade and of the United States navy have adopted such fuel.

Practically the entire tonnage of low volatile coal reaches the market as run-of-mine, consequently the percentage of small coal is considerably in excess of that in a similar quantity of run-of-mine bituminous coals ; but this is not objectionable in steamship operation, because, with the draught much less than in locomotive operation, the loss due to unburned coal is small.

On some freight vessels sailing the Great Lakes mechanical stokers have been installed that use slack of bituminous coals with satisfactory results, as far as can be learned.

The chemical analysis for steamship fuel would be within the range of percentage values given for locomotive fuel—specifications usually requiring ash under 10%, sulphur about 2% or under, volatile matter approaching the lower limit, and British thermal units near the upper limits. Sulphur content is of no importance, and the requirement of 2% or under must be met to satisfy the purchaser rather than because of any detrimental effect the sulphur may have on the fuel value of the coal.

The load of large central power stations usually varies greatly, and to meet the peak loads the fuel must generate steam quickly. The advantages of meeting this demand by using a coal which will give up a large portion of its heat units in the least time after firing becomes more apparent as the boiler load and capacity approach an equality.

The selection of coal for large power plants in densely populated districts is influenced by the smoke laws operative in such localities, and to comply with the law low-volatile coals are given preference.

In boiler plants fired by mechanical stokers in the eastern United States, low-volatile run-of-mine coal is used, crushed so that no piece exceeds 3 in. in size.

In the central section the prevailing practice in large plants is stoker-fired boilers, using slack or screenings of bituminous coal. Fuel of this size avoids the necessity of crushing, and, for the greater portion of the time, affords practically smokeless operation. Bituminous slack is capable of meeting sudden changes in load more readily than low-volatile coal, and allows more uniform operation under widely varying plant loads.

Where freight is the larger part of fuel cost, the highest quality of coal is the best investment for the purchaser, as the cost of transporting coals of any quality for the same distance would be identical. For this reason principally, the eastern United States is the great market for the best grades of low-volatile coal. In the central section slack is the cheapest fuel obtainable.

If the use of low-volatile coal is advantageous in large boiler plants equipped with mechanical stokers, the peak loads are usually provided for by having ample boiler capacity. In small power plants with hand firing, the size of coal is considered of importance. In rare instances do such plants have the variations in load commonly found in large power stations. Taking into consideration steam fuels used in the various regions, it is found that they cover the entire range of chemical values, but in some small plants a coal low in ash is desirable on account of furnace-grate arrangement.

Smoke prevention is frequently of much importance. Care in manner of firing and selection of coal will usually permit of operation within the requirements of the smoke laws.

For by-product coking, requirements as to quality are far more strict than for steam coal, the range for sulphur being confined to rather narrow limits. In the by-product process coal is coked in such a manner that the gas, tar, and ammonia are recovered. Gas is sometimes the principal product and the ammonia, tar, and coke are the by-products.

A satisfactory by-product coal must be rich in gas, and meet certain requirements as to quantity of some of its constituent elements, the limits of which are determined largely by the purposes for which the coke is to be used. This means that gas coals are most desirable for such use. While many low-volatile coals will yield a satisfactory coke, their yield of gas is low, and bituminous coals other than gas coals are unsuited, because they are non-coking and their gas lacks the rich illuminant found in strictly 'gas' coals.

Where illuminating gas is the principal feature and the coke is not intended for use in iron manufacture, limits of sulphur are not so closely drawn, although it is difficult to purify gas made from high-sulphur coal. 2% or over of sulphur in coal has been used in such plants, the coke being used for steam and domestic purposes.

For by-product operation in connection with steel plants, sulphur limits in coal are ordinarily placed at 1·50% as a maximum, with an average value of about 1% to 1·25%.

The standard range for coal for by-product use is 6% to 7·50% ash, which produces a coke ranging from 9% to 11% in ash. Ash is objectionable in coke for iron making, as its final appearance is in the form of slag, which requires heat for its production, representing so much loss of heating value and in part explaining why limits have been established.

In general, the range of analytical values for by-product coals would be represented by:

Volatile matter, from 28% to 38%.

Ash, from 6% to 7·50%.

Sulphur, from minimum to 1·50% for coke used for metallurgical purposes. From minimum to 2·50% for coke not intended for metallurgical purposes.

Fuel for producer gas covers a wide range in values, especially since gas engines are meeting with increased favour as prime movers.

Producer gas found favor by supplying the demand for cheaper fuel, and is used principally for metallurgical and power purposes, although it is applicable for kiln-burning of clay products, lime cement, and firing steam boilers. The principal use of producer gas is in the manufacture of steel, which requires a coal ranging from a minimum to 1·5 % in sulphur content.

All coals will produce about the same quantity of producer gas, but in metallurgical processes gas coal is preferred, for the reason that its gas burns with a long flame, giving more uniform distribution of heat at high temperature.

Other bituminous coal from which the demand could be supplied, and satisfactory temperatures obtained, have a prohibitive sulphur content. The selection of fuel for producer gas is limited by the use to be made of the gas and the type of producer.

Producers are classified according to their method of operation as, (1) suction, (2) pressure, and (3) down draft.

The suction producer has found application in small plants operated by gas engines, and the fuel has been limited almost exclusively to charcoal, anthracite, and coke, or fuels whose gas is free from tar, which is very objectionable in gas-engine operation.

Pressure producers are operated under a low pressure produced by a blast of steam and air. This is the type generally used for metallurgical purposes where the gas is used direct from the producers and tar is not objectionable.

For gas-engine use the gas is stored and the tar removed before passing to the engine, and lignite, peat, and bituminous coals, as well as charcoal, coke, and anthracite, may be satisfactorily used in this type of producer.

Tests with a pressure producer conducted by the United States Geological Survey at the various government testing plants have shown satisfactory results with fuels ranging widely in analytical values, as follows:

Moisture, 1·40 to 39·60%; volatile matter, from 9·70 to 42·50%; fixed carbon, from 23·80 to 73·70%; ash, from 2·70 to 23·40%; sulphur from ·30 to 7·40%.

It is reported that bone coal containing 44% of ash produced a gas affording economical gas-engine operation, and that any coal of commercial value can be successfully used.

Down-draft producers made their appearance after the pressure producers. These fix the tar as a permanent gas, thus making use of all the volatile matter and preparing it for immediate gas-engine use.

Gas producers are utilizing more and more the inferior grades of fuel, and at present are operated with fuels covering the range from anthracite to charcoal and peat.

In illuminating gas manufacture, practically all plants specify a screened gas coal of low sulphur content.

Three-quarter-inch screened coal is specified for several reasons:

1. It can be gasified in less time than run-of-mine coal.
2. It is lower in sulphur than run-of-mine coal.
3. Charging of retorts by hand is readily accomplished.

Low sulphur is required that the gas may be purified to meet the requirements of the law.

The commonly accepted standard of analytical values for illuminating gas coal would be approximately represented by the following: Volatile matter, from 32 to 37%; ash, from 6 to 8%; sulphur, not to exceed 1·50%.

The usual standard for yield of gas is an average of 10,000 cubic feet per ton of 2,000 lb. of coal, with an average candle power of 18.

Coal for cement burning is reduced to a powder before being used, hence slack coal is generally used because of its low price and the readiness with which it can be reduced. The essential requirements are that the fuel be: (1) Sufficiently high in volatile combustible matter to insure quick ignition; (2) sufficiently high in heating power.

While the maximum sulphur content of cement is definitely fixed, the sulphur in the coal has not been closely limited, as it does not enter into combination with the cement as long as proper kiln temperatures are maintained. As the impurities of some coals are approximately of the same composition as the cement, no strict limits have been placed on the amount of ash in such coals, but high ash percentages reduce the heating power, and, for this reason only, a low ash content is desired.

Gas coals have been found to give best results in cement burning, as they are high in volatile matter, which has a high heating value with long flame of quick ignition and maximum temperature at a short distance within the kiln.

If coal which ignites slowly is used, ignition takes place too far inside the kiln, giving the zone of maximum temperature at such a distance from the firing end that difficulties are encountered in securing proper clinking of the cement, besides increasing loss of heat due to the waste gases leaving the kiln at a high temperature. While gas coals are considered the best for cement burning, other high-volatile bituminous coals are used satisfactorily.

Domestic coal must meet consumers' requirements as to size, and is prepared as lump, egg, and nut. Coal firm in structure and suffering the minimum amount of breakage in handling will meet domestic requirements most satisfactorily. It should be non-coking or free burning, so that its heat is given off readily. The ash should be small in quantity and free from clinker.

Export bituminous coal must be in large sizes and of firm tough structure, that it may reach its destination in large sizes with minimum amount of breakage. This coal is usually passed over 1½ in. screens before shipment from mines.

With the low volatile or semibituminous coals which produce a large amount of fine coal in mining, size of coal for domestic purposes is obtained by screening. For export these coals are usually shipped as run-of-mine.

In general, the preparation of coal for market by the removal of slate, bone and sulphur balls, is of importance to the producer as well as the purchaser. It is of importance to the producer in extending his market by maintaining the reputation of the coal he produces at a high standard, and to the purchaser that he may receive the greatest fuel value for the money expended.

Sulphur balls produce about one-third as much heat in burning as the same weight of coal, and on this basis alone are not so objectionable as other impurities, which have no heating value whatever. They are objectionable on account of difficulty in handling the clinker they produce.

The impurities without heating value go to increase the amount of ash, and represent so much worthless material at the same price and freight rate as the coal. As transportation is usually the greater portion of cost of coal to the consumer, the importance of removing as much free impurities as possible is readily apparent.

The selection of coal to be used for any particular purpose is dependent upon, cost to the consumer, limitations imposed by smoke laws, and fuel value. The fuel value is greatly influenced by the readiness with which clinker is formed from the ash, the greater the freedom from clinker the greater is the recoverable 'fuel value.'"—R. E. Rightmire, West Virginian Mining Institute.—*Mines and Minerals*, Feb., 1911, p. 397. (A. R.)

An Early Cyanide Plant.—"One of the first plants, if not actually the pioneer plant to treat ore on a commercial scale by the cyanide process was built near the Calumet Mine in Shasta County, California, in 1891, by A. B. Paul. It was thus described in the *Mining and Scientific Press*, in the issue of October 3, 1891:—

'The plant is intended solely for the working of the McArthur-Forrest process, not alone on the ores of the three counties for which the company owns patent rights, but for all the ores that may be shipped from all portions of the coast. The plant will treat 10 tons of ore every 24 hours, and is so arranged that different lots of ore can be treated at the same time. In the treatment of the ore, the first operation is drying. It is then passed through a rock-breaker and into bins, from which it is fed into a Paul barrel-pulverizer, and when powdered the ore is placed in agitators and a 1% solution of cyanide of potassium added. After an agitation of 6 to 12 hours, the liquor is drawn off into filtering tubs. These filters are of wash-gravel covered with canvas. The liquor passes through the filter and into storage tanks. From this the solution is drawn into a chest of zinc filters, each filled with zinc shavings. The liquor flows down through the first (box), up through the second, down through the third, and so on, to the end of the eight filters. The gold is precipitated upon the zinc shavings in the form of a brown powder. When desired, the chest is unlocked and the zinc shavings washed in clear water, which separates the gold. When it has settled, the water is drawn off, and the gold, in the form of brown powder, melted into bars. The liquor from the filtering tank is pumped back to the first tank, and sufficient cyanide of potassium added to bring the solution up to the original 1%. As will be seen, the process is very simple; no roasting of ores is needed, and no high-priced chemicals required, with a very small loss of materials used. . . . As numerous parties have failed in making small working tests of this process, it may not be amiss to state that very often cyanide of potassium is not more than one-half to two-thirds full strength, and it is therefore necessary to know the exact percentage of cyanide as well as to follow the company's method of treatment.'

This is one of the first articles descriptive of the cyanide process ever published in a technical journal. The details given are sufficiently comprehensive to make the operation of the process easily understood, as far as knowledge of it went, but subsequent development of the process proved it to be far from the simple application of a long recognised fact—that a weak solution of potassium cyanide would dissolve gold. Since that time, the autumn of 1891, tons of literature of the highest type of classical contributions to metallurgy have been published, and experimenters are still earnestly engaged in the study of methods by means of which the process may be still further improved and its commercial application successfully extended to the treatment of those ores which are still refractory."—*Mining and Scientific Press*, January 21, 1911, p. 137. (W. R. D.)

Screening versus Classification Prior to Concentration.—"As to the comparative advantages of screen sizing and hydraulic classification for table feeds, there is room for intelligent difference of opinion. In some cases, an unwarranted prejudice may be due to failure to recognize a fundamental difference in tables. In jigging practice, sized feeds are particularly favorable to making clean gate-discharge products while classified feeds are favorable to making clean hutch products. For vanners, the evidence is in favour of classified feeds, owing to the sizing action of water flowing over smooth surfaces. In the case of reciprocating tables, conditions are more complex. The material on the table is subjected to a combined stratifying action, due to agitation, and a sizing action, due to the water flow. These influences together determine the course of the various mineral particles over the table and the grouping of these mineral particles varies according to the relative activity of the two influences. This relative activity differs on different tables (and on different parts of the same table), according to the number, depth and form of the riffles. Numerous, deep, full length riffles give the stratifying action its maximum importance, and 'sized' feeds are better adapted to making high extraction and clean products. On smooth surfaces, the sizing action of the water flow predominates, and classified feeds are preferable. Between these two extremes can doubtless be found tables which will do equally good work on either class of feed, if properly adjusted. When it is considered, however, that the relative importance of the two influences at work varies on most tables with the slope, the rate of feed, and amount of water used, it is not surprising that contradictory experience is reported, where from superficial data, general agreement might be expected.

The Scope of Classifiers and Sizers. One other factor, which must be given due weight in any comparison, is the efficiency of the 'classifying' and 'sizing' operation in each case. The results attainable by either method improve as the classification or sizing approaches theoretical perfection.

In the end, either method must stand the test of economic utility. Probably mechanical considerations bearing directly on costs, more often control the choice than any direct advantage in extraction or degree of concentration. But for this fact, some general principles might be evolved, which would tend to standardize mill work in this respect. For installations where the largest capacity compatible with clean products is desired, close screen sizing seems to have the strongest claim to consideration. In low-grade milling operations the tendency to over-load tables is very strong, while, at the same time, the low grade of the ore urges extreme simplicity of plant. The number of instances in which hydraulic classification is favoured under these conditions, therefore, serve to emphasize the preponderant importance of the economic side of every concentrating problem."—J. R. Blake, *The Engineering and Mining Journal* (New York), p. 256, 4th Feb., 1911. (W. A. C.)

Gold from Clay Deposits.—"A new method of winning gold from clay deposits is due to J. J. W. H. van der Toorn (German patent 225,810). The processes now employed treat clay with water whereby a good deal of the gold is lost, as the separation of gold from clay is very difficult on account of the great cohesion of the clay. The new process tries to remove this cohesion. The gold-containing clay is spread over a large area, dried and kneaded. Then it is heated in a furnace similar to a brick kiln, till it assumes the hardness of stone. The rocklike masses are then broken in a stamp mill and the gold separated in the usual manner."—J. J. W. H. van der Toorn, *Metallurgie*, Dec. 8, 1910.—*Metallurgical and Chemical Engineering*, ix., 2, 106. (J. A. W.)

Progressive Mill Practice.—"California is often referred to as the cradle of gold mining, and such, indeed, it really is, but it is a matter of general surprise that there is a tendency in some directions in California to remain in the cradle. Particularly is this noticeable in the treatment of gold ores of the State. Other gold-mining regions throughout the world have in their early history, almost without exception, adopted California mill practices, but in many cases the newer countries have promptly drifted away from the 'time-honoured customs' of California's mill-men and evolved newer and better methods through experimentation, or have applied those already tried out in other new regions. In no district is this departure from traditional methods more radical than on the Rand. The ores of the Rand are relatively simple—a quartzose gangue with metallic gold and auriferous pyrite. The ore yields its gold readily to amalgamation and cyanidation. As concentration is not considered necessary on the Rand, the engineers, to reduce cost, years ago began to seek for methods of milling which would permit an increase of stamp duty. Year after year saw the stamp duty increased by various ingenious means, until now, by coarse crushing and re grinding, the capacity per stamp, as at the East Rand mills, for instance, has reached 20 tons daily, and even a higher rate is anticipated, as compared with that of 4 to 6 tons per stamp in the average California mill. That the gold ores of California are mostly readily amenable to the simple methods of recovery practised on the Rand is well known, and why California mill-men do not make an effort to increase mill capacity along lines similar to those that have been successfully evolved on the Rand is one of the things not easy to understand. An increase of stamp duty without any material increase in the cost of power will certainly result in a decrease of milling cost per ton. At many California mines, by adopting the re-grinding methods of the Rand, employing Chilian or tube-mills after coarse-crushing in the stamp battery, the capacity could be raised from 4 to 5 tons per stamp to 10 and possibly to 15 tons per stamp daily, while making as large a saving as by present methods. It seems well worth the effort, at any rate, and if the attempt be successful it would permit a stated capacity at a much lower capital expense for equipment."—*Mining and Scientific Press*, Jan. 21, 1911, p. 129. (W. R. D.)

MILLING AT COBALT.—" An interesting feature of ore dressing at Cobalt, Canada, is that the methods in use had to be devised especially for the ores of the district. No precedent could be referred to by the metallurgist and it is greatly to his credit that he has succeeded so well. At present thirteen mills are in operation by electric power, and three more are in course of erection. Wet methods of dressing are adopted, and in some cases hydrometallurgy is practiced to the extent of the final recovery of the metals at the mill.

No two mills are exactly alike, but are the same in general principle. For illustration, the equipment and practice of the Coniagas mill may be taken as typical. The ore is crushed in Blake and gyratory crushers to $1\frac{1}{2}$ in., then rolled to $\frac{3}{4}$ in. and screened in trommels. The coarse is concentrated in Harz jigs and the fine on Wilfley tables. The residue from these is crushed in 30 stamps, following which the pulp is thickened and treated on Deister and Wilfley tables, followed by a canvas slime plant.

Considerable difficulty was experienced in the early work in saving values from the tailings, owing to the escape of small scales of metallic silver and smaltite which seemed to have a great tendency to float away. By paying more attention to the slime recovery this loss has been largely averted. The average extraction in the district is probably 82%. During 1909 the ratio of concentration was 39 to 1. The cost of milling was higher in 1909 than in 1910 owing to the use of cheaper electric power in the latter year. The cost of the work at the McKinley-Darragh mill was $1.98 in 1909, and is reported to be about $1.30 in 1910. Power formerly cost in the neighbourhood of $150 per hp-year, and is now obtainable at about $50.

At three plants—the O'Brien, Buffalo and Nova Scotia—cyanide plants are operated in connection with the concentration. Dr. Mandy considers the success of this process as yet problematical. The Nova Scotia plant uses pan amalgamation and cyanidation, but no figures are available on the success of this system.

The concentrating plants all treat what is known as the low grade ore, containing 10 to 40 oz. silver.

The high grade ore, varying from 100 to 3000 oz. silver, is treated in smelters. The smelting ore is further divided into low and high grade; the first carrying from 100 to 600 oz. silver and the last from 800 to 3000 oz. The purchasers pay a sliding scale according to the silver content, and have various schedules for the arsenic content. United States smelters treated $\frac{2}{3}$ of the tonnage in 1910, and Canadian plants treated about 32%, the balance going to England and Germany.

The ores are silicious and make a good fluxing ore in silver lead smelting. They also may be used for converter lining, but for this purpose there are several objections, viz., the arsenic fails to volatilize completely, and practically not at all after the first 10 min. of blowing, causing about 1·5% of arsenic to appear in the copper; the slag is inclined to be thick; the charge effervesces owing to carbonates in the ore, causing high copper slags, sometimes as high as 15%; alumina frequently has to be added to the ore for binding; and finally, the lime and iron in the ore tend to rob the lining of active silica, leaving less to do the work of slag production."—Dr. JOSEPH T. MANDY, *London Mining Journal*, Dec., 1910.—*Metallurgical and Chemical Engineering*, February, 1911, p. 105. (J.A.W.)

MILLING NATIVE SILVER ORE IN MEXICO.—" The following is a description of a combination of amalgamation and cyanidation for silver ore. The ores of the Batopilas silver mines are peculiar in that the clean base minerals, galena, blende and pyrite, seldom carry more than 4 oz. silver per ton. The values occur mostly as native silver, argentite and sometimes arsenical silver.

The ore is sorted into two classes, the high grade native silver ore which is treated without concentration, and the low grade milling ore which is concentrated.

The high grade ore is stamped through a fine screen. Some coarse native silver accumulates in the mortar and is removed and washed by hand prior to smelting and refining. The screened material is amalgamated in pans by the following process: The charge for a 4 ft. pan is 1000 lb. ore, 15 to 17 lb. mercury, 4·5 lb. cyanide of sodium, about 4 oz. lead acetate, 3·5 lb. lime, and 1 to 1 of stock solution. The solution in the pan then runs from 40 to 48 lb. cyanide per ton, and never below 33 lb. The ore is first ground for six hours without mercury, when it is added and grinding continued for 10 hours. The charge is then run into a settler where it is leached and agitated, mechanically and by air, for 12 hours. The settler is then washed four hours, and the amalgam and residues removed and washed by hand. The former is properly retorted, and the latter is sent to percolating tanks, referred to later.

The slime washings from the settler are thickened in a Dorr machine, the clear overflow of which goes to strong solution sumps, and the thickened underflow to Pachuca tanks. One of the latter is filling while the other is being agitated and drawn off. Blaisdell filters are used, four charges being put through in 8 hours. The tailings from the filter run about 9 oz. silver per ton.

The low grade ore is concentrated 50 into 1. The concentrate consists of pyrite, galena and blende, with native silver and argentite. It is reconcentrated into two classes, high and low grade. The high grade is amalgamated in a pan, adding mercury pound for pound of silver contained, and a slight excess. Grinding proceeds for 14 hours before amalgamation is started, this being continued for 2 to 5 days. Cyanide, lime and lead acetate also form a part of the charge, and the pan is decanted twice daily into the rich solution sump, the solution being made up again with stock solution holding 10 lb. KCN per ton. After 6 days the pan is discharged into a settler, where it is agitated mechanically and by air for six days, the solution being decanted once a day. The tailings from the settler go to the Dorr thickener, and thence to the Pachuca tanks.

The low grade concentrate, which forms the bulk of the product treated, is leached in the percolating tanks referred to above. The charge consists of from 35 to 60 tons of ore: the chemicals added being lead acetate, lime and cyanide. The latter is added in sufficient quantity to make 50 lb. KCN per ton. The solution is left on 24 hours, then drawn off and the tank allowed to aerate for 48 hours. Stock solution of 10 lb. KCN per ton is then run on and allowed to stand for 12 hours, after which it is withdrawn and the tank allowed to stand dry for 24 hours. This process is continued with 12 hour periods of percolation and aeration for two months when 95 to 96 of the value has been leached from the concentrate. The precipitated silver assays about 800 fine, and the refined product about 990 fine."—W. M. Brodie, *Mexican Mining Journal*, Jan., 1911.—*Metallurgical and Chemical Engineering*, Feb., 1911, p. 105. (J. A. W.)

MINING.

Transvaal Stope Drill Competition.—"The following is a summary of the report of the Underground Manager of the Stope Drill Competition, Mr. Tom Johnson, which gives the practical results of the trials in a concise form:

Machines.—I think all the percussive machines are too light in weight for general work. They should be at least 130 lbs. for the smallest of them, say, a 2½ in. machine. They all require larger feed screws, steel wearing washers between the shoulder of the feed screw and feed screw bridge, and chucks to carry 1 in. diameter shanks. The nuts on the valve chests ought to be more accessible for the use of a spanner, and the cradles should be longer.

Chersen.—This machine proved itself at the elimination trials, and on the testing blocks, to be a fast driller, but in general work in the various stopes it failed to accomplish anything like the work that might have been expected from it. It does fair work in a leader stope where the holes can be put under and over the reef, but when it had to drill in reef there was trouble, as the machine failed on the back stroke. Another drawback was that a very small bit of grit stuck the valve, but this might be overcome with a good type of air strainer.

Holman 2½ in.—This machine, next to the Siskol, is the one most favoured by the miners, for its freedom from trouble. This type allows the miners to drill their holes as they want them, and can put down 5 ft. holes with ease. With a free running machine like this, a miner has a better chance of breaking rock than with a machine which has not a sufficiently powerful back stroke, and through this cause loses holes or has to leave them short in length. The spares costs on this machine are fairly low, which puts it in a good position under the Rules of the Competition—i.e., footage costs.

Holman 2¾ in. This machine has the same fault as the Chersen and the New Century in not being able to run freely for long periods. It drills faster than the Holman 2½ in. when actually running, but cannot make the same drilling time during the shift. A very small piece of grit no larger than a pin-head getting into the valve stops the machine from working. If this valve were arranged that the entrance to the valve was the smallest part, a little grit getting in would go away with the air instead of sticking the valve. The use of a good strainer would prevent a lot of trouble.

Siskol.—This machine is one of the most favoured by the miners, *for one* reason - that when a hole is started there is more certainty of getting the hole down, and also to a greater depth, than with the other machines. This machine often drilled 6 ft. holes in reef in the stope at 2,600 level, Robinson Deep, and I think could drill 7 ft. holes comfortably. This and the Holman 2½ in. are the best machines for all-round work. It has a sufficiently powerful back stroke, and does not get stuck up with every grain of dirt that gets into the valve chest. This machine must have been badly handled during its run on the Village Deep and the Crown Deep E.F.—e.g., on the Crown Deep E.F. the footage per shift was 43·9 and on the W.F. 79·5 in practically the same ground, as it was practically a new stope at the time. The air pressures were 80·5 and 80·18 respectively. This machine is lowest in labour and air costs, but highest in spares, much of the latter being due to the competitors sending out a machine for the use of hollow steel, and having to use these unsuitable spares under the Rules of the Competition. It should be provided with a half chuck bush instead of a full one, as at present, and provision should also be made for wearing strips for taking up the wear in the cradle.

Running Conditions.—Supply of Drills.—Steel handling on two of the mines gave very little trouble, due in a great measure to the stopes being close to the shaft. On the other two mines quite an amount of time has been lost on account of the poor system of handling the steel. There have been times when 200 drills per machine have not been sufficient to keep the machines going, a very large amount of steel having been lost on the various mines. Where the drills could get to no one seems to know. It is true starters easily get buried and the other drills easily mixed with hand drills, and all trace of them is thus lost; but, taking everything into account, the loss is very much more than it ought to have been.

Air Pressure.—The running of the machines was suspended at the Cason mine on November 5, 1909, owing to the prevailing low pressure. It was not resumed, and the footage drilled in this mine was disregarded. At the Crown Deep we have lost time since the beginning of December until about the end of February, owing to low pressure. In the fore part of the competition the Crown Deep provided the highest pressure of air, and in the latter part of the run of the Holman 2½ in. (the last machine to run there) the pressure has again been good. The Villiage Deep and the Robinson Deep have provided a good and fairly continuous supply at good pressure, particularly the Robinson Deep.

Boys.—The boys supplied on the various mines have usually been up to the average of mine boys. There have, of course, being odd shifts with poor boys, as will always happen at times on all mines. The only time the supply of boys has caused unusual loss was during the run on the Siskol machine in the 2,700 stope of the Robinson Deep. Miner Brown had bad boys for 16½ shifts, as reported by me on December 15, which undoubtedly handicapped the machine. The reason that Brown did not have better boys was that during the time he was at the Village Deep two of his boys died and another went home, and when the Siskol started at the Robinson Deep the company was troubled with a sudden sickness that attacked their boys, and consequently was not able to do better for us.

Holes per Bench.—It has not been possible to go thoroughly into the merits of two hole versus four-hole benches. What little has been done in this matter and my own experience leads me to believe that with these light machines, which can be so quickly unrigged and rigged up again, two hole benches with ordinary miners would without doubt be much the best as regards cost per ton of rock broken. With the present heavy machines in ordinary use. I do not think that there would be any gain in using two-hole benches, as the time lost in moving would wipe out the gain. The only way that I can see to be advantageous by two-hole benches with heavy machines is to keep a spare machine in addition to bars so that the machine boys after finishing their two holes move to a fresh machine already rigged up. The machine that had completed its two holes would then be taken down and rigged on a fresh bench ready for the next pair of boys who had got their bench finished, and so on in rotation. I think it is better to give manufacturers encouragement to build a 3 in. or 3⅛ in. machine weighing not more than 200 lbs. for wide stopes, as a machine of this weight, with rigging gear in proportion, would no doubt pay for use on two-hole benches, 200 lbs. not being a prohibitive weight in stopes over 4 ft. wide.

Work Done.—It will be noticed from the attached table that there is quite a difference between the footage drilled per hour per machine during the competition proper and during the elimination trials and air tests. During the elimination trials the guage of the bits was less. One white man was on one machine ; machines were supposed to be in the best of order, and it was pure competition work for the three days. The same thing obtained at the air tests, smaller bits, machines in apple-pie order, special men, and everything handy. Some of the competitors have built too much on the air test figures. A curious fact of the air tests is that of the finishing machines the Holman 2⅛ in. has the smallest average inches per minute, yet stands so well in work done in the mine. In the competition proper there has not been much actual competition work done during my time, whatever may have been the case during the early part when the work was fresh. Machines could not always be at their best, parts must wear and be replaced, the change in personnel, loss of enthusiasm, and a hundred and one little things that must happen on a long run, but not on a three days' run, help to account for the difference in footage drilled.

Machine.		Footage per Hour at Underground Elimination Tests per Single Machine.		Footage per Hour Competition per Single Machine.
Holman 1⅞ in.	...	6 ft. 5 in.	...	3 ft. 8·5 in.
Holman 2¾ in.	...	7 ft. 0 in.	...	3 ft. 5 in.
Chersen ...	...	8 ft. 3 in.	...	3 ft. 4·3 in.
Siskol ...	...	4 ft. 8 in.	...	4 ft. 0·7 in.

Machine v. Hammer Boys.—Costs.—In the competition two sets of machines, the Holman 2⅛ in. and the Siskol, have cost approximately 8·3d. per foot drilled plus cost of steel and drill sharpening, which should come to at the most 1·5d. per foot, making 9·8d. per foot. But it has taken 1·282 mine shifts to make one 8-hour shift ; therefore the wage cost must be increased in that ratio. Further, instead of assuming 10s. per shift for two machines as the white wage, I have taken 25s. per shift for four machines. This alteration means an increase of ½d. per foot drilled, making the total cost per foot for these two sets of machines practically 11·8d. per foot, this being as near as I can get to the actual cost per foot for 28,528 ft. drilled by these machines. This includes the footage drilled with low air pressure. The loss of air in transmission and depreciation of compressors are not charged in these costs. This cost of 11·8d. per foot compares favourably with the cost by native labour, for which 1s. 1d. per foot would be an exceedingly low figure. The tonnage broken should be as 6 to 5 in favour of the machines ; the cost of explosives would be as 6 to 5 in favour of native labour. The tendency is for natives' wages and costs to increase, and this is the heavy item (10d. per foot) in hammer work, but a much smaller item (only 3d.) per foot in machine work.

Depth of Holes.—With a proper size of bit a hole 4 ft. 8 in. or 5 feet deep is much better as regards tonnage than the ordinary boy's hole, for it will carry more explosive, and so a greater burden. A man automatically puts a greater burden on long holes than short ones. For these several reasons I think that even if machine drilling cost us as much per foot drilled as the hammer boys, which I have shown they do not, the extra tonnage from the machine holes would more than pay for the expense of extra compressors—cheaper cost per ton being the true comparison.

Stoping Width.—As to stoping width, these machines have run in stopes of an average width of 30 in., while sometimes the width was as low as 24 in. in S.W. 1 W. at the Cason. There were two periods done at 43 shifts at an average stoping width of 30 in., three others at an average width of 35 in. to 36 in., and seven at an average width of 36 in. to 40 in., showing that the machines are capable of narrow stoping if the mine authorities will insist on narrow stoping, as 12 out of 28 runs have therefore been done at less than 40 in., the last run at the Crown Deep will also be under 40 in., and making 13 out of 30. It has not been the fault of the machine that the whole of the stoping was not done at less than 40 in. In one stope, the reef being in two portions, it was necessary to stope wider ; in other places carelessness on the part of the miners and the mine officials was the cause of the greater stope widths. The average width for the total work is 44 in. to 45 in. I know that there is an idea prevailing that these machines cannot stope narrow, but this competition has proved that 43% of the work done has been in stopes 40in. and under. With 15% of the work it was necessary to carry over 40 in., leaving 42% over 40 in., most of which was due to the mine officials not wishing to interfere with the competion men.

Hammer Drill Machines.—There is little I can say with respect to hammer drill machines during the competition trials, for there was only one hammer drill machine (Climax Imperial) running at the time I joined the competition staff. From what I can gather the troubles were steels sticking in the holes, water flying about, and steels breaking at the shanks. This will always be a trouble in drilling down holes of greater depth than 40 in. or so, as there are sure to be fluctuations in the supply of water to the bits on account of the hole in the steel choking with dirt, or a soft bit getting burred and the hole closed that way, or water getting shut off. Outside the competition it is being proved that the hammer type of drill is capable of doing good work drilling dry uppers. There is no reason why it could not do stoping. A separate jet can be used to allay the dust, and solid steel used as at present in the rises, although, personally, I think a water fed hollow steel arrangement will yet be got out that will not throw the water about, but send it up the hole where it is wanted. If it could be proved that the hammer type of drill could drill the ground cheaper than hammer boys, there is no one who would not be ready to start the stope to the rise instead of to the dip. I do not mean back stoping in a blind stope, but starting from the corner of a winze on the lower face of the block of ground.

Boys.—The number of boys used per two machines was, as usual, five, and this number is necessary to get the best work from the machines, and to reduce the number would, in my opinion, increase the cost per foot. As it is best under the circumstances to use five boys per two machines, there is no reason why the machines should not be made a little heavier than that demanded by the rules under which the competition was held, and so enable them to stand up to their work better and last longer, and thus reduce the cost of maintenance.

Maintenance of Machines.—The maintenance costs show great diversity, ranging from 1·6d. to 3·3d., including depreciation of machines. This is excessive, as spares have been sacrificed at times with the idea of getting greater footage. In ordinary work the spares cost should come below the smaller figure, for some of the above waste will be cut out, and known improvements that will be and are now being effected will lower the costs.

Explosives.—The amount of explosives used compares very well with ordinary work, coming out at over three fathoms per case. I had a mixture of gelatine and gelignite tried, and found that it was cheaper than gelatine alone, but with holes drilled of the diameter suggested hereafter in the 'Recommendations on size of bits.' Gelignite would be best for the ordinary run of miners, although good miners might make gelatine pay by increasing the burden on the holes. The best explosive to use depends on three things—the miner, the depth of the hole, and the diameter of the hole at the bottom.

Skill.—The mining work done during the competition leaves room for improvement. With the four machines left in the competition the average footage required to break a fathom of ground was 51·9 ft. I think it should have been kept down, to, say, 45 ft.' and with the Siskol to 42 ft. per fathom. The reason that it has taken a greater footage than I deem necessary is due to the low average length of hole drilled. On some of the runs good work has been done, whilst at other times the work was unsatisfactory. During my time the miners have been what any manager would class as good men, but the hope of winning prizes was more or less dead, as the competition had lasted too long and the men had lost enthusiasm. In my opinion it would have been much better to have had monthly prizes, and I think in this respect the fathomage bonus system, introduced in October, cheapened costs somewhat. The length of hole drilled on the average has been very low for all machines when taking into account the capabilities of the machines, and the size of bits in use. This in a great measure is due to the shape of the bits used, as there has been too great a clearance and too little metal in the bits. The steel used stood up to its work very well, but I think deeper holes would have been drilled with a better shaped bit.

Drills.—In my opinion the shape of the bit is a very important matter, and it is not always the best steel, as steel, that is the best for cutting rock, as the gauge wears off the drills very quickly, because there is too little metal provided for the wear. Large machine drills of ordinary steel can run with only 3-16 in. difference or gauge of bits. A set of 5 drills can start with a 2 in. bit with a finishing bit of 1¼ in., depth of hole 8 ft. 6 in., and 1¼ in. cartridge will go to the bottom. For the small machines run in the competition ⅛ in. should have been sufficient difference in gauge. All the competitors used too large a guage on the drills, starting with a 1[illegible] in. bit where they should not have gone above 1½ in. bit for a starter. With this gauge five drills could have been used, and still finish large enough to meet the condition of 15-16 in. at the bottom of a hole 6 ft. deep. Many feet of drilling are being lost for the sake of paying a little more in the drill sharpening shop. This shop is one of the main points to commence increasing the efficiency of rock drilling machines, then the machine fitter, then the mine captain and shift bosses, and, lastly, the miner, instead of commencing with the miner. If these four points are properly looked after, small machines should surpass hammer boys easily, and all machines do much better work.

Machine Handling.—The ease of handling these machines can be judged when a boy could, without disconnecting the hose from his machine, lift it off the arm of one bar, carry it away, place it on the arm of another bar ready rigged, and be ready to drill in from 15 to 20 minutes.

Recommendations—I would recommend the lightest reciprocating machines to weigh 120 to 130 lbs. The extra weight is wanted in feed screw, longer cradles, slides and cylinder liner. (Only the Holman 2½ in. has a liner.)

The same size bars and fittings will do for stoping widths up to 4 ft. A heavier machine—say 150 lbs. could be used, if desired, in stopes between 3 ft. and 4 ft., but to take the same fittings as the lighter machines:

In stopes over 4 ft., a machine weighing 180 to 200 lbs. would be heavy enough—but not too heavy for two-hole bench work—and of sufficient diameter (3 in. or 3½ in.) for any of our work. The standard bar for these machines would be, say, 3½ in. diameter.

The bits should be parallel at the sides and flat for about an inch from the cutting edge. The sizes for the smaller machines should be :—

No.	Size of bit.	Size and shape of steel.		Length.
1	1⅞ in.	1½ in.	Cruciform.	2 ft. 0 in.
2	1 11-16 in.	1½ in.	,,	3 ft. 3 in.
3	1½ in.	1 in.	Octagon.	4 ft. 9 in. (1¼ in. grooved.)
4	1/5-16 in.	⅞ in.	,,	6 ft. 3 in.
5	1⅛ in.	⅞ in.	,,	7 ft. 9 in.

1 in. diameter explosives to be used.

Large Machines.—

No.	Size of bit.	Alternate.	Size and shape of steel.		Length.
1	2 in.	2¼ in.	1¾ in.	Cruciform.	2 ft. 6 in.*
2	1 13-16 in.	2 in.	1½ in.		4 ft. 0 in.
3	1⅝ in.	1¾ in.	1¼ in.	Octagon.	5 ft. 9 in.
4	1 7-16 in.	1⅝ in.	1⅛ in.	,,	7 ft. 6 in.
5	1¼ in.	1¼ in.	1 in.		9 ft. 3 in.

1¼ in. diameter explosives to be used.

* Shanks included."

—*Mining Journal*, Feb. 18th, 1911, p. 177. (A. R.)

PROSPECTING FOR SILVER.—"Silver, once called a precious metal, is now an ordinary commercial substance, whose intrinsic value is so low, about 20 cents per ounce, that it is not sought so eagerly as formerly.

It would be folly on the part of the prospector to pass it by simply because it is so common a substance, for there are possibilities that it may be in large quantities or associated with other metals which will enhance its value. While silver is found occasionally in metallic form, it is more frequently found associated with other metal minerals, for instance, copper, lead, and gold. When native it resembles the silver in coin, although possibly a little brighter or darker in colour. It can readily be distinguished in this form, but if there is doubt it can be melted with a blow-pipe on charcoal, or if that useful little instrument is not available it will dissolve in nitric acid. If a little salt be added to a silver nitrate solution white silver chloride will be precipitated. If a few drops of hydrochloric acid be added to a silver nitrate solution silver chloride will be precipitated, which is soluble in ammonia, while lead chloride is not. Silver chloride is insoluble in hot water, while lead chloride is soluble, but will precipitate on the solution becoming cold.

With these few simple tests the novice can distinguish lead from silver, although frequently the two are associated in nature; in fact it is a rather rare occurrence to find lead sulphide free from silver.

The greater part of the silver produced in the United States is obtained when smelting lead and copper ores.

The greater part of the silver produced in Mexico comes from lead-silver ores. These ores also carry some gold, which oftentimes is sufficient to pay for their treatment, although not in such quantity that they could be classed as gold ores.

The greater part of the silver produced in Ontario, Canada, is almost native, although associated with cobalt and nickel minerals.

The first large silver deposit found in this country was the celebrated Comstock lode, in Nevada. The black sulphide of silver was at first thrown away by those who were washing for gold, and even broken up to obtain the free gold in the lumps. As soon, however, as an assay disclosed its nature mining commenced in earnest. The gangue in this deposit was mostly quartz and the vein has been worked to a depth of 3,100 feet, making it evident that the solutions forming the vein came from great depths. Argentite, or silver sulphide, contains 87·1% of silver and 12·9% of sulphur. It occurs in cubes and octahedrons, massive; as scales, coatings, and tree-like markings. It has a blackish lead-gray colour. Its streak on porcelain has a lead-gray colour and a shining metallic luster. It is partly soluble in nitric acid, and when in solution will silver-plate a copper strip. The mineral is as readily cut as lead; will flatten under the hammer, and is easily melted.

Other silver deposits were found in Nevada, although they were not remarkable, and it was not until 1878 that any bonanza silver deposits attracted attention. These were at Leadville, Colo., where the lead sulphides and carbonates were found in limestone adjacent to porphyry. In places very rich silver ore was found and often associated with it was gold. Closely following this were the Butte, Mont., deposits, which were in granite that had been cut by intrusive dykes, forming fissures that were afterwards filled by solutions carrying copper sulphides, gold, and silver. The copper sulphides were weathered, leaving the gold and silver in great masses near the surface. The gangue in this instance is quartz, indicating that the mineral solutions came from below.

Following this discovery came the Mollie Gibson and Aspen silver deposits in Colorado. These mines were in limestone adjacent to porphyry, and like the Leadville mines, were contact deposits. Probably the next most noted silver deposit was found by a burro in the Coner d'Alene district at Wardner, Idaho. The silver-bearing galena seems to be as rich in the Bunker Hill and Sullivan Mine at a depth of 2,100 feet, as at the surface. The mine is said to have paid over $7,000,000 in dividends since its discovery in 1885. It is at present the richest silver mine in the United States, and other silver-lead deposits are being discovered in the same district almost every year. The deposits are in a mineralized zone, having quartzite foot wall and an impregnated hanging wall of brecciated quartzite. The rocks in this section are quartzite and schists that have been folded and otherwise distorted.

In the Colbalt district in Canada there are some exceedingly rich silver veins in diabase. The outcrop of these veins is coloured pink by cobalt bloom, and if nickel be present a greenish tinge is observed. The silver in this district is almost native and is associated with calcite gangue showing that the ascending solutions carrying silver were met by descending solutions carrying carbonate of lime.

While silver is found in the older formations in quartz veins, and in the Cobalt district with calcite gangue, the largest deposits are found in limestone adjacent to porphyry. It seems remarkable at first that this should be the case until the same phenomenon is noticed in the case of copper sulphides and galena,

Both copper and galena when in sulphide form are associated almost invariably with silver. If they become weathered or taken in solution they leave the silver to be absorbed by some other solution, and this travelling along crevices in limestone eventually finds some mineral or solution which precipitates the silver.

For example, silver sulphate in solution, could be precipated by a salt solution forming insoluble silver chloride, known as horn silver or cerargyrite. Large deposits of this mineral are rare, but it is found associated with other silver minerals. It is soluble in ammonia, but not in nitric acid. Being soft and waxy it is readily melted, and will, when wet and placed on zinc, turn black, swell up, and show metallic luster if pressed with the point of a knife. It is usually found near the surface, but also in some depth in deposits formed from other minerals, being oxidized and taken into solution. Horn silver contains 75% silver and 25% chlorine. It is whitish with other shades of colour at times.

Should the silver be in solution and meet another solution containing antimony, ruby silver would be formed. Ruby silver is a black mineral with a dark red tinge. It generally looks massive, but under the glass it is seen to be finely crystallized. It gives a dark red streak on porcelain; is easily melted, giving off sulphur fumes, and is soluble in nitric acid. It might be mistaken for proustite or light red silver, but to the prospector this makes little difference. Light red silver contains sulphur and arsenic instead of sulphur and antimony. It is soft, gives a bright red streak and has a bright red colour. It is decomposed by nitric acid and melts easily. Both minerals are associated and at times carry considerable gold. Several other silver minerals are known, but usually are associated with other metals or nonmetals to such an extent as to be unrecognizable, although at times they can be determined by tests.

Limestone is generally bluish-gray or drab, but cannot always be recognised with the eye; however, a drop of acid will cause it to froth. Silicious limestone will not effervesce, and in all probability silver deposits will not be found in it. Porphyry weathers more or less at the surface unless highly silicious, and is generally white or gray with crystals of some other minerals prominent in a fine mass. It is an intrusive rock that often forms great dykes. It is presumed that solutions bearing minerals accompany the intrusion, and these, finding their way into limestone, deposit their mineral constituents. Another reasonable supposition is that the mineral in porphyry weathers and is carried down by surface waters and is then deposited."—*Mines and Minerals*, Dec., 1910, p. 289. (A.R.)

PREVENTION OF EXPLOSIONS.—"In the course of an interview, Professor Cadman, of the Birmingham University, said that 'in a modern mine you have all the main roads receiving deposits of the very finest coal dust, which, when intimately mixed with the air, is highly explosive. The whole process of mining nowadays tends to produce this dust in large quantities. At present the Government is devoting considerable attention to mine rescue work, but we must go further back than that. We want to devote much more attention to the prevention of explosions, and, it having been absolutely established that coal dust alone can cause an explosion, the obvious remedy is either to remove the dust or render it innocuous. Removal is totally impossible, and, turning to the alternative, we find that the coal-owners are exerting every effort to discover a remedy. During the past two years they have spent £10,000 in experiments to this end. Mr. W. E. Garforth has been acting as chairman and Dr. Wheeler as scientist to a commission appointed by the Mining Association of Great Britain, which has already done admirable experimental work. Their final report is awaited with keen interest. At present their suggested remedy is the use of stone dust, by the introduction of which the explosive coal dust is rendered both inert and innocuous. I only know of one colliery where this remedy has yet been tried, even as an experiment. I feel convinced that it will prove the ultimate solution of the terrible problem, but, if it does, it can never be successfully applied until every one engaged in a mine has realised that coal dust is the chief source of danger."—*Science and Art of Mining*, Jan. 21, 1911, p. 265. (W. R. D.)

VACUUM CLEANERS.—"A Wigan gentleman, who holds a mining appointment at Delagua, Colorado, U.S.A., sends a cutting from the *Denver Post*, which makes an interesting contribution to the present discussion on the coal dust question. The journal says that within the next week or ten days the experiment of removing dust from a coal mine with a vacuum cleaner, just as carpets and tapestries are cleaned in a private dwelling-house, will be tried by the Victor-American Fuel Company in its coal mines at Delagua. Two suction machines of special design, adapted to the peculiar conditions existing in a mine, are being built, and before they are shipped to Delagua they will be tried in the presence of experts and representatives of the Victor Fuel Company In this advance experiment the machines will be turned on a pile of slack coal to discover if their suction power is great enough to draw up coal refuse at that degree of coarseness. As they will be used in the coal mines, the machines will consist of large fans working in tubes, from which radiate vacuum devices fashioned after those used in hotels and residences, but of much larger size. These vacuum machines will be taken on trolley cars to the extreme limit of an entry and stationed at a suitable distance from the place of operation. Two men, each holding one of the 'suction cleaners,' will then go over that part of the mine, drawing the dust from top, ribs, and roadway through the hose back into the car. When a car is full it will be taken to the outside of the mine and dumped. Expert mechanics who have studied the matter estimate that with one such device, manned by two operators in addition to the trolley pilot, a half-mile of entries could be cleaned in a night. 'With two of these machines we could have our sixteen miles of entries made as clean as the floor of my office twice a month,' declared Senator Frank E. Cove, chief attorney of the Victor Company. 'If these machines work successfully,' he continued, 'there isn't a coal mine in the State of Colorado that ought not to have a half-dozen of them.' The cost of one of the machines being built for the Victor Company is $500."—*The Science and Art of Mining*, Jan. 21, 1911, p. 266. (W. R. D.)

REINFORCED CONCRETE IN MINES.—"The use of reinforced concrete in mines is of comparatively recent date. The difficulties in placing concrete and the lack of authoritative figures and estimates for the strengths needed kept this construction out of mines, according to a writer in *Beton u. Eisen*. In 1898 the Wiemelshausen mine was the first one to timber its ventilating shafts with concrete. The reinforced concrete is especially well adapted for ventilator funnels, as the concrete does not transmit the mine noises as do the iron ventilators. Where earth pressure is very great reinforced concrete is far ahead of any other construction for timbering. To obtain final perfection, a strong false work is of necessity, as this has to sustain and resist all the forces till the concrete is hard and firm enough to take up the load. False work of iron construction, made of angle irons and I-beams is by far the best in spite of its greater cost. Lately it has been tried to use ready-made reinforced concrete temporary work in place of the iron. In the Ruhr district where the pressure is enormous and the earth shifts a good deal, the space between the outer mountain wall and the inner shaft wall was tamped in concrete reinforced by iron and old rails. There is a rather interesting construction of a flushing chamber which is to store the material used for flushing the mine, and is in form of a bin. It consists of a storage space with funnel shaped exhaust, and the flushing chamber with washing plant. The chamber is 13 ft. high and topped by a concrete dome. To relieve the dome of pressure and at the same time get greater distribution of load, steel rails were built into walls for reinforcements. The temporary timbering of heavy oak timber connected by iron bars was removed in circular rings so that never more than 4 feet of shaft was without timbering. The thickness of the walls is 12 in. and is reinforced by double vertical and horizontal 0·3 in. round iron rods, the meshes of the network formed being 40 sq. in. Numerous iron loops prevent breaking of reinforcement towards shaft side. The entire timbering is covered with a coat of water-proof mortar. A second interesting piece of work is the completion of supply space in the Rheinelbe mine. On account of the poor condition of the earth (brittle slate) reinforced concrete had to be used and the shaft had to be enclosed all round as the pressure was exerted from all sides. Even the temporary false work had to be made extra strong. The supports and uprights were let into the ground to depth of 12 in and connected with the upper ceiling beams by clamps and pins. To prevent earth from pressing against the walls of mine, separate timbers and supports were used and a double temporary timber wall constructed. When filling concrete the timbering was removed for a distance of 4 ft. and tamped. Iron reinforcements were led into the roof and to a short distance into the sides of the mountain, thus insuring great strength. Between the inside chamber wall and outside shaft wall, a space 8 in. wide was left, so that in case of possible settling of shaft the chamber would not be injured. Another construction was that of a small sub-station platform, where only vertical pressure needed to be figured, and reinforcements were made accordingly."—*Mining World*, Jan. 14th, 1911, p. 62. (A. R.)

MISCELLANEOUS.

Products of Crude Petroleum.—"There are at present nine products, or classes of products, obtained from petroleum. The names of these are given in the accompanying tabulation :—

Kind of product.	Per cent. yield.		Amount Bbl.
1. Kerosene	20·50	...	15,000,000
2. Lubricating oils of all kinds, including greases	10·00	...	7,000,000
3. Naphthas, all grades ..	15·00	...	11,250,000
4. Gas oil, used for enriching water gas in large cities	30·00	...	22,000,000
5. Paraffin wax	1·50	...	1,125,000
6. Roofing pitch	2·50	...	1,875,000
7. Paving pitch and road-making oils	2·00	...	1,500,000 tons.
8. Coke...	3·00	..	300,000 bbl.
9. Fuel oil	14·00	...	1,050,000
	98·50		
Loss in manufacturing ...	1.50		
Total...	100·00		

together with the yield of each class in percentages and the total production in the United States for 1909, stated in barrels."—*Journal of the Society of Chemical Industry.—Australian Mining and Engineering Review*, Dec. 5, 1910, p. 123. (A. R.)

Liquid Fuel for Aeronautics.—Aviation is only made possible with the aid of petroleum products, so that it is largely in the control of the petroleum industrialists, who should endeavour still further to improve their products. Motor fuel should be chemically pure, of the lowest possible specific gravity, of the highest possible heat productivity, free of constituents with a high boiling point, and should not deposit any soot in the mechanism. The higher the heating power the smaller the cylinder of the motor. The specific gravity and heat productivity are inter-related. The lower the former the higher the latter, and the aeromobile can travel further in proportion as its fuel excels in these respects. The new petroleum product, aerobenzine, is of exceptional value. Its specific gravity should be 0·620 to 0·590; or alternatively it should boil with the heat of the hand. Its specific gravity must be under that of benzine, its heat productivity over that of benzine, and should be distinguished by these lightest hydro-carbons, which in the process of producing benzine evaporate and are lost. The lower its specific gravity the purer it is; and the higher its heat productivity the more power it will develop in the motor cylinders. The use of aerobenzine will reduce the weight of the motors, simplify their construction, and lessen their cost.

Aerobenzine must be kept in thick-walled hermetically-sealed vessels that can stand great pressure. It may be used for refrigeration and illumination. It is considered that in the summer the petroleum loses its aerobenzine and part of its benzine by evaporatioon n the way between the well and the factory, so that condensers and refrigerators should be constructed near the derricks. It can be seen that valuable material is now being lost from the light Maikop oil in the open reservoirs there. Refrigeration should be employed in the process.

A remarkable recommendation for aerobenzine besides its high market value to the producers is that its production constitutes a safeguard against fire; for the gaseous hydro-carbons, usually lost in Russian distilleries, issuing from the factory apparatus and mixing with the atmosphere make an explosive mixture easily ignited, whereas their capture would obviate this risk.—E. De Hautpick.—*The Mining Journal*, Jan. 21, 1911, p. 64. (R. A.)

First Aid to Injured in Mining Accidents.—"Sir H. H. Cunynghame, K.C.B., Under Secretary at the Home Office, in response to an enquiry, and as the result of visits he has paid to a considerable number of mines and quarries, has drawn up the following recommendations in connection with mining accidents.

The form of first-aid box for the treatment of simple cuts which I would suggest is as follows :—

For five men the size of the box is $5 \times 3\frac{3}{4} \times 1\frac{1}{2}$ in.; made of tin, with a lid or cover. It contains a 2 oz. bottle labelled "poison," with a cork which has been dipped in melted paraffin wax. Another bottle labelled "poison," 2 in. high, by $\frac{5}{8}$ in. in diameter, contains a dozen pellets each about $\frac{3}{8}$ in. diameter, containing salt, one-third of a grain of perchloride of mercury and any substance, such as a trace of gum, to bind the material together. In fact, the perchloride of mercury may be made into soluble pills or pellets by any practicable method.

The box also contains a paper package, $1\frac{1}{2}$ in. × 2 in. × $3\frac{1}{2}$ in., of tufts of blue wool which have been rendered antiseptic by means of perchloride of mercury and zinc in the usual manner. The box further contains a piece of pink boric lint, 6 in. × 8 in., folded in four, and put into a separate envelope just the size of the box, six or eight safety pins, and a small pair of scissors.

In a small tin pill-box, $1\frac{1}{4}$ in. in diameter, is contained a roll of adhesive indiarubber plaster, 2 yards long by $\frac{1}{4}$ in. wide, of the kind usually employed by surgeons. Finally, there are about 12 yards of good tape 1 in. wide.

The method of use is as follows :—First put one of the mercury pills into the 2 oz. bottle, fill up with water, and shake till dissolved. This makes a solution of strength about 1 in 3,000. With this wash the cut well. The salt will have the effect of preventing the solution from smarting.

It is an interesting fact that if even a little baby's wound is washed with weak salt and water, the child feels no smart and will not shrink. The smarting caused by the use of ordinary water is due, I believe, to the fact that the non-saline water passes by endosmosis into the tissue and swells it, whereas salted water or saliva, being more akin to the liquids of the body, has no such effect. This is the reason why a cut does not smart if sucked, as it does if washed with water. The amount of salt necessary to add to water in order to produce a liquid of the same density as blood-serum is 0·85%.

As soon as the cut has been washed it should be dried with another tuft of blue wool, and both tufts thrown away. On no account should a handkerchief or any other rag be brought near the cut. If the cut yawns, its edges may be brought together and secured by a bit of the sticky adhesive plaster (electricians often use a bit of rubber tape for the purpose). This tape will adhere upon a cut if the blood be wiped off, and the tape quickly put on and held for a little time. Once it has stuck, wet on the top will not affect it. Then a piece of boric lint should be cut and put on the wound and fixed in place with two or three bits of adhesive plaster. Finally, the place should be bound up with a bit of tape and secured by a safety-pin, and then the man may handle his pick again. In two days the wound should be examined and re-dressed; but most small cuts will be well in that time. The great thing to remember is that everything—tools, clothes, handkerchiefs and skin, are all full of microbes, and cause wounds to fester. The object to be obtained is to poison the microbes and then shut up the wound in an anti-microbe dressing.

These cases can be got from Mr. Jacobson, the agent for the Draegar rescue apparatus, 70, Shoe Lane, London, E.C.

The above quantities of contents of the box can, of course, be increased. Thus, for 100 men a box containing about ten times the above-mentioned quantities would be sufficient.

The box is also good for domestic use or for factories, stables or other places. For home use, if perchloride of mercury is considered dangerous, pellets of permanganate of potash may be used—but it is not so effective as perchloride of mercury."—*Iron and Coal Trades Review*, Feb. 17, 1911, p. 243. (A. R.)

TRANSVAAL TOBACCO. — Three bundles of "bright" tobacco leaves in good condition were submitted in February, 1910, to the Imperial Institute, London, by the Department of Agriculture, Transvaal, and were found on analysis to give the following results:—

	Per cent.
Moisture	6·9
Nicotine	1·2
Total Nitrogen	1·6
Ash	7·5

The ash on analysis was found to consist chiefly of:—

	Per cent.
K_2O	27·87
Na_2O	0·27
CaO	21·98
MgO	10·69
SO_3	2.35
Cl	0.17

Bulletin of the Imperial Institute, viii, 4, 404, 1910. (J. A. W.)

A NEW CAPE THERMAL CHALYBEATE SPRING. In 1906 boring for oil was started in the neighbourhood of Port Elizabeth, but on the 29th May, 1909, at a depth of 3,400 ft. a spring of hot water was struck. Boring was stopped at 3,460 ft. and the spring was found to yield 250,000 gallons per diem at a temperature of 128·5° F., the chemical composition being:

	gr. per gal.
Sodium Chloride, $NaCl$.	25·71
Magnesium Sulphate, $MgSO_4$ ..	3 07
Do. Carbonate, $MgCO_3$ ..	2·10
Calcium Carbonate, $CaCO_3$ ...	2·06
Iron Bicarbonate . .	1·66
Alumina, Al_2O_3	0·02
Silica, SiO_2	1·60

Gases in solution, analyzed by Sir James Dewar, were found to consist of:

Carbon Dioxide, CO_2	3·43
Oxygen, O_2	11·54
N_2, H_2, He., Ne.	85·03

In 1,000,000 parts of gas from Zwartkops the hydrogen, helium and neon are 741 parts, while at King's Well, Bath, England, they form 1,516 parts of the gas in solution. The amount of ferrous carbonate at first seemed to vary from 0·8 to 3·78 gr. per gallon, but now seems to remain constant at 1·7 gr. per gallon. H_2S has also been noticed, but other springs were tapped in the course of the work, some of which reach the surface outside the casing, and it seems most probable that the gas in question comes from one of these. As a table water when charged with CO_2 the water of the spring is very palatable, and baths have been erected so as to take advantage of the medicinal qualities of the waters.—*The South African Journal of Science*, Vol. VII., 5, 202, March 1911. (J. A. W.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

Lafar, Franz. Technical Mycology. Vol. 2. Eumycetic Fermentation. 8vo, pp. 568. *C. Griffin.* Net 24s.

Hill, Claude W. Electric Crane Construction. 8vo, pp. 334. *C. Griffin.* Net 25s.

Campbell, M. Contributions to Economic Geology. Part 2, Mineral Fuels. (U. S. Geological Survey Bulletin 381), 24 Plates. 8vo. pp. 559. *W. Wesley.* Net 10s.

Duffield, W. G. The Effect of Pressure Upon Arc Spectra. No. 3, Silver, 4,000 to 4,600. No. 4,600. No. 4, Gold. With 4 Plates. 4to, sd. *Dulau.* Net. 2s. 6d.

Hayes, C. and Lindgren, W. Contributions to Economic Geology. Part 1, Metal- and Non Metals, except Fuel. (U. S. Geol. Survey Bulletin 430). 14 Plates and Engravings. 8vo, pp. 653. *W. Wesley.* Net 6s.

Rice, G. The Explosibility of Coal Dust. (U. S. Geol. Survey Bulletin 433). 16 Plates and Engravings. 8vo, pp. 234. *W. Wesley.* Net 4s.

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Sherman, H. C. Chemistry of Food and Nutrition. Cr 8vo. *Macmillan.* Net 6s. 6d.

Tutton, A. E. H. Crystals. (The Internal Scientific Series). Cr 8vo, pp. 312: *K. Paul.* 5s.

Moor, C. G. and Partridge, William. Aids to Bacteriology. 2nd Edition. (The Student's Series). 12mo. *Baillierc.* Net 3s. 6d., sewed Net 3s.

Abstracts of Patent Applications.

(C.) 22/11. William Watts Bonson. Process for dry ore separation. 13.1.11.

This application relates to a process of dry ore separation, which consists of delivering the mass to a moving surface with air pressure thence through, then forming a bed of ore (mineral) which is maintained while the varying percentage of mineral is being delivered on this bed and drawn off as collected.

As a method of achieving this process, an appartus is described consisting of a reciprocating table, floored with a fine mesh to allow the air to be distributed and passed through the mass and provided with diagonal slots over which the gangue is passed while the mineral is led towards a series of gates in which the ore bed referred to is formed by retardation.

(C.) 23/11. William Watts Bonson. Improvements in dry ore separators. 13.1.11.

This application should be taken in conjunction with No. 22/11, by the same applicant. The former application is for the process of maintaining an ore bed in the dry separation of ore from gangue, whereas the present application relates to a particular device based on the same principle.

The device similarly consists of a reciprocating table, floored with fine mesh fabric to distribute air pressure from beneath and providing parallel diagonal slots, super-imposed at an angle; the upper series to guide the gangue to the tail of the table and the lower series to guide the ore to · longitudinal division, consisting of gates wherein is maintained an ore bed from the bottom of which the pure concentrated ore is drawn at the same rate that it is collected, and thereby maintaining an efficient separation.

(C.) 31/11. Eduardo Jose Maria Madero. Novel rotary engine. 19.1.11.

This relates to a form of rotary engine to be driven by a working fluid. It consists essentially of a plug or piston running in a circular path and transferring its motion to the driving shaft by an arm. A system of levers operate gates which divide the circular path of the piston in two and also valves which serve to admit and exhaust the working fluid.

(C.) 35/11. Jacobus Jan Willem Hendrik van der Toorn. A process for the extraction of gold from clay. 21.1.11.

This application relates to a process for extracting gold from clay, and is intended to prevent losses that occur when such material is treated with water.

The applicant claims that by kneeding and heating this clay, stone-like lumps are obtained, which may be broken up and the gold extracted.

(C.) 48/11. Frederick Ringrose. Improvements in automatic lubricators. 27.1.11.

This application refers to an automatic lubricator arranged in connection with a mixing tank for mixing graphite or other solid lubricant with liquid lubricant.

(C.) 66/11. Lewis Condict Bayles. Improvements in combined throttle valves and oilers. 3.2.11.

This is a device for admitting a small quantity of oil to a steam or air pipe at each time of opening the supply cock.

(C.) 169/10. Frederick Johnson Knight. Improvements in the preparation of sodium cyanide. 15.4.10.

This refers to the preparation of sodium cyanide by the fusion of a mixture of sodium ferro-cyanide, sodium carbonate and carbon.

(C.) 196/10. Tullio Bubola. Improvements in apparatus for electro-chemical treatment of ores. 28.4.10.

This patent deals with an apparatus for the electro-chemical treatment of ores :—

It consists in a combination of a vat (coated internally with non-conducting material), to contain the ore and treatment liquid, an electrode placed in the lower part of the vat, movable rod-like electrodes depending into the upper part of the vat, the mechanism required to move the upper electrodes, and a few other details consistent with the working of the apparatus. The electrode in the bottom of the vat constitues the anode and the movable electrodes in the upper part of the vat the cathodes, The solution is injected below the filter floor.

The combination further includes the means for passing the electric current through the ore and liquid in the vat, a precipitation box, and the means of circulating the solution.

(C.) 202/10. Bertram Gray. Improvements in pipe linings. 29.4.10.

This application consists of lining metal pipes with woven fabric treated with anti-corrosive substance, such as a bituminous composition.

(C.) 632/10. Bertie Gray Moorcroft Edwards. An improvement tappet. 24.12.10.

This application is for improvements in tappets for stamps consisting of two wedges arranged parallel with the stem.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 107/11. Josiah Phillips. Improvements in barrows. 17.2.18.

(P.) 109/11. John Denyer. Improvements in furnaces for heating drills and the like. 20.2.11

(P.) 110/11. George Warren (1), Joseph Gribble (2). Telescopic rock drill bar. 21.2.11.

(C.) 111/11. Jacob Wilkinson (1), Albert Edward Payne (2). An improved apparatus for regulating the pressure of the current in an electric circuit. 21.2.11.

(P.) 112/11. Alexander Purser. A new and improved device for regulating the flow of compressed air and the like. 22.2.11.

(P.) 113/11. George Harold Leeney. A new and improved roofing for houses and the like. 22.2.11.

(C.) 114/11. Sydney Russell Barrett. Improvements in and relating to pipe joints. 22.2.11.

(P.) 115/11. Charles William Dowsett. Device for maintaining an even flow of pulp from a centrifugal pump. 22.2.11.

(P.) 116/11. Robert Blyth (1), Thomas Connolly (2). Improvements in means for protecting miners from dust produced in rock drilling, also applicable to other occupations. 23.2.11.

(C.) 117/11. Henry Bell. Improvements in switches for electrical mine signalling apparatus. 23.2.11.

(P.) 118/11. G. H. Rotherham (1), Vernon D. J. Hoar (2). Segmental disk filter. 23.2.11.

(C.) 119/11. Jehu Berry. Improvements pertaining to winding and hauling engines. 24.2.11.

(C.) 120/11. Hermann Wolf. Improvements in the spraying matter in suspension and in spraying nozzles therefor. 24.2.11.

(C.) 121/11. Marcellin Mallet (1), Joseph Hubert Puthod (2). Improvements in or relating to blasting charges. 24.2.11.

(C.) 122/11. Henry Squarebrig Mackay. Process and apparatus for extracting metals from their ores. 24.2.11.

(P.) 123/11. Allan Thomas Cocking (1). Kynoch, Ltd. (2). Improvements in and relating to the manufacture of explosives. 24.2.11.

(P.) 127/11. Alexander Edward Kressmann (1), John Hart (2). Improvements applicable to acetylene lamps. 25.2.11.

(C.) 129/11. Fritz Deimel. An igniter acting through a spark produced by friction. 25.2.11.

(C.) 130/11. Carl Nitschke. Telescopic lifting gear. 27.2.11.

(P.) 131/11. John Law Aymard (1), Arthur Payne (2). A new and improved foot leg guard. 28.2.11.

(P.) 132/11. Bertram Poidevin Richardson. Improvements in pipe and similar couplings. 28.2.11.

(P.) 133/11. Lewis Watkins. Pulp or tailings separator. 28.2.11.

(P.) 135/11. William Jamieson (1), Andrew Francois Brink (2). Improvements in spraying nozzles. 2.3.11.

(C.) 138/11. Peter Mommertz (1), Gewerkschaft Deutscher Kaiser (2). Improvements in linings for pipes. 3.3.11.

(C.) 139/11. Peter Mommertz (1), Gewerkschaft Deutscher Kaiser (2). Improvements in linings for disintegrating machines. 3.3.11.

(P.) 142/11. Edmond Ernest Johnson. Improvements in extracting metals from their ores, and apparatus therefor. 1.3.11.

(C.) 143/11. Emil Schranz. Improvements in jigging machines for separating minerals. 7.3.11.

(C.) 144/11. Oscar Horace Schuller. Improvements in vibratory sifting machines. 7.3.11.

(P.) 145/11. George Rodd. An adjustable scraper. 7.3.11.

(P.) 147/11. Charles Henry Wilkins (1), James Wilshir (2). Improvements in rock drill chucks. 7.3.11.

(P.) 148/11. Geo. H. Rotherham (1), Vernon D. J. Hoar (2). Segmental revolving disc filter. 8.3.11.

(P.) 149/11. William York. A new and improved method, instrument, or machine for scooping or dredging dams, drains, etc. 8.3.11.

(P.) 150/11. Samuel Alford. Improvements in means for consuming smoke applicable to domestic fire grates, the furnaces of steam generators and the like. 9.3.11.

(C.) 151/11. Emanuel Rosenberg (1), John Sedgwick Peck (2). Improvements in braking induction motors. 10.3.11.

(P.) 152/11. Edward James Rule (1), James Ferguson (2). Improvements in chucks or tool holders for rock drilling machines. 10.3.11.

(C.) 153/11. Hermann Georg Spengel. Improvements in conveying apparatus. 10.3.11.

(P.) 155/11. Hermann Stadler. Improvements in sifting apparatus. 14.3.11.

(P.) 156/11. John Pitkethly Smith. Improvements in dust collecting attachments for rock-drilling machines. 15.3.11.

(P.) 157/11. Arthur Dixon Arnold. Improvements in shaft and haulage rollers. 15.3.11.

(P.) 158/11. Charles William Dowsett (1), Frank Gowyne Alfred Robert (2). Improvements in tube mill linings. 16.3.11.

(P.) 159/11. Martin Wagner. An improved apparatus for the spraying of solutions and the like. 16.3.11.

(P.) 160/11. John van Nostrand Dorr. Improvements in settling apparatus. 17.3.11.

(P.) 161/11. Richard Nicholson (1), Carl Davenport (2) Improvements in or relating to percussive rock drills and the like. 17.3.11.

(C.) 162/11. Victor Nightingall. Improvements in electric cooking appliances and water heaters. 17.3.11.

(C.) 163/11. Harry Pauling. Improved electric furnace for treating gases. 17.3.11.

(P.) 164/11. Robert Allen. Improvement in drill heating furnaces. 23.3.11.

(P.) 165/11. Alexander Bennett Ritchie. Improvements appertaining to the bearings of cam shafts. 23.3.11.

(P.) 166/11. Alexander Bennett Ritchie. Improvements in means for fixing collars and the like upon shafts. 23.3.11.

(P.) 167/11. Jacques Berthold Schlesinger. Improvements in rock drills. 23.3.11.

(P.) 168/11. Adolf Reichenberg. Pneumatoplane. 23.3.11.

(C.) 169/11. Stewart Woodford Young. Fume destroying composition. 24.3.11.

(C.) 171/11. Jacobus Matthias Lofthouse (1) and Emily Rosalie Booty (2). Improved carburetter for air gases. 24.3.11.

(C.) 172/11. Edmund Seal Donisthorpe. Improvements in or relating to apparatus for synchronising the working of machinery. 24.3.11.

(C.) 173/11. Stephen Ware (1) and Charles Ware (2). An improved rail joint and fish plate fastening therefor. 24.3.11.

(C.) 174/11. Edward Macaulay. Improved method of and apparatus for reducing or neutralising the noxious fumes resulting from the combustion of nitro-glycerine explosives. 24.3.11.

(C.) 175/11. John Joseph Keppel (1) and Samuel Alexander Lyttle (2). Improvements in wire strainers. 24.3.11.

(P.) 176/11. Hans Charles Behr. Auxiliary operating means for electric hoists. 25.3.11.

(P.) 177/11. Augustus Carron. An improved process for separating the oils from the pitch of tar. 27.3.11.

(P.) 178/11. Johannes Tobias Engelbrecht. Automats pump. 27.3.11.

(P.) 179/11. Frank Ondra. Apparatus for precipitation treatment of liquid. 27.3.11.

(P.) 180/11. Henry George Aulsebrook. Improvements in respirators. 27.3.11.

(P.) 181/11. William McFarlane (1) and Thomas Harrison (2). Improvements in jockeys or rope grips. 28.3.11.

(P.) 182/11. Frederick Pattersen (1), Thomas Evan James (2) and Williams Bradwin Varian. A device for the prevention of railway accidents through collision of trains; the antomatic stopping of opposing trains; and the ringing of alarm bells at level crossings or elsewhere. 28.3.11.

(P). 183/11. John Kenneth Addie. Rope jockey attachment. 30.3.11.

(C.) 184/11. Thomas Nogier. Apparatus for sterilizing water by means of the nltraviolet rays.

(C.) 185/11. Charles Francois (1) and Alfred Chapitel (2). Improvements in breaking machines. 31.3.11.

(C.(186/11. Clemens Graaff. Improvements in a method and an apparatus for extinguishing fires. 31.3.11.

(P.) 187/11. Hans Charles Behr. Improvements relating to pneumatic stamps. 31.3.11.

(P.) 188/11. George Kirby Chambers (1), Arthur Whitten Brown (2) and the British Westing-house Electric and Manufacturing Co., Ltd. (3). Improvements in devices for controlling hoisting and haulage mechanism. 31.3.11.

(P.) 189/11. Detmar Eggers. A process for the extraction of metal from slime. 4.4.11.

(P.) 190/11. William Worthy. An exhaust air system of damping the dust made by compressed air rock drills when drilling dry holes. 4.4.11.

(P.) 191/11. John Murdoch Skelley (1) and Rollo Bowman Ballantine (2). An improved apparatus for decreasing radiation from boilers and the like. 4.4.11.

(P.) 193/11. William Davies (1) and William Henry Hocking. Extension Bar. 4.4.11.

(P.) 194/11. James Catford. Improvements in ore concentrating apparatus. 5.4.11.

(P.) 195/11. James Esson (1) and Denis Frances Breslin (2). Improvement in respirators. 6.4.11.

(P.) 196/11. Alfred James Lake. Turbine wind motor. 6.4.11.

(P.) 198/11. Harry August Stockman. An improvement in the method of reinforcing concrete. 7.4.11.

(C.) 200/11. Matthew White (1) and James Henry White (2). Improvements in gas generating apparatus. 7.4.11.

(C.) 202/11. Louis Friedrichs. A safety device for valves, more particularly gascocks. 7 4.11.

(P.) 205/11. Richard Rabbich. An improved appliance for cleaning the surface and grooves in tram rails while cars are in motion. 8.4.11.

(C.) 206/11. William Hardie Cheyne (1) and George Richmond Newburn (2). Improvements in or connected with rapid action of steam hammers, more especially intended for use with tools for scaling boiler tubes. 8.4.11.

(P.) 207/11. Harry Mercer. Acid resisting pipe. 10.4.11.

(P.) 208/11. Thomas Graham Martyn. Improvements in or relating to apparatus for separating suspended solids from liquid or gases; mixed liquids from each other and gases from liquids. 11.4.11.

(C.) 209/11. Edward A. Blanton. Improvements in nut locks. 11.4.11.

(P.) 210/11. Junins Ford Cook. Device for separating ore. 11.4.11.

(P.) 213/11. John Murdoch Skelley (1), Rollo Bowman Ballantine (2) and Norman David Penny (3). An improved apparatus or means of preventing radiation of heat from steam pipes. 12.4.11.

(P.) 214/11. William Frederick Keatingstock. An improved fertiliser, insecticide and fungicide. 13.4.11.

(P.) 215/11. Charles Graham Stone. Improvements in stamp mills. 13.4.11.

(C.) 216/11. Oscar Clarence Beach (1) and Arthur Judson Hodge (2). Improvements in disc pulverizers. 13.4.11.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

ADAMSON, R., *l/o* Rhodesia; c/o W. M. Thomson, Esq., Earnscliffe, Hamilton, Scotland.

BALL, H. S., *l/o* Montreal, Canada; c/o Standard Bank of South Africa, 10, Clements Lane, London, E.C.

BAUMANN, M., *l/o* Shankurn Mine, Rhodesia; c/o R. R. Hollins, Esq., 25 Austin Friars, London, E.C.

BOLITHO, E. J., *l/o* Nigel; Tebekwe Mines, Selukwe, Rhodesia.

BRADING, T., *l/o* Gwanda; The Whittaker Mine, P. O. Kimberley Reefs, Mazoe, Rhodesia.

BRANFORD, E. W., *l/o* Lonely Mine, Rhodesia; Woodbridge, Suffolk, England.

CELLIER, J. S., *l/o* Nigel; P. O. Box 662, Johannesburg.

CHOMLEY, W. B., *l/o* Beira; Northern Mines, Lawlers, W. Australia.

COOMBE, M. H., *l/o* Randfontein; Govt. Gold Mining Areas (Modderfontein) Cons. Ltd., P. O. Box 28, Brakpan.

GILBERT, T. W., *l/o* Waterberg; New Heriot, G. M. Co. (Survey Dept.), P. O. Box 10, Cleveland.

GRAHAM, H. W., *l/o* London; P. O. Box 1,155, Johannesburg.

HOLMAN, B. W., *to* Simmer and Jack Proprietary Mines, P. O. Box 192, Germiston.

LEA, W., *l/o* Johannesburg; Cornelia Colliery, Viljoen's Drift, O.R.C.

LESLIE, H. M., *l/o* S. India; c/o G. Dickson, Esq., 2, Caledonian Mansions, Glasgow, W., Scotland.

MARQUARD, J. D., *l/o* Rose Deep, Ltd.; Inspector of Mines Office, Boksburg.

MARTYN, T. GRAHAM, *l/o* New Primrose G. M. Co.; The New Unified Mine, P. O. Box 5, Maraisburg.

PARRY, C. F., *l/o* Simmer and Jack Proprietary Mines; Sub-Nigel G. M. Co., Nigel.

PHILLIPS, F. D., *l/o* Germiston; Bwlch R. S. O., Breconshire, South Wales, Great Britain.

SCAER, V. E., *l/o* Bulawayo; Arcturus Mines, Salisbury, Rhodesia.

SEEAR, H. E., *l/o* Umsweswe; The Owl Mine, Gatooma Rhodesia.

STACPOOLE, A. R., *l/o* Johannesburg; The Durban Roodepoort Deep, Ltd., P. O. Box 110, Roodepoort.

STRANGE, E. H., *l/o* Denver; City Hotel, Victor, Colo., U. S. A.

TENNANT, A., *l/o* Roodepoort; South Randfontein Deep, Ltd., P. O. Box 176, Randfontein.

THE JOURNAL
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Vol. XI. MAY, 1911. No. 11.

Proceedings
AT
Ordinary General Meeting, May 20, 1911.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the South African School of Mines, on Saturday evening, May 20th, Dr. James Moir (President) in the chair. There were also present :—

55 Members: Messrs. W. R. Dowling, Tom Johnson, A. Richardson, A. Whitby, H. A. White, Prof. J. A. Wilkinson, James Littlejohn, A. F. Crosse, R G Bevington. A. McA. Johnston (Members of Council), S. Beaton, W. Beaver, A. J. Bowness, J. Chilton, A. A. Coaton, M. H. Coombe. D. E. Damant, J. M. Dixon, P. L. Edwards, R. Gascoyne, James Gray, B. C. Gullachsen, B. J Hastings, J. l'Ons, J. A. Jones, W. W. Lawrie. J. Lea, L. Marks, T. G. Martyn, Hans Meyer. P T. Morrisby, S. Newton, W. Nicklin, J. C. Phillips, J. B. Polglase, S. C. Quin, E. Roberts, G. A. Robertson, A. Salkinson, A. Schwarz, S. Shlom, G. Hildick Smith, A. L. Spoor, R. Stokes, J. A. Taylor, Prof. G. R. Thompson, O. Tonnesen, J. T. Triggs, J. Walker, J. P. Ward, H. Warren, F. W. Watson, John Watson and H. Wyndham.

18 Associates and Students: Dr. J. L. Aymard, G. J. V. Clarence, P. A. Cloete, C. A. Damant J. Davidson, W. J. N. Dunnachie, J. Gibson, A. King, H. Lomberg, F. J. Pooler, Alex. Purser, J. M. Robinson, A. G. Rusden, H. Rusden, W. Russell, R. Sawyer, H. Stadler, and S. A. Woolf.

12 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as printed in the April *Journal*, were confirmed.

NEW MEMBERS.

Messrs. W. Beaver and A. Salkinson were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that the candidates for membership had been unanimously elected, as follows :—

LAMB, JOHN, P. O. Box 217, Bulawayo, Rhodesia. Mining Engineer.
SMITH, ARTHUR LOUIS, Esperanza Gold Mine, Umsweswe, Rhodesia. Mine Manager.

The Secretary : Since the last meeting of the Society the following have been admitted by the Council :—

As Associates.—

CLOETE, PHILIP ALBERT, Ferreira Deep, Ltd., P. O. Box 1056, Johannesburg. Surveyor.
GOW, GEORGE AUBREY, Lebong Soelit, Benkoelen, Sumatra. Reduction Works Assistant.
IRWIN, R. W., Knights Deep, P. O. Box 143, Germiston. Amalgamator
POOLE, CLAUDE WILLIAM, Mercur Gold Mines Co., Mercur, Utah, U.S.A. Superintendent.

GENERAL BUSINESS.

The President : The first thing under general business is the election of scrutineers for our annual election next month.

On the motion of Mr. H. A. White the following gentlemen were appointed scrutineers :—

Messrs. S. Newton, W. Beaver, Harry Lea, F. G Guthrie, A. Thomas, W. M. Coulter, G. Melvill and G. Hildick Smith.

Mr. A. F. Crosse (*Past-President*) : I should like to propose a hearty vote of thanks to the Dinner Committee. I may say that although our Annual Dinner was not as numerously attended as I should like to have seen it—there were so many other meetings the same evening—yet the dinner was a great success. I think, as a dinner, it was one of the best we have had for many years.

The President : I am sure the Dinner Committee will much appreciate Mr. Crosse's remarks. It was very unfortunate certainly that the railway people had that accident which mixed up their arrangements and spoiled the evening for so many members coming from out of town.

SOME ASPECTS OF MINE VENTILATION AND TEMPERATURE ON THE WITWATERSRAND.

By J. Whitehouse, B.Sc., M.I.M.E. (*Associate*) and W. L. Wotherspoon

The ventilation of the mines of the Witwatersrand, more especially the deeper ones, has, within the last three years, received very careful consideration from those whose interests lie in this great industry. It has been realised that in the past very little attention has been paid to the health conditions underground, the whole question having been neglected, and as a result of this the death rate amongst underground workers has been very high.

Attention has already been drawn to the chief causes of disease amongst miners; namely, dust from machine drills, infective processes, and the gases resulting from the use of explosives. The object of this paper is not to dwell further upon these points or to explain how any one mine has been ventilated, but rather to view the subject from the standpoint of economics.

The mines of the Witwatersrand are year by year becoming deeper: hence new undertakings will be on a much larger scale than in the past, and owing to greater capital outlay and to the greater expense of handling rock, the efficiency of the underground workers must be increased. As we go deeper, however, the rock temperatures increase: hence the workings of the deeper levels will be considerably hotter than those of outcrop mines. This, instead of making for efficiency, will tend to reduce the miner's capacity for work and hence some means must be introduced to counteract this increase of temperature at depth.

Fig. I is a chart showing rock temperatures in deep level mines on the Rand. The plain circles represent observations made by Mr. H. F. Marriott some years ago,* and the black circles indicate the more recent observations by Mr. Wotherspoon. It is apparent from the chart that the estimated rate of mean temperature rise is now represented to be 1° F. for 253·9 ft. vertical depth, the temperature of the rock varying from 65° F. at 200 ft. to 98° Fah. at 7,000 ft. depth, whereas the former results indicated a rise of 1° F. for 208 ft. vertical depth. The favourable nature of Mr. Marriott's observations occasioned some surprise, and the authors therefore think an explanation for the still more favourable results recently obtained will be of interest. On examination of the former data it is apparent that the estimate of the rate of increase was based on the mean of two distinct types of observations or conditions: (1) those in mines having open workings; (2) those in boreholes or shafts during sinking, and although Mr. Marriott in his paper said "the experiments have shown that the natural ventilation of the mines decreases the rock temperature in the vicinity of the workings by from 5° to 10° F.," yet he did not give weight to that statement in plotting his chart and in estimating his temperature-gradient. In view of this and also the fact that facilities for obtaining deep level temperatures are now more abundant, it therefore follows that recent data should be more reliable. It is interesting to see from the chart that the line connecting three observations taken at the Cinderella Deep at depths of 1,000, 2,000 and 3,000 ft. respectively is exactly parallel to the main line of temperature rise, the only difference being that it falls somewhat below it, and this is due to the locality of the observations being in large pump chambers close to the shaft and favourably situated for cooling effect. The question of rock temperatures at depth is of considerable interest since the rate of increase varies very much in different parts of the world. For instance at the Dolcoath mine in Cornwall the rate of increase in virgin rock is about 1° F. in 60 ft., whereas at Hamstead and other deep mines near Birmingham the rate of increase is about 1° F. in 110 ft. and then we have the local rate of increase as already shewn, viz., 1° F. in 253·9 ft. If, however, mining is to be carried on at the great depths which we expect, even at the slow rate of increase—which by the way is exceedingly fortunate and makes the problem much easier—our temperatures will be very considerable.

With regard to the temperature of mine air it is equally important to consider the quantity of moisture it contains, for that decides the temperature at which men cease to work efficiently. The point has already been dealt with by Dr. Haldane, and one of the authors is indebted to him for information on the subject during his last visit to England, and it is proposed to repeat some of the points raised by him in a recent paper on this subject. The cooling down of the body of a man at work occurs partly by conduction and partly by conversion into latent heat by the evaporation of moisture. In a hot working place when a miner is at work, the increased skin circulation cannot get rid of the surplus heat by conduction and radiation, and in this case the loss by evaporation is increased by sweating, and when the external temperature exceeds the body temperature the whole of the heat is got rid of by evaporation. If the air temperature is below the body-temperature, heat loss by conduction is increased greatly if the air is moving and, of course, evaporation is also promoted if the air is

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* This *Journal*, Vol. VI., p. 339.

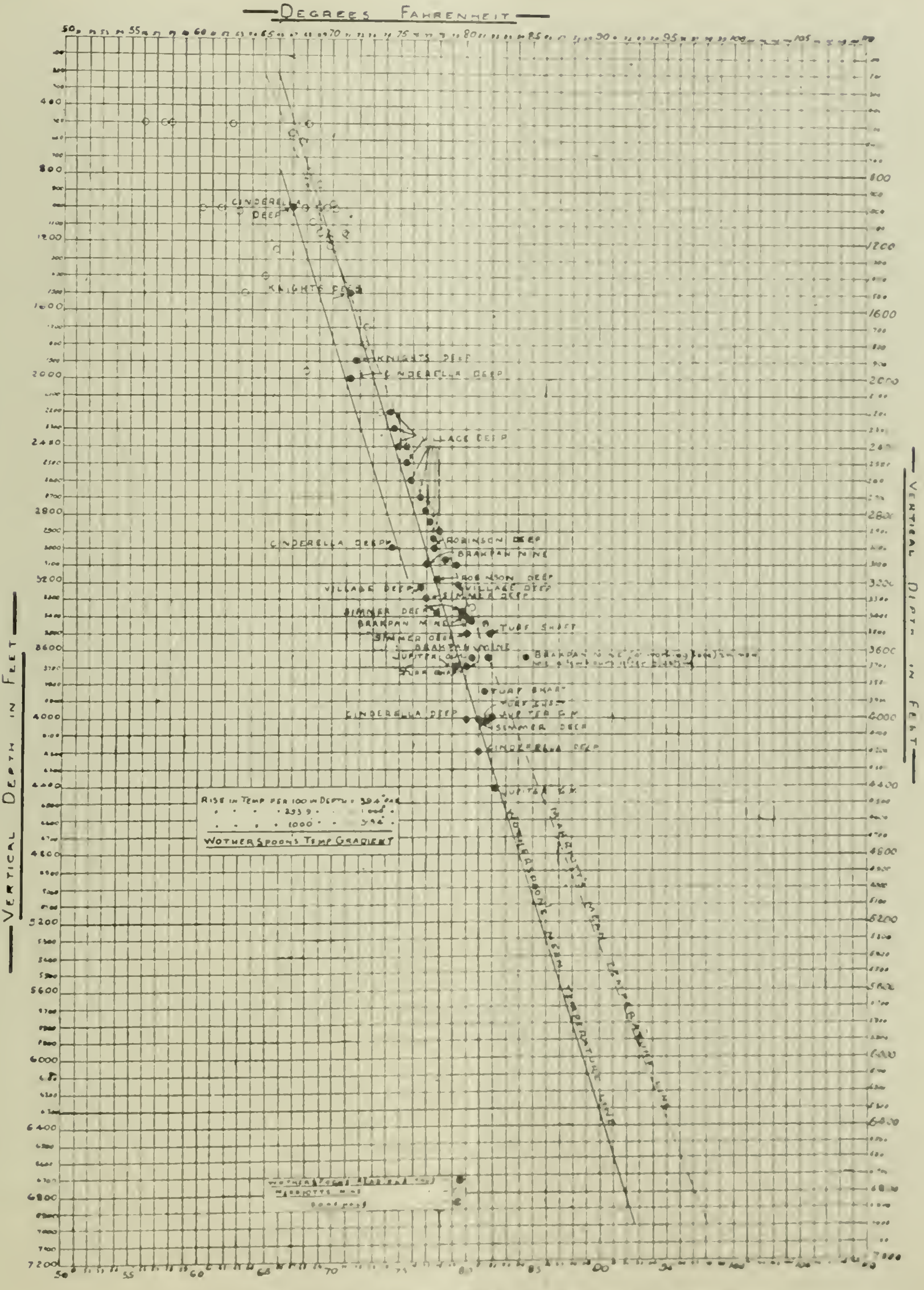


Fig. 1.—Chart showing the Geothermic Degree on the Witwatersrand.

in motion, provided that the dew-point is below the body temperature. Hence the velocity of the air is of considerable importance. Evaporation from the body can only occur if the dew point is below the body temperature, but even if the dew point were below the body temperature it does not follow that any cooling action would be produced by such evaporation for if the air were warmer than the body, it might communicate more heat than was lost by evaporation. The body during sweating might be compared to a wet bulb thermometer which indicates not the dew point but a temperature intermediate between the dew point and the actual temperature of the air and dependent on both. If the wet bulb temperature is below the body temperature heat is lost by the body, but otherwise there can be no loss of heat. Really the wet bulb temperature and the movement of the air are the only things that matter, and hence we may say that what we want in our workings are large volumes of air circulating at a good velocity, this air to be as free from moisture as possible. It would appear that where there is no current of air the temperature should never exceed 78° F., and where there is a good current it should not exceed 85° F. If the intake air takes up much moisture in the downcast shaft or main drives and crosscuts, it will be cooled or prevented from heating. Every grain of moisture added by evaporation to a cubic foot of air corresponds to a cooling effect to the extent of about 8° F. There is no immediate disadvantage in cooling the intake air by evaporation of moisture, for in spite of the increased saturation of the air the wet bulb temperature is slightly lowered. The secondary effects, however, must be disadvantageous in deep mines, for the cooler the intake air is kept the more rapidly will it tend to take up heat from the surrounding strata ; and if it is already highly charged with moisture, the range within which its temperature can be allowed to rise, without interference with the working capacity of the men, is much restricted. If, for instance, the air has already been saturated with moisture at 70°, its temperature cannot subsequently rise beyond about 96° even if no further moisture is added, without affecting working capacity ; whereas if no moisture had been added greater temperatures could be borne without great inconvenience.

It will be seen from the above remarks that our air currents must be sufficiently large in volume and have an ample velocity to cool the intake roads or drives so that the temperature of the air cannot be raised too much before reaching the faces, and also to prevent the air gathering moisture. In connection with this subject, the results obtained at the Village Deep Mine are interesting, and show a decided improvement on the starting up of the fan. Temperatures were taken before the fan started and were again taken after the fan had been running three months : the results were as follows :—

No. 1 *Shaft, Stations Nos.* 2 *to* 10 *inclusive.*—

Average temperature of air before	...	74·6° F.
,, ,, ,, after	...	66·7° F.
Difference ...	...	7·9° F.

The average drop in humidity was 9·3%.

No. 2 *Shaft, Stations Nos.* 1 *to* 13 *inclusive.*—

Average temperature of air before	...	76·6° F.
,, ,, ,, after	...	68·3° F.
Difference ...	...	8·3° F.

The average drop in humidity was 10·9%.

Tables A, B, C, D, are records of the observations taken underground at the Village Deep, and Table E shows surface readings. The first column shows the date and time, and it will be noticed that in each line we have an observation before and after the fan started. In the second column the reference number on the plan is given. In the succeeding columns are given the locality of observations, the depth from the surface, distance along the level from shaft, barometer readings, dry bulb and wet bulb readings, and the relative humidity.

The positions where these observations were made are shown on Figs. 2 and 3.

The following is an observation made at the beginning of the year in the upcast at fan chamber : —

Depth below surface 2,100 ft. ; height above sea level, 3,640 ft. ; surface temperature, 84° F.; mine temperature, 76·5° F.

Again the observations of rock temperatures are very interesting, and illustrate what has already been said concerning the reduction in temperature of the rock surface due to the passage of large volumes of air through the shafts.

Locality.	Date.	Time. p.m.	Dry Bulb $= t$.	Wet Bulb $= t_1$.	Difference $t - t_1$.	From t_1 Table 40 $= f_1$.	$t - t_1$ and Barometric Table 41 $= f_2$.	Pressure of Aqueous Vapour $f_1 - f_2 = f$.	Table 40 from T. $=$ F.	Humidity $\frac{f}{F} \times 100$ $= \%$.	Barometer.
No. 2 Level	5-1-11	3.15 to 3 27	73·67	71·96	1·71	7·81	·0163	·7647	·8291	92·2%	26 in.

EFFECT ON MINE TEMPERATURES OF INSTALLING FAN.

A.—PSYCHROMETRICAL OBSERVATIONS UNDERGROUND AT VILLAGE DEEP, LIMITED, JOHANNESBURG.

Date and Time of Observations.	Reference Number on Diagram.	Location of Observations.	Depth from Surface In feet.	Distance along Level from Shaft. In feet.	Barometric Readings.	Psychrometric Readings. Dry Bulb Fah.	Psychrometric Readings. Wet Bulb Fah.	Relative Humidity Saturation = 100.	REMARKS.
1st Feb., 1910, 10 a.m.	1	No. 1a Level, Station No. 2 Shaft	1938	—	26·30	73·0	72·3	97·0	3 Persons present
30th May, 1910, 10.50 a.m.					26·16	63·7	57·9	72·2	3 ,, ,,
1st Feb., 1910, 10.30 a.m.	2	No. 1a Level, Cross-cut ,, ,,	1938	190	26·30	73·4	72·7	97·0	3 ,, ,,
30th May, 1910, 10.30 a.m.					26·14	69·8	69·1	72·7	3 ,, ,,
1st Feb., 1910, 11 a.m.	3	No. 1a Level, West Boundary	1927	1125	26·30	74·1	73·4	97·0	3 ,, ,,
30th May, 1910, 10.25 a.m.					26·15	73·4	73·0	98·2	3 ,, ,,
1st Feb., 1910, 12.50 p.m.	4	No. 1 Level, Station No. 1 Shaft	1999	—	26·35	74·1	73·6	97·5	4 ,, ,,
—					—	—	—	—	
1st Feb., 1910, 11.35 a.m.	5	No. 1 Level, East Boundary	1987	1206	26·53	73·9	73·0	95·5	Worked out part of mine 4 persons present
30th May, 1910, 11.50 a.m.					26·38	71·6	65·8	74·6	3 Persons present
1st Feb., 1910, 12.30 p.m.	6	No. 1 Level, Cross-cut	1997	180	26·35	74·1	73·0	95·0	4 ,, ,,
—					—	—	—	—	
14th Jan., 1910, 12.20 p.m.	7	No. 2 Level, East Boundary	2090	842	26·38	75·2	74·1	95·1	5 ,, ,,
17th May, 1910, 10.30 a.m.					26·47	72·7	71·2	93·1	3 ,, ,,
14th Jan., 1910, 12.[illegible] p.m.	8	No. 2 Level, Station No. 1 Shaft	2098	—	26·35	74·8	73·6	93·0	2 Air hoists working here, hauling skips 8 persons present
17th May, 1910, 10.4[illegible] a.m.					26·50	59·1	56·5	84·6	12 Persons present
14th Jan., 1910, 12.[illegible] p.m.	9	At Fan Chamber	1959	700	26·28	72·3	71·3	95·0	25 Persons here, 20 of these working in chamber erecting fan, etc.
17th May, 1910, 11 a.m.					26·37	72·3	71·2	94·9	Fan in operation
14th Jan., 1910, 1 p.m.	10	No. 2 Level, Station No. 2 Shaft	2117	—	26·38	75·9	74·5	92·8	6 Persons present
17th May, 1910, 11.[illegible] a.m.					26·55	61·2	58·6	86·6	6 ,, ,,
19th Jan., 1910, 10.15 a.m.	11	No. 3 Level, Station No. 2 Shaft	2209	—	26·40	73·4	72·7	97·1	6 ,, ,, Air pump working here
17th May, 1910, 12.25 p.m.					26·57	61·2	58·3	85·2	3 ,, ,,
19th Jan., 1910, 10.20 a.m.	12	No. 3 Level, West Boundary	2203	183	26·40	75·2	74·5	97·0	3 ,, ,,
17th May, 1910, 11.40 a.m.					26·56	61·1	60·1	79·0	6 ,, ,,
19th Jan., 1910, 11.[illegible] a.m.	14	No. 3 Level, between Shafts	2186	1011	26·40	73·2	72·5	97·0	5 ,, ,,
17th May, 1910, 12.[illegible] p.m.					26·55	69·1	67·6	92·8	Door shut at No. 2 shaft, strong current when opened
19th Jan., 1910, 11 a.m.	15	No. 3 Level, Station No. 1 Shaft	2193	—	26·42	73·0	72·0	95·0	Air pump working here 5 persons present
17th May, 1910, 1 p.m.					26·56	61·2	58·3	85·2	Door open between shaft and drive to No. 2 shaft
19th Jan., 1910, 11.[illegible] a.m.	16	No. 3 Level, East Boundary	2185	1116	26·3[illegible]	75·0	74·7	98·0	3 Persons present
17th May, 1910, 1.1[illegible] p.m.					26·58	79·2	78·4	96·7	12 ,, ,,
19th Jan., 1910, 11.[illegible] a.m.	17	No. 4 Level, Station No. 1 Shaft	2288	—	26·50	74·[illegible]	72·1	89·6	8 ,, ,,
18th May, 1910, 10.[illegible] a.m.					26·66	61·0	60·1	80·7	4 ,, ,, Door open west of station
19th Jan., 1910, 1[illegible] a.m.	18	No. 4 Level, East Drive	2280	834	26·48	74·1	73·4	97·0	8 ,, ,,
18th May, 1910, 11.10 a.m.					26·5[illegible]	68·4	64·4	81·5	3 ,, ,, Door open east of station
19th Jan., 1910, [illegible]	19	No. 4 Level, between Shafts	2281	738	26·4[illegible]	74·8	73·8	95·3	1 ,, ,,
18th May, 1910, 10.40 a.m.					26·[illegible]	72·0	70·2	91·8	Door at No. 2 shaft shut 3 persons present
19th Jan., 1910, 12 noon.	20	No. 4 Level, Station No. 2 Shaft	2316	—	26·52	74·8	73·8	95·3	7 Persons present
18th May, 1910, 9.40 a.m.					26·62	67·6	59·0	81·8	9 ,, ,,
19th Jan., 1910, 12.[illegible] p.m.	21	No. 4 Level, West Boundary	2300	1112	26·46	75·2	74·5	97·0	3 ,, ,,
18th May, [illegible]					26·60	73·8	71·6	90·3	2 ,, ,, Door open west of station
19th Jan., 1910, 12.[illegible] p.m.	22	No. 4 Level, S. R. East Drive	2296	1422	26·50	75·2	74·1	95·8	3 ,, ,,
18th May, 1910, 1[illegible] a.m.					26·60	75·0	7[illegible]	93·1	2 ,, ,,
19th Jan., 1910, 12.[illegible] p.m.	23	No. 4 Level, S. R. Drive "Centre"	2408	—	26·47	76·3	75·6	96·8	3 ,, ,,
18th May, 1910, 1[illegible] a.m.					26·61	79·2	71·6	70·2	3 ,, ,,
19th Jan., 1910, 12.[illegible] p.m.	24	No. 4 Level, S. R. West Drive	2301	1265	26·46	74·7	73·4	94·5	4 ,, ,,
18th May, 1910, 1[illegible] a.m.					26·60	7[illegible]	73·0	90·1	4 ,, ,, Small winch here, running slow
[illegible] Jan., 1910, 1[illegible] p.m.	25	No. 5 Level, Station No. 2 Shaft	2407	—	26·[illegible]	71·8	71·7	99·0	4 ,, ,,
18th May, 1910, [illegible] p.m.					26·6[illegible]	65·1	60·4	77·3	4 ,, ,,

NOTE.—1st line of observation taken BEFORE Installing Fan.
2nd ,, ,, ,, AFTER ,, ,,

EFFECT ON MINE TEMPERATURES OF INSTALLING FAN.

B. PSYCHROMETRICAL OBSERVATIONS UNDERGROUND AT VILLAGE DEEP, LIMITED, JOHANNESBURG.

Date and Time of Observations.	Reference Number on Diagram.	Location of Observations.	Depth from Surface In feet.	Distance along Level from Shaft. In feet.	Barometric Readings.	Psychrometric Readings. Dry Bulb Fah.	Psychrometric Readings. Wet Bulb Fah.	Relative Humidity Saturation = 100	REMARKS.
20th Jan., 1910, 10.15 a.m.	26	No. 5 Level, S. D. Centre	2395	80	26·58	75·6	74·5	95·0	Curtain between Box and South Drive 3 Persons present
19th May, 1910, 10.10 a.m.					26·69	71·2	65·1	73·2	Strong Current running East
20th Jan., 1910, 10.20 a.m.	27	No. 5 Level, S. D. West	2395	980	26·56	73·9	72·7	94·0	4 Persons present
19th May, 1910, 10.15 a.m.					26·67	75·6	74·1	93·1	Air very bad
20th Jan., 1910, 10.30 a.m.	28	No. 5 Level, S. D. East	2395	956	26·55	75·0	74·3	97·0	4 Persons present
19th May, 1910, 10 a.m.					26·68	75·6	73·0	89·1	4 boys and 2 Machines here. Door shut to Crossent
20th Jan., 1910, 10.45 a.m.	29	No. 5 Level, West Drive	2394	1228	26·53	75·2	72·1	86·0	Door shut, also large leakage of air from machine here
18th May, 1910, 12.50 p.m.					26·62	75·0	73·9	95·1	3 Persons present
20th Jan., 1910, 11 a.m.	30	No. 5 Level, between Shafts	2375	772	26·54	74·7	73·8	96·0	4 Persons present
18th May, 1910, 12.30 p.m.					26·62	72·9	69·6	85·5	Door open West of No. 1 Station
20th Jan., 1910, 11.10 a.m.	31	No. 5 Level Station, No. 1 Shaft	2382	—	26·55	73·4	71·6	92·0	8 Persons present
18th May, 1910, 12 noon.					26·63	66·6	61·5	77·6	4 Persons and Trammer Boys here
20th Jan., 1910, 11.25 a.m.	32	No. 5 Level, East Drive	2373	911	26·53	75·4	74·8	97·5	6 Persons present
18th May, 1910, 11.50 a.m.					26·62	72·3	70·5	91·7	3 Persons present. Sacking over door passing air
20th Jan., 1910, 11.50 a.m.	33	No. 6 Level Station, No. 1 Shaft	2471	—	26·63	73·8	72·0	92·0	4 Persons present
21st May, 1910, 10.12 a.m.					26·81	65·3	61·2	80·1	3 Persons present. Very little air passing
20th Jan., 1910, 12 noon.	34	No. 6 Level, East Drive	2467	995	26·60	75·9	75·2	97·0	3 Persons present Stopes being fully worked along this Drive Sack Curtain East of Station
21st May, 1910, 10.25 a.m.					26·80	76·3	75·2	98·7	
20th Jan., 1910, 12.10 p.m.	35	No. 6 Level, between Shafts	2467	821	26·62	75·6	74·5	95·5	4 Persons present
21st May, 1910, 10.45 a.m.					26·81	74·5	72·5	91·3	4 Persons present
20th Jan., 1910, 12.25 p.m.	36	No. 6 Level, West Drive	2488	1136	26·65	75·6	73·9	93·3	6 Persons present
21st May, 1910, 10.50 a.m.					26·80	68·4	62·6	73·5	Strong wind along Drive. 3 Persons present
20th Jan., 1910, 12.35 p.m.	37	No. 6 Level, S. D. Centre	2497	90	26·63	75·6	75·2	98·0	8 Persons present
19th May, 1910, 10.45 a.m.					26·77	73·4	69·8	84·2	No door from box
20th Jan., 1910, 12.40 p.m.	38	No. 6 Level, S. D. West	2494	410	26·65	76·1	74·5	93·3	9 Persons present
19th May, 1910, 11.5 a.m.					26·74	75·2	70·5	80·0	3 Persons present
20th Jan., 1910, 12.45 p.m.	39	No. 6 Level, S. D. East	2497	410	26·62	75·9	75·1	96·0	4 Persons present
19th May, 1910, 10.55 a.m.					26·75	77·4	76·3	95·3	3 Persons present
20th Jan., 1910, 12.55 p.m.	40	No. 6 Level Station, No. 2 Shaft	2500	—	26·61	76·6	75·6	95·4	16 Persons present
19th May, 1910, 10.35 a.m.					26·78	65·8	60·8	76·0	6 Persons present. Doors to Drives shut
21st Jan., 1910, 9.55 a.m.	41	No. 7 Level Station, No. 2 Shaft	2600	—	26·90	75·4	75·2	99·2	1 Persons present
19th May, 1910, 12.10 p.m.					26·86	66·9	63·0	81·4	6 Persons present Door here shut, but passing air
21st Jan., 1910, 10.15 a.m.	42	No. 7 Level, West Drive	2587	939	26·87	75·1	70·9	82·0	7 Persons present. 2 Machines working here
21st May, 1910, 12 noon.					26·91	78·1	75·7	89·9	Door open to Station. 4 Persons present
21st Jan., 1910, 10.19 a.m.	43	No. 7 Level, S. D. Centre	2602	90	26·89	75·9	75·2	97·0	4 Persons present
19th May, 1910, 12.22 p.m.					26·82	75·7	75·2	97·7	3 Persons present
21st Jan., 1910, 10.25 a.m.	44	No. 7 Level, S. D. West	2602	455	26·85	76·3	75·6	97·0	4 Persons present
19th May, 1910, 12.30 p.m.					26·81	76·3	75·2	95·1	3 Persons present
21st Jan., 1910, 10.30 a.m.	45	No. 7 Level, S. D. East	2602	469	26·87	76·3	75·4	97·0	4 Persons pcesent
19th May, 1910, 12.37 p.m.					26·82	76·5	75·6	95·1	3 Persons present
21st Jan., 1910, 10.40 a.m.	46	No. 7 Level, between Shafts	2557	739	26·85	76·5	75·2	94·5	4 Persons pesent
21st May, 1910, 11.33 a.m.					26·91	74·1	72·0	90·6	Doors open at No. 2 Shaft. Wind West to East
21st Jan., 1910, 10.50 a.m.	47	No. 7 Level, East Drive	2560	1059	26·73	76·1	75·2	96·0	4 Persons present
21st May, 1910, 11.15 a.m.					26·90	73·8	71·2	88·4	3 Persons present. Door open to Station
21st Jan., 1910, 11 a.m.	48	No. 7 Level Station, No. 1 Shaft	2569	—	26·87	74·7	72·7	91·0	4 Persons present
21st May, 1910, 11.5 a.m.					26·91	66·9	63·3	82·8	4 Persons present
21st Jan., 1910, 11.23 a.m.	49	No. 8 Level Station, No. 1 Shaft	2663	—	26·95	74·5	72·7	92·0	4 Persons present
23rd May, 1910, 11.15 a.m.					26·80	71·6	65·5	73·3	3 Persons present. Door open West of Station

NOTE.—1st line of observations taken BEFORE Installing Fan.
2nd ,, ,, ,, AFTER ,, ,,

EFFECT ON MINE TEMPERATURES OF INSTALLING FAN.

C.—PSYCHROMETRICAL OBSERVATIONS UNDERGROUND AT VILLAGE DEEP, LIMITED, JOHANNESBURG.

Date and Time of Observations.	Reference Number on Diagram.	Location of Observations.	Depth from Surface In feet.	Distance along Level from Shaft. In feet.	Barometric Readings.	Psychrometric Readings. Dry Bulb °Fah.	Psychrometric Readings. Wet Bulb °Fah.	Relative Humidity Saturation = 100.	REMARKS.
21st Jan, 1910, 11.37 am	59	No. 8 Level Station, No. 2 Shaft	2688	—	26·95	77·4	76·8	97·5	4 persons present
25th May, 1910, 9.30 am					26·82	68·0	64·8	84·9	3 ,, ,, Fowler hoist here
11th Feb, 1910, 10.50 am	51	No. 8 Level, West Drive	2685	500	26·79	77·5	76·7	97·0	4 ,, ,,
23rd May, 1910, 10.15 am					26·82	78·8	77·4	94·7	8 ,, ,, 4 boys filling trucks
14th Jan, 1910, 10.[illegible] am	52	No. 8 Level, East Drive	2653	1230	26·85	76·6	75·6	95·4	5 ,, ,,
23rd May, 1910, 11.30 am					26·82	76·3	75·6	96·9	3 ,, ,,
14th Jan, 1910, 10.40 am	53	No. 8 Level, between Shafts	2619	975	26·88	76·3	75·6	97·1	5 ,, ,,
23rd May, 1910 11.15 am					26·80	74·8	72·0	87·8	Good wind west to east. Door open at No 2 shaft
20th Jan, 1910, 0 10 am	54	No. 9 Level Station, No. 1 Shaft	2759	250	27·07	76·3	73·0	86·0	9 persons present
23rd May, 1910, 11.40 am					26·97	71·6	68·1	85·5	8 ,, ,,
[illegible]th Jan, 1910, 10.[illegible] am	55	No 9 Level, East Drive	2747	1068	27·05	78·3	76·7	93·0	7 ,, ,,
23rd May, 1910, 11.[illegible] am					26·90	75·6	74·1	93·4	4 ,, ,, Good wind west to east
[illegible]th Jan, 1910, 10.17 am	56	No 9 Level, West Drive	2745	1059	27·04	77·9	76·6	94·5	8 ,, ,,
23rd May, 1910, 12.[illegible] pm					26·85	76·6	75·6	95·5	3 ,, ,,
18th Jan, 1910, 10.[illegible] am	57	No. 9 Level, East Drive	2768	1043	27·05	77·0	72·3	80·9	7 ,, ,, 2 machines working here
23rd May, 1910, 12.[illegible]7 pm					26·92	75·6	71·5	95·1	3 ,, ,,
18th Jan, 1910, 10.0 am	58	No. 9 Level Station, No. 2 Shaft	2783	—	27·08	78·3	77·1	96·2	5 ,, ,,
25th May, 1910, 9.15 am					26·89	69·8	67·6	95·1	6 ,, ,,
18th Jan, 1910, 10.[illegible] am	59	No. 9 Level, West Drive	2766	1024	27·06	78·8	77·7	96·6	5 ,, ,, 1 machine working here
23rd May, 1910, 12.40 pm					26·85	76·6	75·6	95·5	3 ,, ,,
18th Jan, 1910, 11.10 am	60	No. 10 Level Station, No. 2 Shaft	2881	—	27·21	78·8	78·1	97·0	5 ,, ,,
25th May, 1910, 9.[illegible] am					27·12	74·7	73·2	93·7	3 ,, ,,
18th Jan, 1910 11.[illegible] am	61	No. 10 Level, West Drive	2869	776	27·19	76·6	74·1	89·2	8 ,, ,,
25th May, 1910, 10.[illegible] am					27·10	77·1	76·3	95·3	6 ,, ,, 1 machine working here
18th Jan, 1910, 11.[illegible] am	62	No. 10 Level, East Drive	2868	791	27·11	78·3	77·1	96·2	7 ,, ,,
25th May, 1910, 10.[illegible] am					27·10	78·4	77·0	[illegible]	1 ,, ,, Waugh drill working west of observation. Door open east of station
[illegible]th Jan, 1910, 10.[illegible] am	63	No. 10 Level Station, No. 1 Shaft	2858	—	27·15	75·9	71·3	93·0	6 ,, ,,
27th May, 1910, 11.[illegible] am					27·09	69·1	66·9	88·3	1 ,, ,,
[illegible]th Jan, 1910, 1[illegible] am	64	No. 10 Level, West Drive	2850	761	27·15	77·0	76·6	98·0	9 ,, ,, 1 machine working here
[illegible]th May, 1910, 10.[illegible] am					27·10	77·7	76·3	94·0	1 ,, ,,
[illegible]th Jan, 1910, 1[illegible].10 am	65	No. 10 Level, East Drive	2851	766	27·15	76·4	75·6	97·0	6 ,, ,,
[illegible]th May, 1910, 11.[illegible] am					27·10	77·7	76·3	94·0	1 ,, ,,
1st Feb, 1910, [illegible] am	66	No. 11 Level Station, No. 1 Shaft	2861	—	27·21	75·2	74·7	97·5	5 ,, ,, Air pump in cross cut
27th May, 1910, 10.[illegible] am					27·18	71·6	69·4	89·9	Large pump here and small one in cross cut
1st Feb, 1910, 10.[illegible] am	67	No. 11 Level, East Drive	2943	580	27·25	78·1	77·2	96·0	3 persons present
[illegible]th May, 1910, 10.15 am					27·18	75·6	73·1	90·1	3 ,, ,, Good wind west to east
1st Feb, 1910, 10.[illegible] am	68	No. 11 Level, West Drive	2942	611	27·27	78·8	77·7	95·5	4 ,, ,,
27th May, 1910, 11.[illegible] am					27·21	77·7	77·6	[illegible]	[illegible] ,, ,, 3 trammers here
[illegible]th Jan, 1910, 11.[illegible] am	69	No. 11 Level Station, No. 2 Shaft	2976	—	27·34	79·5	78·1	95·5	3 ,, ,,
[illegible]th May, 1910, 11.10 am					27·17	75·2	73·0	90·1	1 ,, ,,
[illegible]th Jan, 1910, 12.[illegible] pm	70	No 11 Level, East Drive	2969	620	27·20	79·5	78·8	97·1	3 ,, ,,
[illegible]th May, 1910, 11.[illegible] am					27·16	78·1	77·1	95·9	3 ,, ,, and 3 trammers
[illegible]th Jan, 1910, 12.[illegible] pm	71	No. 11 Level West Drive	2971	618	27·22	78·1	77·4	95·1	3 ,, ,,
[illegible]th May, 1910, 11.[illegible] am					27·15	77·0	76·8	98·3	3 ,, ,, [illegible] machines and [illegible] boys here
[illegible]th Jan, 1910, 11.[illegible] am	72	No. 12 Level Station, No. 2 Shaft	3095	—	27·28	79·5	78·8	97·0	5 ,, ,,
[illegible]th May, 1910, 1.[illegible] pm					27·20	77·4	75·7	93·1	6 ,, ,,
[illegible]th Jan, 1910, 11.[illegible] am	73	No. 12 Level Drive West	3091	365	27·28	79·2	77·7	94·2	1 ,, ,,
[illegible]th May, 1910, 1[illegible] am						78·8	77·7	95·4	3 ,, ,,

NOTE.—1st line of observations taken BEFORE Installing Fan.
2nd ,, ,, ,, AFTER ,, ,,

EFFECT ON MINE TEMPERATURES OF INSTALLING FAN

D. PSYCHROMETRICAL OBSERVATIONS UNDERGROUND AT VILLAGE DEEP, LIMITED, JOHANNESBURG.

Date and Time of Observations.	Reference Number on Diagram.	Location of Observations.	Depth from Surface In feet.	Distance along Level from Shaft. In feet.	Barometric Readings.	Psychrometric Readings.		Relative Humidity Saturation = 100.	REMARKS.
						Dry Bulb Fah.	Wet Bulb Fah.		
13th Jan. 1910, 12 noon	74	No. 12 Level, East Drive	3090	411	27·28	79·2	78·4	97·2	5 Persons present
28th May, 1910, 11.25 am					27·22	78·8	76·3	89·6	3 ,, ,, 3 trammers here
1st Feb. 1910, 10.5 am	75	No. 12 Level, Station No. 1 Shaft	3073	—	27·29	76·6	75·2	94·0	Water on this level
—					—	—	—	—	
th January, 1910, 10.8 am	78	No. 13 Level, Station No. 2 Shaft	3240	—	27·37	79·2	78·4	97·0	4 Persons present
27th May, 1910, 10.28 am					27·31	77·7	77·0	96·9	4 ,, ,,
13th Jan. 1910, 10.15 am	79	No. 13 Level, East Drive	3238	236	27·37	79·5	78·8	95·3	4 ,, ,,
25th May, 1910, 11.25 am					27·30	80·3	79·9	98·4	4 ,, ,, 4 trammers here
13th Jan. 1910, 10.25 am	80	No. 13 Level, West Drive	3238	241	27·37	79·5	78·8	97·0	4 ,, ,,
25th May, 1910, 11.37 am					27·30	79·2	76·6	89·4	4 Boys and 1 machine hera
13th Jan. 1910, 10.45 am	81	No. 13 Level, S.R.D. Centre	3239	98	27·37	79·2	78·4	97·1	4 Persons present
25th May, 1910, 11.15 am					27·30	78·8	78·1	96·1	3 ,, ,,
13th Jan, 1910, 11.10 am	82	No. 13 Level, S.R.D East	3239	170	27·37	80·2	79·0	97·0	4 ,, ,,
25th May, 1910, 11.50 am					27·30	78·8	77·9	96·2	8 Boys and two machines here
13th Jan, 1910, 11.30 am	83	No. 13 Level, S.R.D. West	3239	165	27·37	79·5	78·6	96·0	4 Persons present
25th May, 1910, 11.55 am					27·30	78·8	77·9	96·2	3 ,, ,,
13th Jan, 1910, 12.5 pm	84	No. 12 Level, S.R.D. Centre	3095	—	27·28	78·4	77·7	96·8	4 ,, ,,
28th May, 1910, 11.15 am					27·22	78·8	77·4	94·1	3 ,, ,,
13th Jan, 1910, 12.12 am	85	No. 12 Level, S.R.D. East	3096	198	27·28	78·4	78·1	97·9	4 ,, ,,
28th May, 1910, 11 15 am					27·22	78·8	77·4	94·1	3 ,, ,,
13th Jan, 1910, 12.20 pm	86	No. 12 Level, S.R.D. West	3094	210	27·28	78·8	78·1	96·8	4 ,, ,,
28th May, 1910, 11.15 am					27·22	78·8	77·4	94·1	3 ,, ,,
18th Jan, 1910, 10.15 am	87	No. 9 Level, S.R.D. West	2781	447	27·08	77·5	76·6	96·0	3 ,, ,,
23rd May, 1910, 12 50 pm					26·86	77·0	76·3	97·0	3 ,, ,,
18th Jan, 1910, 10.20 am	88	No. 9 Level, S.R.D. Centre	2780	—	27·07	77·0	76·3	97·0	3 ,, ,,
23rd May, 1910, 12.55 pm					26·87	77·0	76·0	95·3	3 ,, ,,
18th Jan, 1910, 10.25 am	89	No. 9 Level, S.R.D. East	2782	435	27·09	77·4	76·6	97·0	3 ,, ,,
23rd May, 1910, 1 pm					26·87	77·4	76·3	95·3	3 ,, ,,
18th Jan, 1910, 11.40 am	90	No. 10 Level, S.R.D. Centre	2879	—	27·15	77·7	77·0	97·0	4 ,, ,,
26th May, 1910, 10.50 am					27·10	77·4	75·9	93·7	3 ,, ,, 3 boys and 1 track layer here
18th Jan, 1910, 11.45 am	91	No. 10 Level, S.R.D. East	2878	278	27·14	77·9	77·2	97·1	4 ,, ,,
26th May, 1910, 11 am					27·10	77·7	76·3	94·0	3 ,, ,,
18th Jan, 1910, 11.44 am	92	No. 10 Level, S.R D. West	2877	—	27·14	78·4	77·4	95·4	3 ,, ,,
26th May, 1910, 10.55 am					27·10	77·7	76·3	94·0	3 ,, ,,
18th Jan, 1910, 12.6 pm	93	No. 11 Level, S.R.D. Centre	—	—	27·23	78·1	77·4	97·3	3 ,, ,,
26th May, 1910, 12.10 pm					27·15	77·7	76·6	95·3	3 ,, ,,
18th Jan, 1910, 12.8 pm	94	No. 11 Level, S.R.D. East	—	—	27·25	78·8	78·1	97·0	3 ,, ,,
26th May, 1910, 12.17 pm					27·16	77·7	77·0	97·3	2 ,, ,,
18th Jan, 1910, 12.15 pm	95	No. 11 Level, S.R.D. West	—	—	27·25	78·1	77·4	97·3	3 ,, ,,
26th May, 1910, 12.25 pm					27·16	78·1	77·4	97·0	2 ,, ,,

EFFECT ON MINE TEMPERATURES OF INSTALLING FAN.

E.—PSYCHROMETRICAL OBSERVATIONS AT VILLAGE DEEP, LIMITED, JOHANNESBURG.

Date and Time of Observations.	Reference Number on Diagram.	Location of Observations.	Depth from Surface In feet.	Distance along Level from Shaft In feet	Barometric Readings.	Psychrometric Readings.		Relative Humidity Saturation = 100.	REMARKS.
						Dry Bulb Fah.	Wet Bulb Fah		
14th Jan, 1910, 9.40 am		On Surface Collar, No. 2 Shaft			24·46	71·9	64·0	67·1	5,717 feet above sea level
14th Jan, 1910, 1.30 pm					24·46	77·4	63·3	48·2	Max and min. thermometer 72·0° to 78·5° Fah
18th Jan, 1910, 9.15 am		,, ,, ,,			24·48	66·9	62·6	79·8	,, ,, ,, 64° to 79° Fah
,, ,, ,, 1.30 pm					24·50	80·2	77·9	91·0	
19th Jan, 1910, 9.30 am		,, ,, ,,			24·40	66·9	61·9	77·1	
,, ,, ,, 1.30 pm					24·33	71·2	63·3	63·8	,, ,, ,, 67° to 71° Fah
20th Jan, 1910, 9.15 am		,, ,, ,,			24·38	60·8	58·8	90·0	
,, ,, ,, 1.40 pm					24·32	72·7	67·1	76·7	,, ,, ,, 62° to 69° Fah
21st Jan, 1910, 9.20 am		,, ,, ,,			24·54	65·5	58·3	67·2	
,, ,, ,, 12.10 pm					24·48	71·6	62·9	64·5	
,, ,, ,, 12.[illegible] pm		On Surface Collar, No. 1 Shaft			24·50	69·1	59·7	59·0	5,632 feet above sea level
,, ,, ,, 1.0 pm					24·63	57·2	53·9	82·4	
,, ,, ,, 11.20 pm					24·63	57·9	55·0	84·9	
25th Jan, 1910, 9.15 am		On Surface Collar, No. 2 Shaft			24·50	73·2	63·3	61·0	Max. and min thermometer 70° to 71° Fah
,, ,, ,, 12.30 pm					24·48	76·6	64·0	53·0	
1st Feb, 1910, 9.0 am		,, ,, ,,			24·48	65·5	58·8	69·5	,, ,, ,, 66° to 75° Fah
,, ,, ,, 1.30 pm					24·45	70·9	61·2	60·0	
4th Feb, 1910, 1.0 pm		,, ,, ,,			24·51	71·1	59·7	54·0	,, ,, ,, 74° to 76° Fah.
17th May, 1910, 9.0 am		,, ,, ,,			24·62	60·6	45·7	32·4	Bright morning
,, ,, ,,					24·52	65·1	47·1	25·7	Max. and min. thermometer 64° to 66° Fah
18th May, 1910, 9.15 am		,, ,, ,,			24·54	60·1	45·7	33·7	Fine day, cloudless
,, ,, ,, 1.15 pm					24·47	66·6	47·8	24·7	
19th May, 1910, 9.[illegible] am		,, ,, ,,			24·55	63·7	47·1	30·9	Bright morning
,, ,, ,, 1.10 pm					24·45	69·1	49·3	24·1	
,, ,, ,, 1.[illegible] pm		On Surface Collar, No. 1 Shaft			24·58	71·6	50·7	23·[illegible]	5,692 feet above sea level
					—	—	—	—	
21st May, 1910, [illegible] am		On Surface Collar, No. 2 Shaft			24·60	62·6	48·2	36·3	Bright morning
,, ,, ,, 12.1[illegible] pm					24·51	68·4	50·4	29·0	Max. and min. thermometer 63° to 68° Fah
23rd May, 1910, 9.10 am		,, ,, ,,			24·47	61·5	47·8	38·1	Bright morning
,, ,, ,, 1.[illegible] pm					24·32	67·3	66·9	98·1	Max. and min. thermometer 60° to 65° Fah
24th May, 1910, 9.[illegible] am		,, ,, ,,			24·46	38·1	36·7	88·1	Cloudy weather
,, ,, ,, 1.[illegible] pm					24·30	41·6	39·2	81·6	Max and min. thermometer 42° to 44° Fah
26th May, 1910, 9.10 am		,, ,, ,,			24·52	47·5	42·3	67·7	Bright morning
,, ,, ,, 1.15 pm					24·40	59·7	48·6	46·9	Max. and min thermometer 56° to 64° Fah
27th May, 1910, 9.15 am		,, ,, ,,			24·50	55·4	48·2	62·1	Bright morning
,, ,, ,, 1.[illegible] pm					24·40	58·6	48·6	51·0	Max. and min thermometer 58° to 61° Fah
28th May, 1910, [illegible] am		,, ,, ,,			24·50	54·7	41·2	45·9	Cloudy weather
,, ,, ,, 12.[illegible] pm					24·46	55·1	45·3	48·2	Max. and min. thermometer 50° Fah
29th May, 1910, 10.0 am		,, ,, ,,			24·49	54·0	39·6	26·9	Cloudy weather
,, ,, ,, 11.30 am					24·45	59·7	41·4	36·1	Max and min thermometer 59° Fah

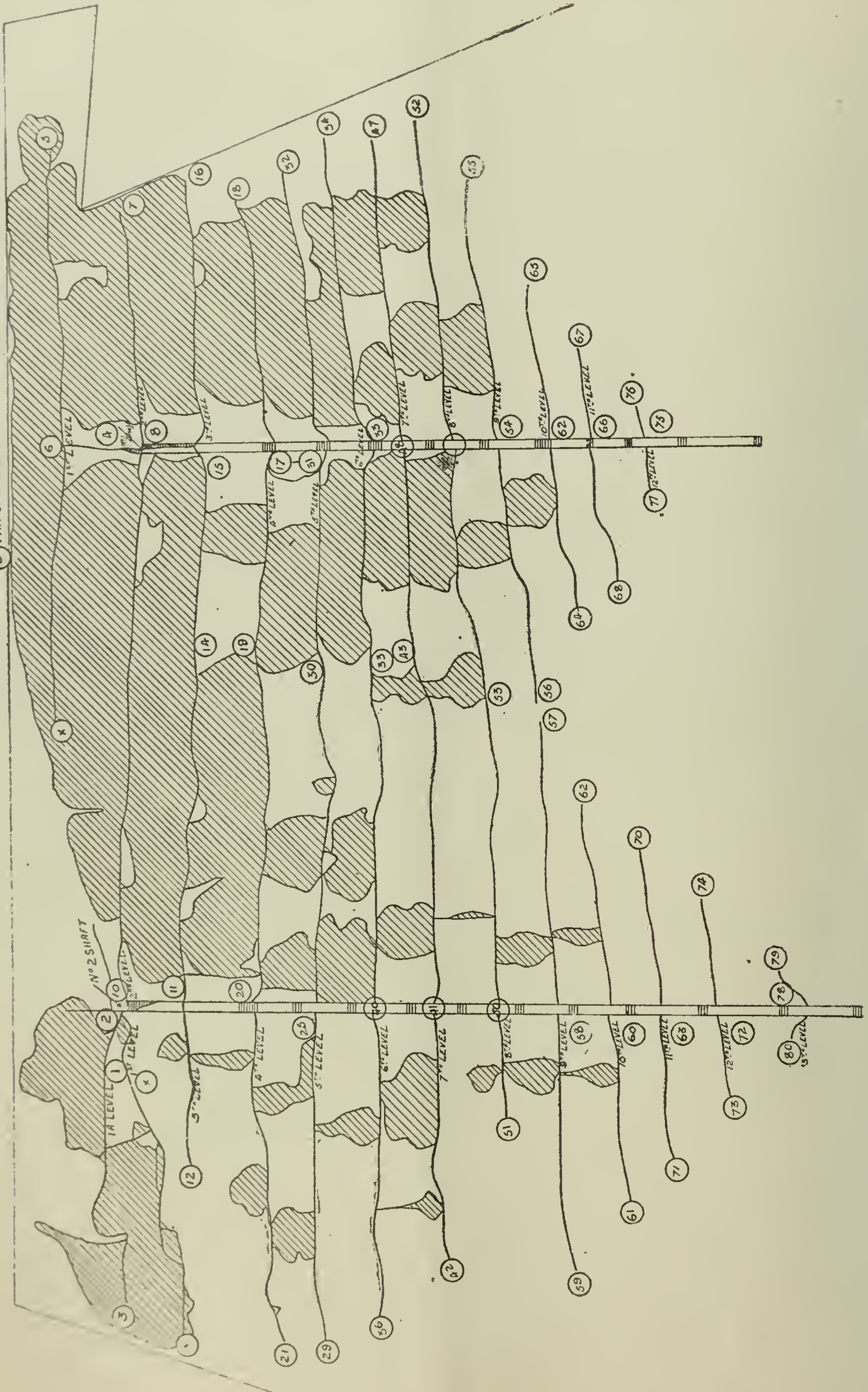


FIG. 2.—Village Deep, Ltd. Plan of Main Reef Leader, giving Position of Underground Psychrometrical Observations.

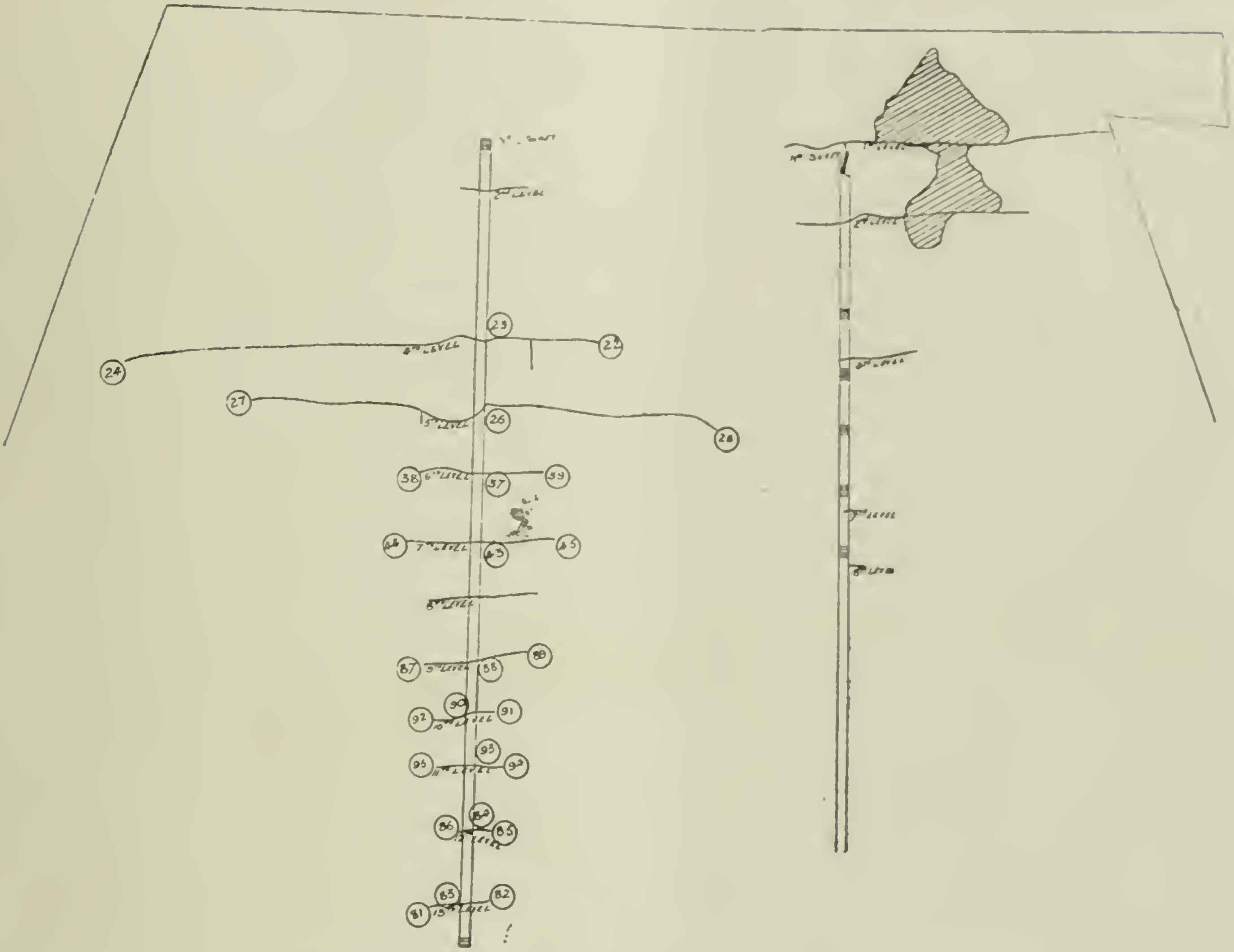


FIG. 3.—Village Deep, Ltd. Plan of South Reef, giving Position of underground Psychrometrical Observations.

Rock temperatures, Nos. 3 to 12 levels inclusive:—

Average temperature before ... 76·2° F.
„ „ after ... 73·0 F.
difference ... 3·2 F.

The rock temperatures were taken at dead ends and in stopes, with an electric resistance thermometer placed in holes of an average depth of 5 ft.

With regard to dead ends, although no special means such as pipes were provided for transmitting or aiding the circulation of fresh air from the stations to dead ends, yet the observations in general show tangible improvements both in temperature and humidity. At distances up to 1,000 ft. from the shaft stations a decrease of 2 to 3° F. occurred, and the humidity which had been previously very high was reduced by from 2% to 6%. The records also show that the fan is exhausting a very large quantity of water from the mine. The average quantity of air dealt with at the time of these observations was 260,000 cub. ft. per minute, for 23½ hours per day. The average moisture in the upcast is about 8·2 grains per cub. ft. The surface or intake conditions of the air show a wide range of humidity, but careful records show the moisture extracted from the mine by means of the fan to have been as high as 40,400 gallons per day, or roughly 202 tons. The removal of such a large quantity of water from the mine workings daily has naturally a tendency to make them very dry, which on the Rand results in their being very dusty, and hence in the workings themselves a plentiful supply of water is absolutely necessary to make up for this daily drying process and to keep the dust down, otherwise good ventilation as far as dust is concerned may make matters worse than before. Therefore the efforts to keep the air as free from moisture as possible, already referred to, can only be applied to the currents before they reach the faces.

Mr. Whitehouse cannot say what has been the experience on the East Rand Proprietary Mines, since he left there soon after the ventilation scheme was completed, but the effect of this drying process has been very marked on the Village Deep where, as shown, the air carries away large quantities of water, and it has been necessary to provide a plentiful supply of water for laying dust in stopes.

Again, the presence of varying quantities of CO and CO_2 in mine air calls for some consideration. The average figures for the East Rand Proprietary Mine have been given as :—

(*a*) Before the adoption of the ventilation scheme—

	per cent.
CO_2	0·493
CO	0 012

(*b*) After the adoption of the ventilation scheme—

	per cent.
CO_2	0·127
CO	0·005

The corresponding figures for the Village Deep will be seen in Table F and the following diagram Fig. 4 :

F.—ANALYSIS OF SAMPLES OF MINE AIR BEFORE INSTALLATION OF FAN—VILLAGE DEEP.

Date and Time.	Chart No.	Position.	Analysis.			Remarks.
			Oxygen.	Carbon Monoxide.	Carbon Dioxide.	
Jan. 18, 1910, 6.10 a.m.	50	No. 8 station No. 2 shaft	20·7	0·003	0·295	Samples taken previous to installation of ventilating fan.
Jan. 14, 1910, 6.30 p.m.	50	No. 8 station No. 2 shaft	20·65	0·007	0·366	
Jan. 18, 1910, 6.15 p.m.	9	Fan chamber	20·80	0·001	0·205	
Jan. 14, 1910, 6 p.m.	9	Fan chamber	20 70	0·005	0·254	
Feb. 10, 1910, 4.15 p.m.	50	No. 8 station No. 2 shaft	20·60	0·002	0·293	Station crowded with men and boys waiting for skips

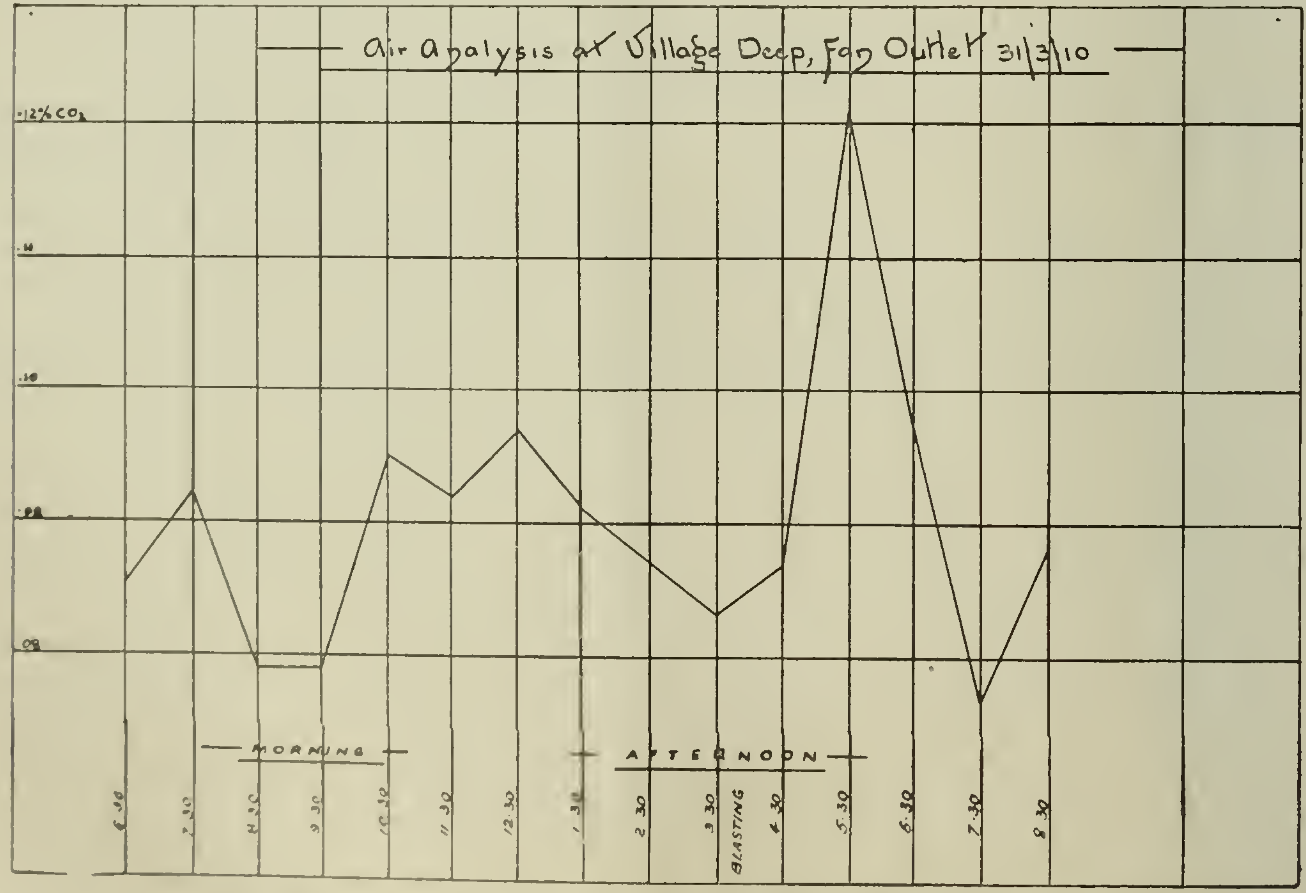


Fig. 4.—Curve of Daily Variation of CO_2 after Installation of Fan.

Before the fan started the average percentage of CO_2 was 0·282 and afterwards it will be seen that the highest percentage reached is 0·12, this being after blasting in the afternoon, whereas during the day it averages less than 0·09.

From the foregoing remarks it will be seen that the only way to overcome the increased temperature in our deep mines so that the workings may be kept cool and free from foul air, and men may work efficiently, is to provide large volumes of fresh air throughout the workings. The question of how this should be attained may then be considered. It will be clear that the deeper the mine the stronger will be the natural ventilation, and it might be supposed that the deeper mines would ventilate naturally for this reason. The shafts, however, are generally speaking of far too small a section to permit of large quantities being passed through them by the natural pressure obtainable and due to difference of temperatures.

Where a property has two shafts the natural ventilation cannot exceed a certain amount. This amount depends upon the difference of temperature in the upcast and downcast shafts, and also on the resistance offered to the currents in the shafts and mine. The pressure due to the difference of temperature can be calculated by taking the difference in weight of two columns of air, say 1 sq. ft. section, and of the same length as the depths of the shafts in question, one in the downcast and one in the upcast. The temperature must be taken from the downcast to be that experienced on the surface during summer, when the difference in temperature of the two shafts is least. Assuming the temperature at the surface to be say 70 and that in the upcast 85 we have a difference of 15° F. For shafts having a depth of say 3,000 feet this represents a difference of pressure of 5·4 lb. per sq. foot or just over 1 in. of W.G. If natural ventilation is to be used, this pressure must overcome the resistance of the downcast, the mine, and the upcast. Taking this pressure with two five compartment shafts, 3,000 ft. deep, and an assumed length of drives, and working out by the formula

$$P = \frac{KSQ^2}{A^3}$$

it will be seen that only about 50,000 cub. ft. of air can be circulated. Since the Government regulations require 70 cub. ft. per person underground, this would only allow about 750 at one time, and with this small labour force no deep level company could obtain a sufficiently large output to justify the large outlay required. Hence mechanical ventilation is absolutely necessary in the deep mines, and the large undertakings which are now in hand cannot be considered without its adoption.

Again, in considering the deep level mines, the question of the size and shape of the shafts is of considerable importance. Let us assume, for instance, that 250,000 cub. ft. per minute are required, and have to pass through a five compartment shaft, 3,000 ft. deep. The pressure required would be something like 7 in. of W.G. This would be far too expensive since the power required to circulate air at 7 in. W. G. would be very great. It is most important, therefore, that as large an area as possible should be provided through which to pass the air from the surface, and hence the advantage of circular shafts. One of these is now being sunk at New Modderfontein and it will deliver a fresh supply of air right to the deepest workings. It is also desirable that the incline shaft leading from the verticals should be of sufficient size and be sunk in the footwall so that they may always stand as the intakes of the mine and the workings the upcast.

The importance of large areas and hence low W.G. can be seen in the following example:—

At 5 in. W.G. the fan exhausted 456,000 c. ft. requiring 480 B.H.P.

At 2½ in. W.G. the fan exhausted 322,000 c. ft. requiring 191 B.H.P.

i.e., when the W.G. was halved the volume of air circulating was 75% of the former volume and the power required less than half. From these remarks it will be seen that in laying out a deep level property the question of ventilation must receive very serious consideration, and provision must be made in the shafts and in the laying out of the workings for the passage of large volumes of air free from excessive resistance. If this is not done the mine will have to carry throughout its life a much greater expense for ventilation owing to its high W.G. In cases where a connection can be made with an outcrop mine this difficulty can be overcome, since the two deep level shafts can be made downcast and the whole of the workings to the surface become the upcast. This is the arrangement on the Village Deep. There is also the timber in the shafts to be considered, and if the upcast air is passed through them the life of the timber is shortened considerably.

In laying out a ventilation scheme it is desirable that the air should be split into small circuits, in order to give each district a cool supply, since air which has been carried over great distances is usually hot and moist. After traversing these districts it should be brought into the main return as late as possible, so that the velocity in the latter is kept down and hence the pressure required is kept low. When calculating the required W.G. for a mine about to be ventilated the W.G. to

No. 88 Double Inlet "Sirocco" Mine Fan. Inlet area = 259·7 sq. ft. Discharge area = 95 sq. ft.
When working on a mine having an equivalent orifice = 81·55 sq. ft.

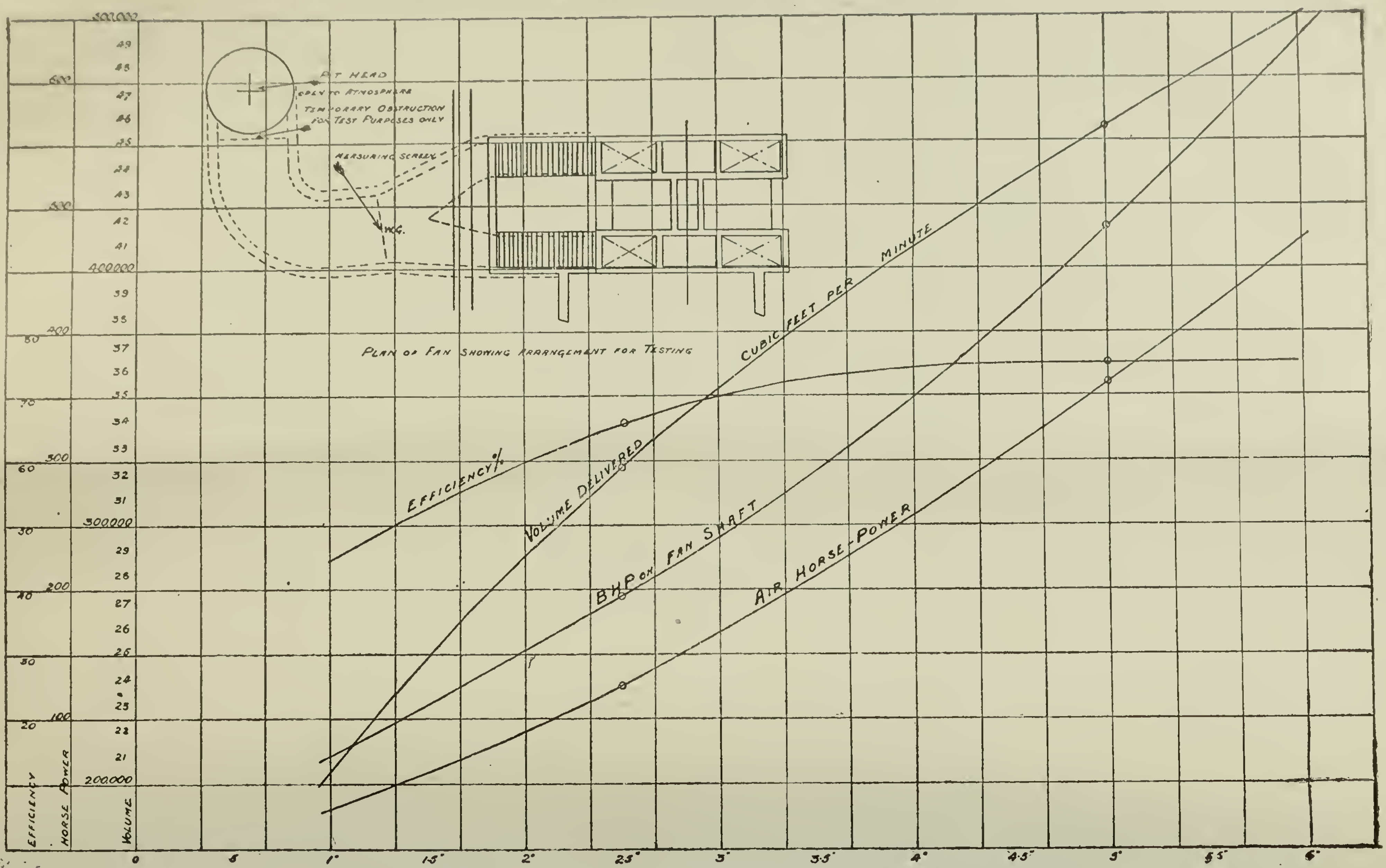
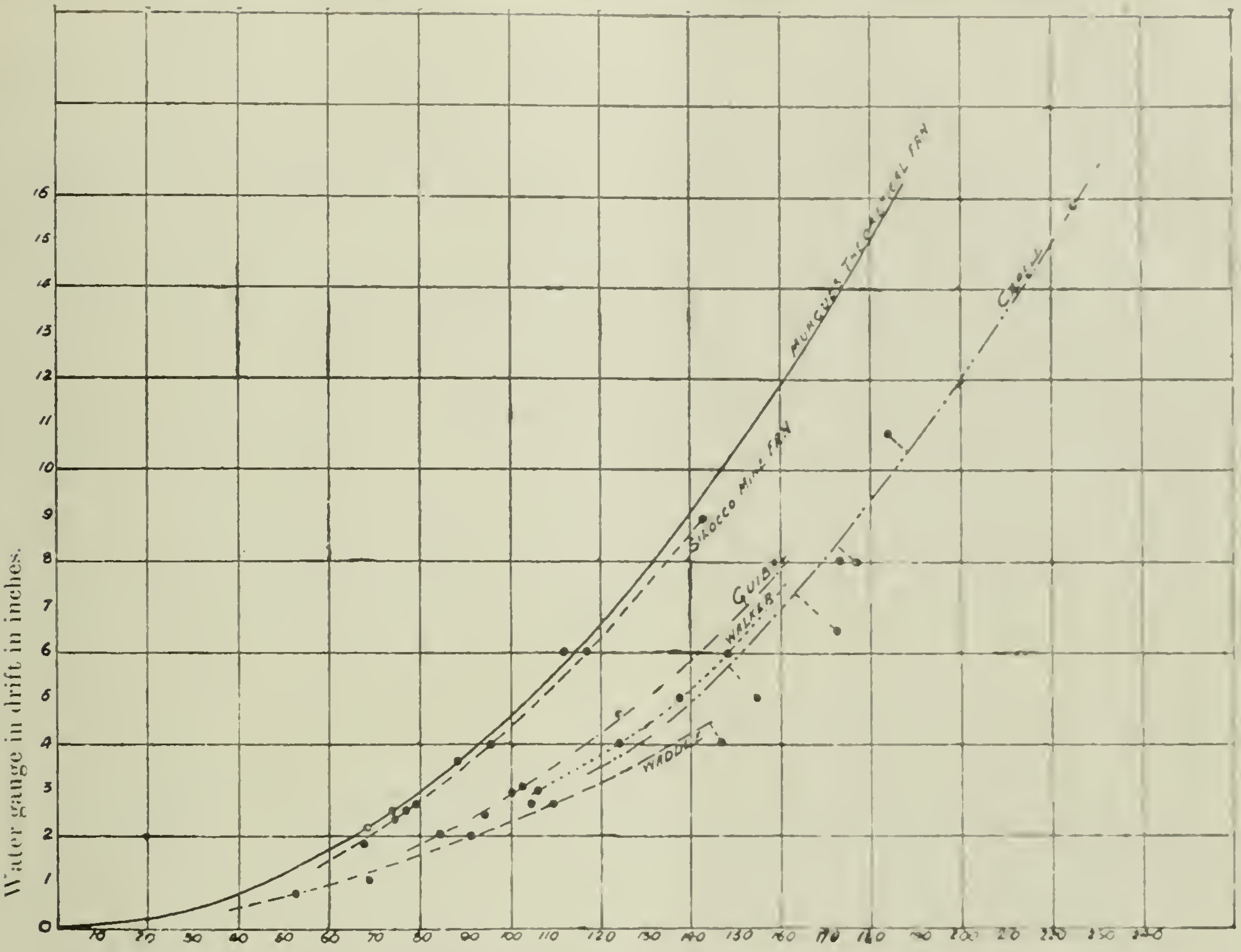


FIG. 5.—Showing Efficiency, Volume and Horse Power Required at Different Water Gauge

Examples } At 5 in. water gauge the fan will exhaust 456,000 cub. ft. per minute requiring 480 B.H.P. on fan shaft. Efficiency 75%. Air H.P. = 360.
2½ ,, ,, ,, 322,000 ,, ,, 194 ,, ,, ,, 65·5%. ,, 127.



Periphery speed of fan wheel in feet per second.

FIG. 6.—Curves showing the water gauge set up in drift for various periphery speeds on different types of mine ventilating fans, compiled from actual tests and published data. Murgues formula for theoretical fan, $H = \frac{u^2}{g}$

ventilate the most distant workings is the one that must be found, and the resistance of the other splits must be increased by the use of regulators. If this were not done all the air would pass through the nearest splits and the more distant parts of the mine would be left stagnant. Having picked out the split which has the greatest resistance, the W.G. required for the mine can then be calculated by taking the resistance of the shafts, drives leading to this split, and the resistance of the split itself separately. These added together will give the W.G. of the mine.

Local ventilation, too, is of great assistance where conditions do not allow of part of the workings being brought into the general system, and some parts of our mines are to-day being very well supplied by this means. The small sets required for this are cheap and extremely handy, and where cables are already laid they prove most convenient. Again, the condition of the air in our development drives can be considerably improved by having the winzes well forward, so that there is never a long length of drive unconnected.

The cost of mechanical ventilation is by no means excessive, although at one time anyone suggesting it was looked upon with amazement by the so-called practical men, who would tell you that it was impossible, and some miners regarded it with suspicion thinking either that if healthy conditions were provided the large cheques would cease, or perhaps that it would be too cold for him to spend the day on his box. However it has been noticed that miners generally show very great appreciation of the improved conditions, and to-day those mines which can offer the best conditions attract the best men. The cost of ventilation per ton milled on the East Rand Proprietory Mines has been given as less than 1d. per ton. The actual running cost at the Village Deep is 0·543d. per ton milled. This fan passes at present 288,000 cubic feet of air at a W.G. of 2⅞", the revolutions of the fan being 176, and H.P. required to drive it 205.

The position of the fan at any mine depends entirely upon the conditions obtaining therein, and it will be seen that this varies in almost every case. For instance at the East Rand Proprietary Mines the fans are placed at the surface. At the Village Deep it is placed underground at the northern boundary and exhausts into the outcrop mine. In the other example given (Fig. 7) it will be noticed that the fan is placed at the very bottom of the mine.

Fig. 8 shows the conditions obtaining on the western section of the East Rand Proprietary Mine before the fan was installed. The Angelo (or No. 1 South) shaft was the natural upcast for the Driefontein, Angelo and Comet (part of) mines. This condition of things was exceedingly detrimental to the Angelo mine. The smoke at blasting time travelled to the shaft before the men could get there, and they would have to remain in the smoke until they could all be raised by the skips. The mechanical ventilation scheme as now installed will be found in this *Journal* (Aug., 1910, p. 62, Supplement), and from this it will be seen that all the shafts are now downcasts, and after blasting the men go to stations where they are in air as good as that at the surface.

Fig. 9 shews the ventilation chart at the Village Deep, and here again it will be seen that all the shafts are downcasts. The air descends through these and splits off east and west along the different levels, and from these it passes through the workings to the fan at the northern boundary of the property.

Fig. 7 represents a third scheme which it has been considered worth including as an exceedingly simple arrangement made possible by the natural conditions of the mine, the dykes acting as natural brattices. The diagrams with their arrows explain the arrangement.

It may interest members to know that there are to-day far more ventilation schemes in hand than have already been completed, and in the Rand

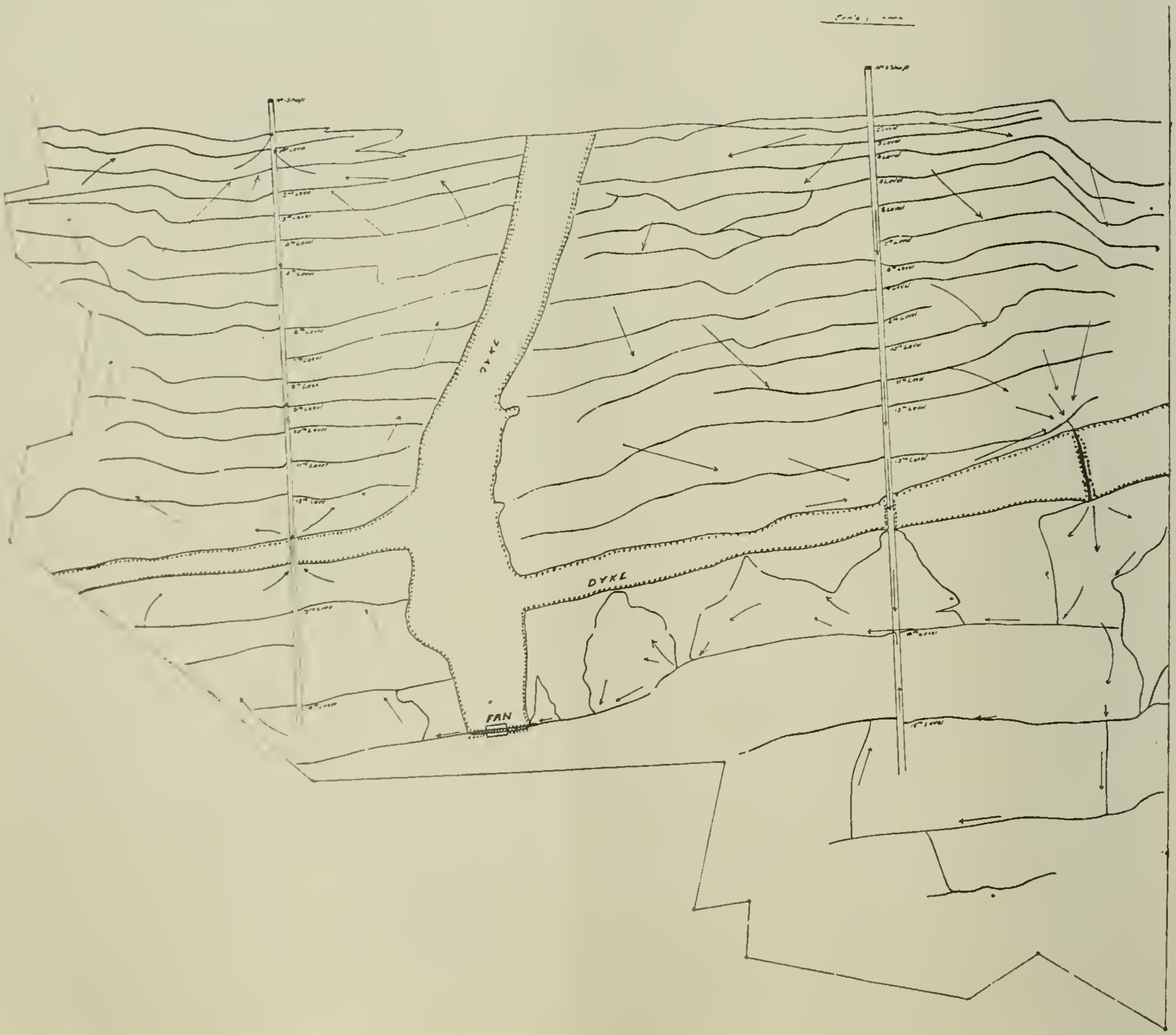


Fig. 7. Example of a Witwatersrand Gold Mine in which the natural conditions render mechanical ventilation exceedingly simple and inexpensive.

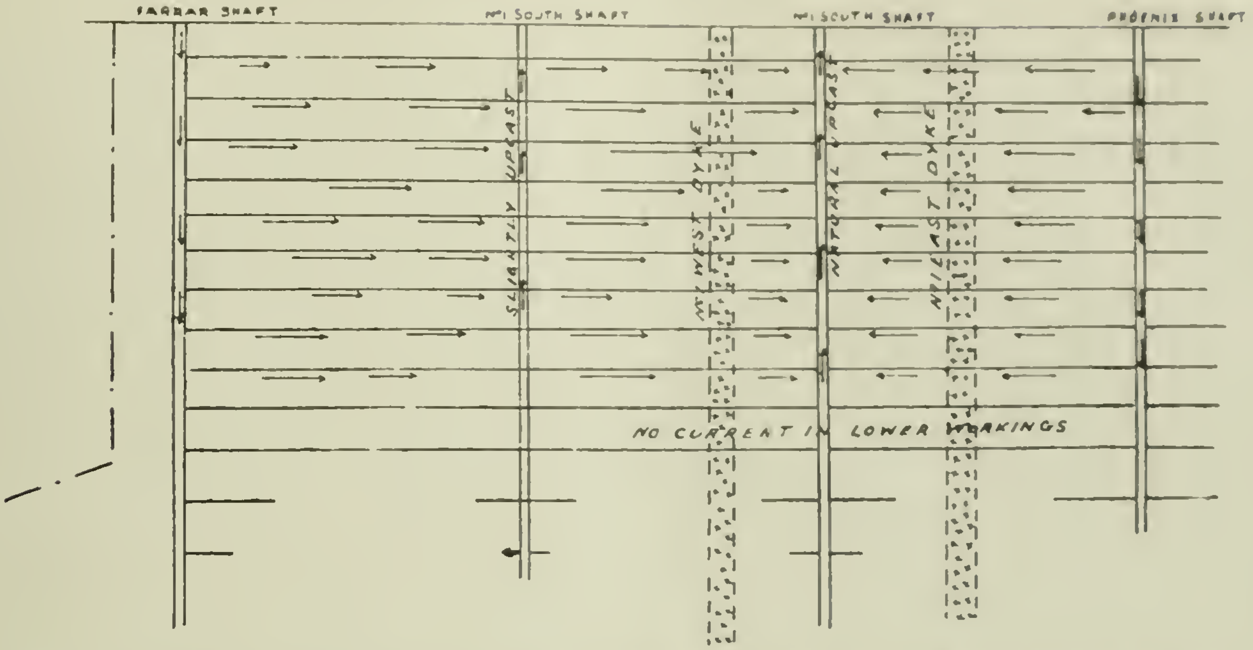


Fig. 8.—Diagram showing natural ventilation at East Rand Proprietary Mines previous to fan being installed.

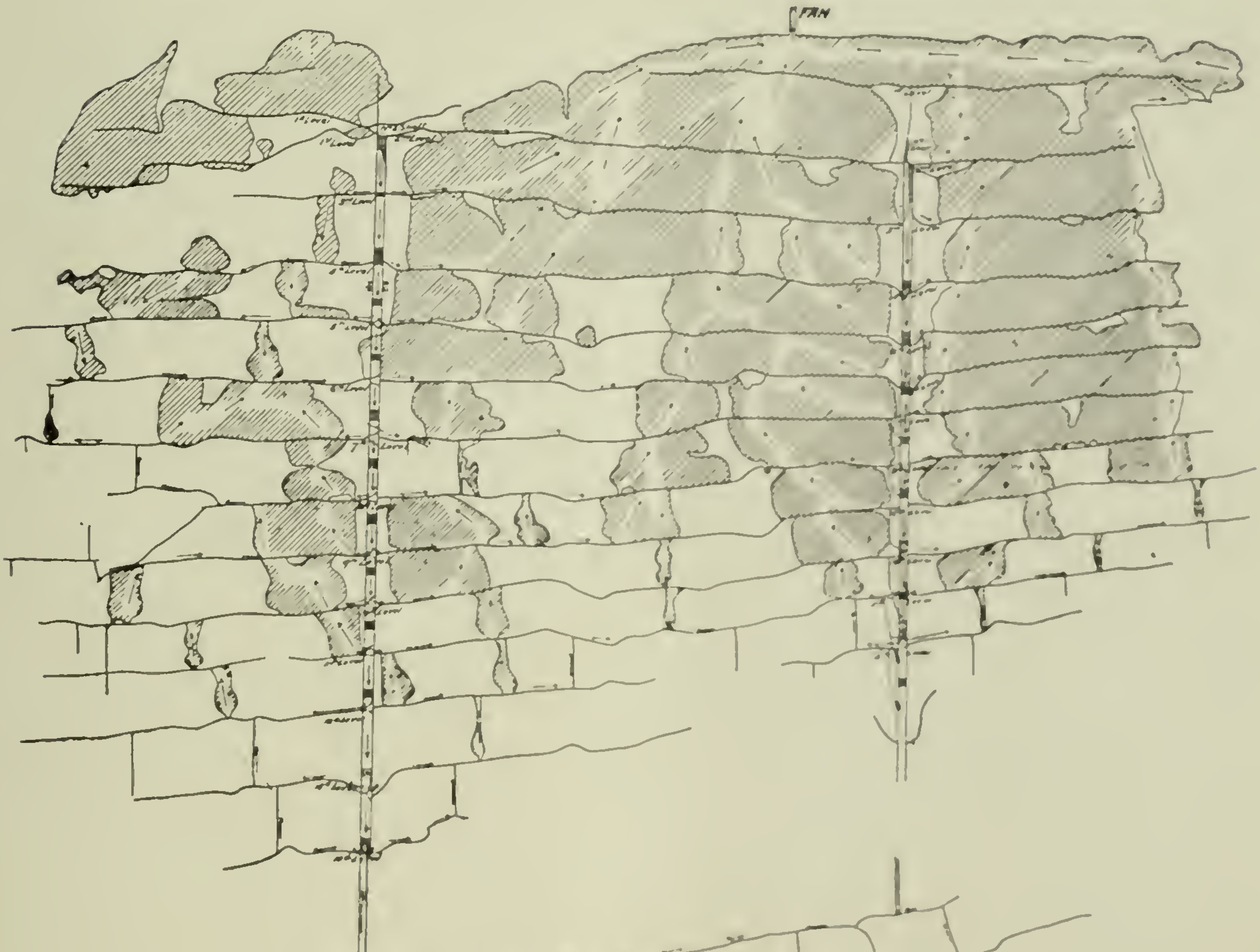


Fig. 9.—Ventilation Chart of Village Deep Gold Mine

Mines and Eckstein Groups alone, there are nine different fans already installed.

In a paper of this description it is realised that such a wide subject cannot fully be dealt with, but it will have served its purpose if it has made clear to the members of this Society that in the great undertakings of the future, if we are to get efficient work from our labour, the very best underground conditions must be provided.

The President: This is a most valuable paper. I have, however, a certain grievance against Mr. Whitehouse—at the annual dinner I said the mine managers were very slack, yet here we have a mine manager coming along immediately afterwards and giving us an excellent paper. The striking point about this paper is its wealth of experimental detail—detail which we did not get in the previous one on ventilation. I do not think I shall be hurting anyone's feelings if I say these experimental results are very satisfactory indeed, and are much more reasonable than those we received from the East Rand Proprietary Mines. These results show that Dr. Krause's Commission on Mining Regulations did not go far enough : in fact they had not the courage of their convictions. I do not know if Mr. Whitehouse can tell us how many cubic feet a minute per head he is giving. I think it must be somewhere about 70, whereas Dr. Krause's Commission recommended a figure which works out at only about 25 cub. ft. per man. Therefore I think that no mining house has any right to complain and say that Dr. Krause's Commission was too strict. I think this paper shows that it will actually pay the mines to give 70 cub. ft. or more. They may have to spend a halfpenny per ton in doing so, but they will save a great deal more than that in efficiency. I would like to move a hearty vote of thanks to the authors.

Mr. J. Whitehouse (*Associate*): I think I may say that the number of cubic feet for every person underground is very considerably over 70 a minute ; nearly double I fancy. Provision, of course, has been made in the scheme for the natural increase in the workings.

THE MINE DUST PROBLEM.

(Read at January Meeting, 1911.)

By Dr. J. L. Aymard (Associate).

REPLY TO DISCUSSION.

Dr. J. L. Aymard (*Associate*): I wish to thank you for the kind remarks which have been expressed from time to time with regard to my contributions to this extremely important and urgent subject. It is somewhat difficult for a medical man to tackle a mining subject with success, opposed to such a combination of professional intelligence. The doctrine which I have preached both privately and publicly for many months past as to the undesirability of circulating dust-laden air throughout a mine, either slowly or rapidly, has I am glad to say, taken effect. I am told that the one mining subject I may dilate upon is dust, but that with regard to ventilation I must leave it alone. If you remember in my preamble to this subject I assumed no special knowledge. I have since come to the conclusion, that neither does anyone else, unless it be the mine manager, who under the shadow of the mining engineer does not care to air his practical knowledge in print.

Mr. Tom Johnson accuses me of using a special terminology of my own, which he is unable to understand. I have often listened to this gentleman criticising other papers, and it appears to me that he does so more from a feeling of necessity to say something, than from any real desire to assist the subject or the contributor. With regard to my remarks on the small fan which I used to cause an upward movement of the air in the upcast shaft of my model, I merely said I had not got a proper exhaust fan but made use of what came to hand, and with regard to my reference to the ventilation of the Village Deep Mr. Johnson accuses me of passing the foul air into the Village Main. I was not discussing the ventilation of the Village Main. Mr. Gascoyne states that I think managers ought to sandpaper their drives. Here we have another valuable contribution in the form of criticism on a par with Mr. Johnson's remarks. Mr. Gascoyne says what is wanted is area in a drive and that it is not the height that makes it better for ventilation. Mr. Gascoyne knows perfectly well that I asked for no absurd conditions such as he depicts, but only a decently dressed roof or hanging wall; one which will offer the least resistance to the passage of air and give an element of safety for the workers. Some of the mines in this respect are a disgrace. One member says I am entirely wrong in saying that disused stopes should be left open because gases accumulate there. If the writer will demonstrate before this Society the retention of mine gases in a vessel with neither top nor bottom I for one will be quite interested. A stope is usually open above and below, and gases collect only in such places as winzes and the bottom of shafts.

Another member stated that I was wrong as to the usual method in which air enters a drive from an upcast shaft. I said it entered along the roof. If this member will renew his acquaintance with Rand conditions he will see the error of this and other remarks which he made. I ask him however in returning after a blast, would he advise the men to crawl along the floor of the drive! When making experiments on this subject some years ago I found that only by keeping my head above the smoke could effectual progress be made. This I think goes to prove my theory. I must once again take Mr. Johnson to task. Some months ago he stated in the *Journal* that my destructor was not original, and when shown my new pattern he again repeated the remark but could not substantiate his statement by any facts. Two of my medical brethren at different times have made precisely similar statements which they are quite unable to confirm. I most strongly resent such statements, and I repeat I am not in the habit of copying anyone, having sufficient intelligence and perseverance to make use of my own brains in original form. I hope, therefore, that this will be the last occasion upon which I shall have to refer to this matter.

Mr. Griffiths raised one point to the effect that dust once precipitated upon the rock easily became dislodged and he suggests the use of some adhesive substance. I think this is quite unnecessary, dust once adhering to a damp rock surface cakes quite hard and is not easily dislodged. The ordinary slime pit walls do not readily part with their component parts otherwise the result on the surface would be disastrous.

At our last meeting Mr. Purser expressed regret that I had not discussed his invention and suggested that I had departed from some prearranged understanding with him. I wrote to Mr. Purser inviting him to send his invention to be shown when I read my paper, and intimated that I did not propose to criticise it myself as I did not think it would be becoming on my part. Mr. Purser did not send a reply until the day before I read my paper, therefore, the few remarks I did make were not printed in the paper nor did the reporter consider them of sufficient importance to insert at the time of reading. Mr. Purser in his letter says, "referring to your letter of the 16th I shall be glad of any criticism you can offer provided it is reported in the daily papers, etc." Now the remarks I made were these: "Many of you are aware of the nature of Mr. Purser's clever invention, but it would be quite out of place for me to either criticise from his blue print or model." Now I fail to see how the above can in anyway be construed into a breach of faith on my part. It is a matter for regret that Mr. Purser's invention has not been supported at any of these meetings by even a representative of the East Rand Proprietary Mines. I need only add that I quite agree with Mr. Purser's remarks as to the four points that ought to be aimed at, and in my simple device, applicable for both kind of drills the whole of these four points have been complied with. The Nourse Mines' management is, I believe, willing to supply my destructors for a small sum to any mine managers desiring to adopt the system, but I strongly urge that under these circumstances, a competent man should be sent in the first instance to see the full system of working; one day spent under the able guidance of Mr. Murray, of the Nourse Mine, will give a better idea than any amount of paper explanation.

One member caused considerable amusement by referring to a portable air purifier I think he is inventing; he should not be discouraged, as I am convinced that there will be a great future for some simple electrical-driven contrivance on these lines.

One member regrets that I have not more fully gone into the medical aspect. Advanced work in this subject can only be undertaken by those who have access to a well equipped laboratory and who are specially licensed to experiment on animals, and until that work has been undertaken, I fear it would be only waste of time to try and elaborate the physical symptoms. Upon one point alone I would like to make a statement, because it is one which has got such an extensive hold not only upon employes but employers and even medical men. It is as to whether mine gases have any permanent effect upon a miner's health. I unhesitatingly state that in my opinion such gases have no permanent effect upon the miner whatever. My long connection with mines where no drills were used, but where the hard dolomite involved the extensive use of explosives, leads me to the conclusion that no more healthy body of miners were to be found under these conditions. The anaemia which one so often sees on the Rand was conspicuous by its absence. I therefore give my unqualified opinion that the gases produced by blasting are in no way permanently detrimental either directly or indirectly to the health of the miner. The argument that in combination with the dust permanent damage may ensue has no fact to support it. Mr. Hildick Smith has very kindly supplied me with several rats from the Ferreira Gold Mine. The really old rat I have here shows little sign of an abnormal lung condition, nor any of the others, even after long life in the mine. There is only one test for a doubtful case of miners' phthisis, and that is the X ray.

The fact that we are all agreed on this question of prevention is indeed a hopeful sign. Every one of us could have done better than we have, and we might have hastened matters to the logical conclusion. Great irritation has in the past been shown by all parties concerned when brought face to face with their individual shortcomings. I appeal to you, gentlemen, not to put off till to-morrow what you can do to-day, bearing ever in mind that the prevention of this scourge must begin immediately. You ask me what is being done. Well, you are all aware of the Parliamentary proceedings which have taken place. If the result of such proceedings delay in any way the beginning of the enforcement of precautionary measures then the waste of time will have disastrous results. I can only hope that wise practical regulations are being framed which will in the immediate future be put into force. There appears to me to be a very unnecessary delay about to be. or being caused, by reference to further commissions for information. I do not like this further waste of time in the least; I say waste of time because I am convinced that it will be. Surely every mine on the Rand has, or should have, its own competent medical man, and surely every such medical officer is in complete touch with the men under him. These medical men are able to furnish the mining houses with all the particulars within a week, which Parliament asks a year for. Here is where a deplorable waste of time is asked for, which only savours of a very ugly appearance of procrastination.

Prof. Wilkinson objects to the private examination of the men, and with him I strongly agree; there is no argument which can be brought forward of sufficient weight to support a private examination. The examinations should be straightforward and open if they are to be of the least value.

Again I very strongly condemn those who want to make a mystery as to the cause of the disease. A hundred further commissions or enquiries will only confirm the fact that 90% of phthisis is caused by the dust from the drill and the remaining ten from subsidiary causes, such as blasting and shovelling. This 90% can be stopped within one month from this date, if all are willing to co-operate and make a beginning. Effective proved means are now at hand and the cost is trivial.

You are all aware that many trials have been made during the past twelve months for the prevention of rock drill dust. The work of instructing the men, of seeing the work is carried out, and of seeing the machines are in order, must be entrusted to one or more permanent special men with authority and with an efficient staff of boys working under them. The shift boss is not the man for this work. It wants a man, or men, who will go to all parts of the mine. The post is a very arduous one, and until all the drills are equipped a very dangerous one. Mr. Murray, who has done such excellent work during the past six months will, on my advice, soon have to relinquish the same, as he is developing the disease even in this short period. I do sincerely hope that this excellent example now set by the Nourse Mines will be rapidly copied in all its details throughout the whole of the mines without waiting for the formulation of regulations. What can be done in one large mine with success can be done in all mines. It only remains for you to assist with all possible speed, those whose duties it is to move in this great question.

I have not referred to the Phthisis Bill, because it is of secondary consideration to the question of prevention, now that it has been postponed indefinitely. Miners' phthisis is a preventible disease, and bears no relation to any general insurance scheme. General insurance can wait, but miners' phthisis can not. Why tuberculosis is to be included in the Miners' Phthisis Bill I fail to see, as in my opinion the connection is only incidental and not in any way related by cause or effect.

The President: We must congratulate Dr. Aymard on his reply. He has certainly given us some very straight speaking indeed, and I think I may safely say that it was needed.

MINE DUST SLIDES.

Prof. J. A. Wilkinson (*Member of Council*) here exhibited a number of slides showing the character of the dust prevalent in the air, from samples taken underground (see p. 553, and pp. 496-7 of the April *Journal*).

PROFIT PER FATHOM.

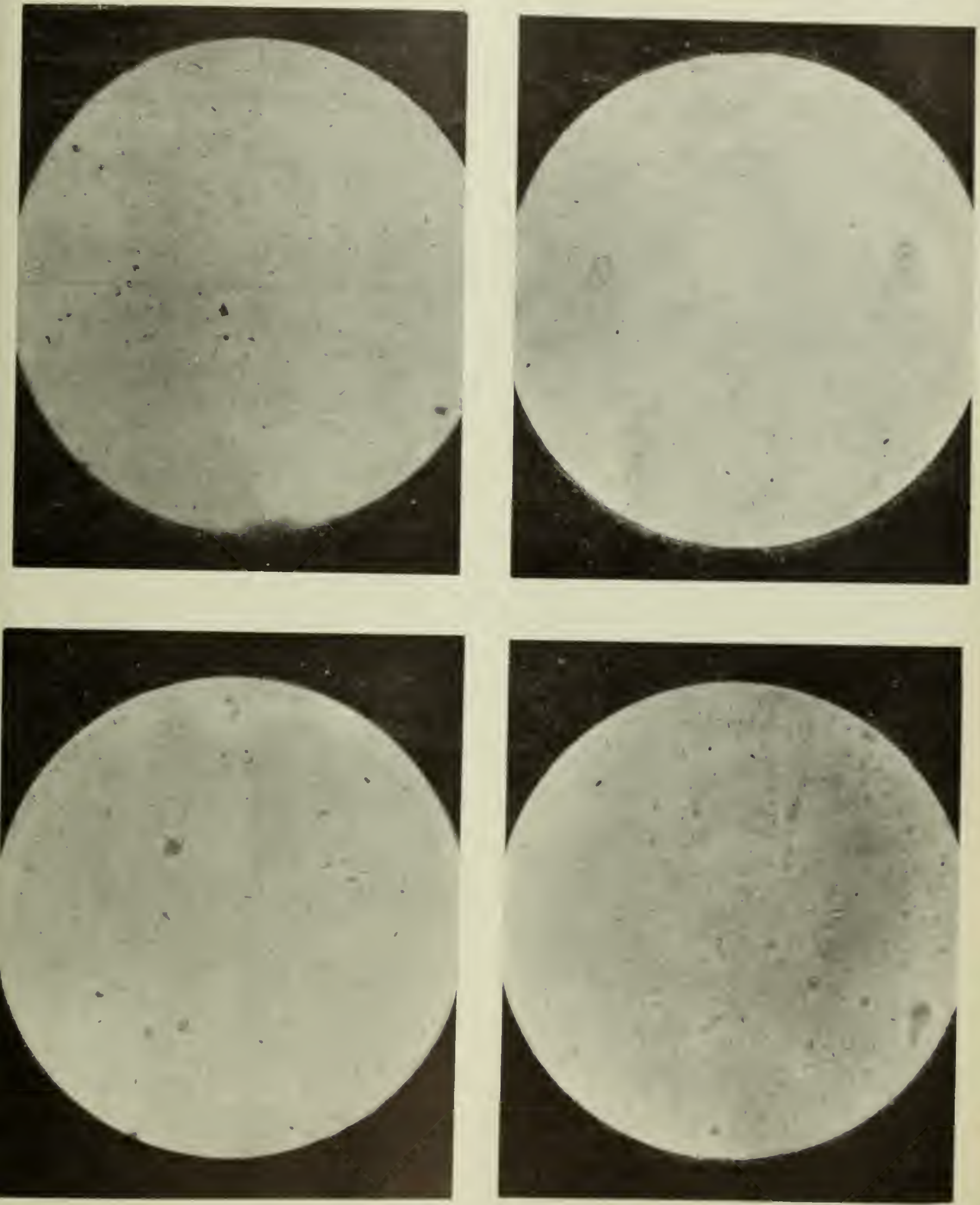
(*Read at February Meeting, 1911.*)

By R. E. Sawyer, A.R.S.M., A.I.M.M
(Associate).

DISCUSSION.

Mr. Tom Johnson (*Member of Council*): The author, like all other advocates of the square fathom system of distributing costs, starts with the poor despised ton, gets to fathoms and back to the ton again. It seems strange to me, but if the ton is not good enough, why do all square fathom-advocates use it as a base?

The author thinks that we cannot get the actual costs of any particular stope; certainly he



Dust from Rock Drill. ×130

has not helped us much. I am sure that it is possible for a manager knowing his average costs for the different branches of work, on looking through a stope, to figure out in less than five minutes the profit (to within 3d. per ton) to be made from that stope until conditions change.

Take the author's tables. He shows in Table I the average costs of different classes of stopes, hammer, large machine and small machine, and uses these average costs for stopes of varying widths in each class. This is very far from being good work and is nothing like as close an approximation as at present obtains on many mines. There is a mistake of 6s. short per fathom in the reduction costs for the hammer stopes. Why are small machines used in narrower stopes than hammer boys are?

In speaking of fathomage the author lays stress on the difficulty of getting it correct, yet he builds a system on fathoms.

In Table III the author has 1W 3W stope 133 in. wide costing 92·8s. per fathom and 6W 1E stope 137 in. wide costing 42·3s. per fathom (or say 1·5s. per ton difference in stores and labour costs); but in Table I he uses the average cost for that class of stope—that is 65·5s. In the same class of stope we have a 55 in. stope costing 61·9s. for reduction charged up the same as a 143 in. stope costing 160·9s. for reduction, that is 112·7s.; a 37 in. or an 80 in. hammer stope is charged for reduction 60·1s. instead of 42·75s. and 90·0s. respectively; a 26 in. small machine stope and a 52 in. one costing 29·25s. and 58·5s. respectively are charged up at the same figure—that is 41·8s. The same thing applies right through. This is not the way to find the cost of working the different stopes.

By dealing in this way with average costs of a class of stope, instead of different individual stopes, the author's figures and curves lose some of their value. I notice that the curves represent stopes up to 18 ft. in width or 54 tons per fathom. If we take the 100s. curve it comes to 1·85s. per ton. The author has twelve charges against this, of which tramming and hoisting take 1·5s., leaving 0·35s. to pay for the breaking and other nine charges enumerated. Shovelling, tramming and hoisting are cheap at 1·5s., but the other work is cheaper. Although I criticise the lower parts of the curves and the use of them for a certain class of stope, there is quite an amount of useful knowledge to be gained from them if used for the usual stoping widths and the cost of individual stopes instead of the average cost of a class of stope.

For instance, take the 100s. curve again. To get the figures for a 6 ft. stope and a 3 ft. stope, I find on figuring out in my own way, that the author's values shown as necessary to enable those stopes to be worked at 100s. per fathom each are as nearly correct as one can get. I allow 50s. for the breaking costs in the stope and 50s. for the other charges in the 3 ft. stope and 40s. for breaking and 60s. other charges in the 6 ft. stope. The values are 144s. and 198s. respectively; this gives the 6 ft. stope 54s. advantage in gold value, and 10s. advantage in breaking per fathom, equal to 64s. From this 64s. there are tramming and hoisting charges 1·5s. per ton on 9 tons equals 13.5s., also 45s. reduction charges making 58·5s.; to which must be added the charges for the extra 9 tons for pumping, administration, etc., so the total charges approximate closely to the 64s. difference.

From the higher parts of the curve it can be seen how quickly the value must increase to make narrow stoping pay. On the 100s. curve the difference in value between the 6 ft. and 3 ft. stopes is 5s., that is a 3 ft. drop in width; but for the next foot alone the difference is 4·75s.—practically the same as for the 3 ft. From the curves one could, with a little figuring, get the most economical lowest stoping width. On calculation a mistake or two will be found in the value line at the bottom of the curve sheet.

Also in Table I general charges per fathom on reduction will be found to be 11s., whether the reduction charges proper are 30·6s. or 101·5s., and whether the fathom comes from a 26 in. stope or a 143 in. stope. In fact Table I bristles with superlative accuracy.

Up to this point I have disagreed with the paper, but I can heartily agree with the author's assertion that it does not always pay to shut down stopes of less value per ton than the average cost per ton for the reasons he gives, and this is one of the faults to be found in Mr. Marriott's square fathom system where all stopes must at least equal the average working costs in value.

I have in mind a mine where they are trying to follow out Mr. Marriott's idea. This mine, before the trial of the square fathom method, never had its mill shut down for want of rock for some years. The square fathom method had not been in use long, however, before that mill could be shut down on Sundays. Now it is a certainty that that mill could be run on rock of less value than average cost and make a profit on these Sundays—or perhaps they hoped to forestal the Government's idea of shutting down the mills on Sundays. I know other instances where the mill cannot be kept going, and yet 10s. rock would return a profit on the amount necessary to keep the mill going full time, even if the average costs be 20s. per ton.

I have seen stopes in several mines, in which part of the stope width is of better value than the remainder. The best portion is worked: it may be of a value of 7 dwt. or more per ton for this portion of, let us say, 42 in. stoping width: under or over this may be another 42 in. of 3 or 4 dwt. rock. Now looking at the figures casually one would say that this 3 or 4 dwt. rock will not pay with costs at an average of 20s per ton, but it will. If one stops to consider a moment, it is found that a 7 ft. stope is cheaper to break than a 3½ ft. stope, so there are no breaking costs against the low value rock: also that there are other costs that are small. Taking all these things into consideration, it pays to break this low value rock. There are cases where rock of still lower value can be milled at a profit, supposing a plant fully staffed to handle 4,000 or 5,000 tons more than it is doing per month. Then in the case of stopes, as mentioned above, 2 dwt. rock will provide a profit; for as I said before, such rock costs nothing to break, in fact saves money: also, all other labour charges, except shovelling and tramming and emptying vats, are paid for, so that the cost against this 4,000 to 5,000 tons is at most 5s. per ton.

It must be distinctly understood that I am not saying that the rock can be broken separately and handled for 5s. per ton, but it can be under the conditions mentioned.

Now for the claims the author makes for his system.—I.—Granted that the gold contents are the same: even if stope width is not mentioned, it is nevertheless a very important factor. For instance, a 12 ft. stope of 2 dwt. value carries as much gold as a 4 ft. stope of 6 dwt. value, or a 2 ft. stope of 12 dwt. value, but I know which I should prefer to work. Again, a stope of 4 feet width, with 3 ft. of reef with such a value as to make the stope worth 5 dwt. per ton (equal to 240 in.-dwt.) is not so good or valuable as a 6 in. leader stope of the same in.-dwt. value. Although the gold per fathom is the same, it is certain that the profit per fathom would be greater from the 6 in. leader stope. The old Jumpers and other mines have turned non-paying places into paying places, although containing the same amount of gold per fathom — not by changing their system of distributing costs, but by changing their system of working.

It is this possibility of having non-payable and payable stopes containing the same amount of gold per ton that has caused Mr. Marriott and his followers to miss the whole crux of the question.

II.—Profit per fathom conveys more to the manager than any other figure. Before believing this, I should like a little confirmation from some of the managers. The fathoms broken in any stope fluctuate more, month by month, than the tons shovelled do. If a manager wanted more rock from a particular stope, he would probably get it from the rock already broken, that is, he would skin that stope. He could only increase his breaking by putting a better man in the place, or by increasing the machines, steps which are not taken in a hurry. To my mind a manager has a better idea of his cost per ton than cost per fathom, and he gets a fair idea of the money coming in from the mill and cyanide works assay values. The author says the manager knows what stopes to push when the profit falls. How does he know when the profit falls? He can not tell from any of the figures the author uses. If the value contained in each square fathom of a stope remained the same, the thickness of the reef, the stoping width and the amount broken per shift also, then he would have a chance, but unfortunately they do not. So the sensible manager will work on the ton basis. The author himself shows us truly enough that stopes of the same width vary greatly in cost for stores and labour per fathom, not so much because of anything inherent in the stope, but because of the different value of the human element as a director of rock breaking appliances. Seeing that stoping widths are nearly always given in inches and not feet, why not use this rule:—tons per fathom equals stoping width in inches divided by four?

III.—That it will help the manager to get away from the terrible bogey "grade." There seems, to my mind, to be a misconception of the meaning of grade. Grade, to me, means the handling of rock that will give a return greater than the cost of handling that rock, that is, return a certain amount of profit. Surely the author admits that in his system we must look to the same thing. It is impossible to get away from grade. In the author's system you must get a fathom of rock of value greater than the cost of handling—"grade" represents the value. A 20 ton per fathom stope giving a profit of 6s. per ton, gives only a profit of £6 per fathom and no more, even if you do say "profit per fathom."

I must say I have not yet heard, or seen, any valid argument from the square fathom advocates to cause us to alter the method of recording our working costs. It is not this that needs altering, but the manner of doing the work that should be altered so as to better results.

I have purposely used the word "rock" instead of "ore" or "reef," seeing that we mill so much that is not ore. If we milled a less diluted ore instead of the mixture we do, there would have been no square fathom controversy.

FAULTING PHENOMENA IN RAND MINES.

(*Read at February Meeting, 1911.*)

By G. H. Beatty (Member).

DISCUSSION

Mr. Tom Johnson (*Member of Council*): I must congratulate the author on his interesting paper, which is on a subject that constantly crops up in work, but which to my knowledge has not been brought to the notice of this Society before. I agree with much that the author has said, and on some points would like a little more enlightenment.

The terms "upthrow" and "downthrow," when applied to normal faults are much used by other than ordinary miners: although the term upthrow is not correct, it is very expressive, and is in use the world over. The one term downthrow would suffice if the direction in which the fault did downthrow was mentioned. In the diagram D.F. 2 is a downthrow S.W. dipping 85° and D.F. 3 is a downthrow E. dipping 85°. I agree with the author that normal faults must be mostly due to gravity, and reverse faults to pressure. If note be taken of drive pillars in some of the mines where the pillars are being crushed, the action that causes normal faults can be seen. A break, or in other words a fault, is seen going right through the pillar and dipping towards the drive, the southern portion of the pillar slipping down the hanging of the break. The amount of faulting can sometimes be seen on the bottom of the southern portion of the pillar. Then if props are noticed, some will be found which give a good illustration of how reverse faults are formed. It will be seen on those props that break by crushing and not buckling. Anyone whose business it is to deal with faults, should take all opportunities of examining the ground both above and below the reefs, noting the characteristics of any reef matter near to and also of the reefs being worked. When crossing faults in drives, if the reef has gone out of sight, an extra hole put up in the roof near the dip side, and one in the foot near the rise side gives information, as the holes sometimes expose the reef. Care should be used when deciding whether a fault is downthrow towards or away from one, when the break is vertical or nearly so, as most fault planes vary a little in their dip in different parts. It is a good precaution to have the places surveyed and plotted up to date when dealing with a fault that is new to one, as it is bad policy to drive in all directions just for the purpose of keeping the place going. It is better to be sure even if it does mean the place standing for a shift or two. Faults crossing a rise or winze are more difficult to solve with the eye alone than the same faults in the drive, the pitch of the reef complicating the problem somewhat. The author says "almost every fault will be found to have an increasing or diminishing throw, ultimately dying out in both directions." My experience teaches me otherwise as main faults diminish both ways, but as the author remarks "very often a big fault is accompanied by smaller faults of similar trend and dip." These smaller faults start at the big fault with maximum throw and die out in the distance. I have seen this often with both faults and dykes, and there are more branch faults than mains. The small faults do not always have the same dip or throw the same way as the parent fault, as the author says. For example, see his sketch between the underground shaft and D.F. 3; there on the 6th level South Reef a downthrow west fault is seen coming off D.F. 2 which is also a downthrow west fault, but on the 5th level there is a small downthrow east fault coming off D.F. 2; that is two faults of opposite dip and throw coming from the same parent fault, forming what is called a trough fault. It will also be seen that the fault crossing the 5th level dies out before reaching the 6th level. The author says on page 369 of the February *Journal* that "it would be expected to find on the underlap portion corresponding to the section Fig. 1 dip faults similar to those of the overlap." This is very true, and I should like to know the author's opinion why they are not there, for it is certain they should be there if the faults were formed before the dyke. Also I would ask where is the reef that has moved horizontally to the north, as there should be some trace of it. Possibly the westerly movement accounts for it. I think all the ground between the George Goch and Metropolitan faults moved westwards a considerable distance. I heartily agree with the author that a deal of inefficient driving is done and as to the benefits to be gained from straight drives. I do not think that any official should, in a manner, be forced to work just for his own term of office, instead of for the ultimate good of the mine itself.

Coming back to faults again, it is always interesting to hear the experiences of others, whether successes or failures there is something to be learnt.

I have seen some very stupid work in connection with faults in both old and new work in mines here and elsewhere. I have seen cases on the Rand, in which very prominent men in the mining world have been concerned, that would be nothing but disgraceful if a student just from college had been responsible.

It is bad work when mistakes made on one level where the signs of what had happened were quite plain, but the same mistake repeated on two levels with the same fault is inexcusable. I have noticed this in several mines, and the only conclusion is that the perpetrators would rather trust to luck and keep the places going, than let one stand while the truth is found in the other. It seems to have been the aim to keep moving right or wrong, like the mine-captain who used to blast down the hanging to find rock to keep the mill going; it did not trouble him that there was no gold in it, as work was going on.

I remember two cases I saw when I first came to the Rand. One was a drive west in which the reef had been lost somehow and could not be found even by boreholes both up and down. There was no fault, the dip of the reef having swung to the north a little, but the drive had been driven straight ahead and over the reef. A new man coming in traced the reef into the floor and got the mine-captain to have the machines brought back, and started working bearing northwards a little and the reef was found to be continuous. One may ask how was it that the bore holes did not find the reef seeing that one went into the footwall. It did, but the reef was not noticed, it being so close that the guide pipe at the mouth of the hole was actually in the reef. They were mistaken because they thought they were dealing with a fault where there was none, and did not think that the reef they were chipping up was the one they were searching for. In the same mine I saw another case in which boreholes were used with no result beyond perplexing the officials. This case was in a back stope which had reached disturbed ground. A short cross-cut was put in to get room for the diamond-drill to work beyond the fault, as they thought, and a bottom and top hole were drilled but no reef found. The man who had shown them what was wrong in the drive was asked to look at this stope. He found the stope to be on the main reef, the reef to be up beyond the fault, so of course the down hole could not find reef and the top hole was angled back across the measures and crossed the fault, missing the main reef on the north side of the fault and the south reef on the south side of the fault. This was his reading of the indications, and subsequent work proved him correct.

I mention these two cases to show that the idea that boreholes will prove a fault is fallacious unless commonsense is used in directing operations, also that in neither of these cases were boreholes necessary.

It is to be hoped that this paper of the author will stimulate interest in the study of faults. It is a very fascinating study, and it is really very little trouble to solve most of the faults met with. A study of the author's paper would amply repay anyone.

THE AMALGAMATION OF GOLD IN BANKET ORE.

(Read at March Meeting, 1911.)

By W. R. Dowling, M.I.M.M. (Vice-President).

DISCUSSION.

Mr. H. A. White (*Member of Council*): The conclusion at which the author has arrived respecting the reduction in plate area, feasible without serious decrease in the extraction by amalgamation, is undoubtedly satisfactorily demonstrated by the careful experiments that have been performed and the proof is emphasized by the results he reports from both an outcrop and a very deep level mine. For the ordinary case along the reef I consider that he has shown that the stationary plate can be used, with the suitable precautions he has mentioned, to give approximately as good results as the more expensive shaking plate. It is curious that before a single tube mill was erected on the Rand their introducer stated that the stationary plate would be superior to a shaking plate.*

The experiments above referred to are, as is too frequently the case here, too closely confined to the immediate point at issue, and I would suggest that the Mines Trials Committee might well pursue the matter further. For example an 18% grade is selected as the most suitable and a water percentage of 55. It would be valuable to know the exact effect of varying the fall and the water percentage: further it would not be too difficult to have a plate cut up into strips a foot in width and mount them on a table in such a manner that each strip might be cleaned up separately. It would then be possible to ascertain the amalgam caught by each foot of the plate and thus determine what area of plate per ton of ore per diem can most profitably be employed. If some such course is not adopted some future discoverer may come along and shew that 100 sq. ft. of copper suitably disposed can do the work the author now requires 1,700 sq. ft. for, and still more copper may be scrapped, more in triumph than in sorrow. As presented, the figures from the Simmer Deep suggest that an increase in extraction may be expected by eliminating amalgamation altogether—in this case no doubt the increase reported must be entirely due to the finer grinding mentioned, though it is curious that the slime value has not increased. It is no doubt true that on several mines experiment on a small laboratory scale indicates that total extraction would be scarcely affected by omitting amalgamation. Admitting that in many cases amalgamation is not absolutely necessary to securing the highest possible (theoretical) extraction the question remains why do all the mines here retain it in spite of the many prophecies of doom in the past—prophecies still current with the exponents of the "New Metallurgy."*

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* H. S. Denny, Proceedings of this Society, Vol. IV., April, 1901, p. 227.

The answer to this question brings me to the only point in this interesting paper where I feel bound to join issue with the conclusions arrived at. The reduction in extraction by amalgamation consequent upon neglect of the tube mill overflow is regarded by the author with too friendly an eye as being in itself not necessarily an evil. I do not entirely agree, and I will briefly outline the case for the other side. One of the standard requirements upon the mine management is a uniform monthly production, and as fluctuations adversely affect the average value of shares, this is not wholly unreasonable. The establishment of a gold reserve is only a slight help, for the effect upon the value of an investment of a drop of 1,000 ounces in a reserve must be nearly as bad as that of a similar fall in output. The maintenance of an even output has been in the past estimated as costing more than 6d. per ton extra for mining owing to excessive development ahead of requirements, extra tramming costs from numerous levels and other reasons. Whether this figure be excessive or no it is obvious that regularity of profit is worth something and in order to secure it some idea of the gold actually won must be available daily. From a given amount of amalgam it is possible to gauge the yield in fine gold within 1% or 2% and the amount of amalgam to be obtained by steaming the plates when required may be estimated without very large error. In this manner the forecast of results is very much more certain than is the case with the yield from the cyanide works. There a variation of 10% between theoretical and actual extraction for any one month is easily possible, and if all the gold were derived from this source the fluctuation on a low grade mine might be greater than the average monthly profit unless extra clean up expenses were undertaken in order to palliate the evil. A reduction manager, even as things now are, is only prudent in having his cyanide clean up finished before the end of the month.

This consideration alone demands that all the gold possible shall be won by amalgamation even though some slight loss of profit is involved in securing it. I will attempt to shew that such loss is problematic at least and that a small gain in actual extraction sufficient to shew a profit on extra cost of amalgamation will be secured in most cases.

It will be evident that the point at issue is not the total abandonment of amalgamation, which the author in no way supports, but whether the overflow from the tube mill classifiers shall go over plates or not A certain proportion, depending upon coarseness of preliminary crushing, does not pass through the tube mill circuit, and consequently will not go over plates at all unless special arrangements are made, assuming that the mill plates are discarded. In this way a loss of about 15% of the possible amalgam recovery is shewn by the author in the case of the Simmer and Jack plant. This loss may be considered under four heads :

(1) Fine gold to sand plant.
(2) Black sand to sand plant.
(3) Fine gold to slime plant.
(4) Black sand to slime plant.

(1) The fine gold sent to sand plant will no doubt be very speedily dissolved by the strong cyanide solution, but will either cause a small loss in dissolved gold in the residue or occasion extra washing treatment in consequence of the higher gold value of the solution. If the perfectly collected sand from the Caldecott table be compared with the usual more slimy product from vat collection, no doubt this real disadvantage will be hopelessly concealed and the only useful comparison will be with everything unchanged except amalgamation. The cost of recovering the extra gold from solution is greater than from amalgam and this extra expense must also be borne in all four cases.

(2) As black sand particles caught on an amalgamated plate are of all sizes down to fine slime it is probable that the tube mill outflow will carry some away to the sand and slime plants which would be retained on an amalgamated plate. As such particles yield an extra profit to differential treatment a small loss must be caused by their neglect.

(3) The very fine free gold sent to the slime plant is probably very quickly dissolved : in decantation plants this will occasion greater loss in dissolved gold sent to the residue dam or will require greater vat capacity and extra dilution, that is must increase either the capital and working costs or decrease the extraction obtainable. The author's presentation of this point almost implies that the better washing owing to denser settlement is in some way connected with absence of overflow amalgamation. If, as he leads us to believe, the difference in original value of slime amounts to 0·3 dwt. the actual loss on the residue (35% moisture considered) is 0·011 dwt. or ½d. per ton. In some cases the difference in value of slime before and after amalgamation is

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* See S.A. Mining Journal, April 15, 1911, p. 251.

much higher than 0·3 dwt. (Durant* shows 1·75 dwt. difference in one case.) I have obtained differences of 0·5 dwt. Even where vacuum filters are used this point would not be absolutely negligible as absolute displacement is never possible, not only because of the necessary "dead" spaces but because of adsorption and surface concentration effects. This is illustrated by the washing diagram on the Crown Mines filter plant and by common experience in removing zinc salts from filter pressed gold slime. In this case also there would therefore be a slightly greater proportion of wash solution necessary and a similar though smaller increase in the expense.

(4) It might be doubted that black sand could be fine enough to get away to the slime plant, but I have a case where similar material going 40 oz. to the ton and in quantity equivalent to about 5% of the total value of the slime is collected from launders after the sand return spitzkasten, though the value of the slime is less than 1·5 dwt. and it contains less than 0·5% of +200 sand. If an increase in this fine black sand is caused by omitting the amalgamation of the tube mill overflow, the undissolved gold in the slime residue will be a little higher because such material requires a longer treatment. It is possible that part of the difference of 0·02 dwt. in the undissolved gold of the slime residue quoted by the author is due to this cause for the East Rand Proprietary Mines, results given by Mr. Jory† where a similar collection arrangement is in use, shew that their washed residue value remains at 0·08 dwt. and all their pulp was amalgamated at the time and the other conditions are not apparently more favourable. Now I do not pretend that any of these items amounts to very much, I only contend that in the aggregate they would slightly more than cover the increased cost of complete amalgamation. It must be borne in mind that the small normal variations in extraction percentages common to all gold mines are yet large enough to hide such small losses unless they are specially searched for, but they are not the less certain on that account. I am fairly safe in saying that no special black sand plants would have been erected on the mines with which the author is connected if their warrant had been searched for in the total residue value only.

I would point out with regard to cost of extracting the extra gold on overflow plates that a small extra elevation only is necessary and in few cases would a separate elevation be required. If we put the extra lift at 1 ft. the extra power required, all other costs being practically unaltered, would be about 0.03 of a penny per ton crushed. Under the specially favourable circumstances we cannot put the extra cost of mercury higher than 0·10d., the extra interest and depreciation on plates, tables, buildings, etc., will hardly exceed 0·20d. and the extra labour 0·67d., although in a case I have in mind owing to gold law regulations no extra labour costs will be involved. These figures are of course only estimates, but even if in special cases they had to be doubled I am under the impression that I have shewn the 2d. per ton will be profitably spent.

Mr. G. A. Robertson (*Member*): I am quite in agreement with the author as to the merits of stationary plates, and to install shaking plates nowadays means expenditure for no economic purpose. In this respect the Wolhuter and Knight Central Mines were amongst the first to follow the lead of the Randfontein Group, by adopting stationary plates

The author does not make his point quite clear when he states that diagram "A" is the same as at the Randfontein Central and Knight Central Mines There is only one elevation of pulp on the Knight Central, as on diagram B, there being however four stationary plates in the circuit and seven stationary plates for the overflow *en route* for the cyanide works When it is borne in mind that staging and platforms are required for the cones and feed launders to cones, also that separate trestles for the overflow launders are required, the cost of building a plate house is not a heavy item if designed on the principle of reduced amalgamated plate area, sufficient plate area being provided in the circuit to prevent the enrichment of pulp. On the Knight Central the water-tight plate house floor forms the roof of the pump and motor house. A fall of 10% to 12% for amalgamated plates was a more generally accepted principle by mill designers than it was by operators. During additions to the tube mill equipment of the Knights Deep Mine in 1906 the shaking plates (12 for 3 tube mills) were made stationary and lifted up at their heads to slopes allowing the pulp to run without any appreciable silting of the sand on the plates. The idea was a makeshift not intended to be permanent and as it successfully served its purpose for two months, the results appeared to me to justify experiment in connection with the distribution of pulp over the plate. The method of distribution then was a process of concentration, the pulp discharging itself through holes in the launder which carried the tube outlet flow.

The reduction of the amalgamated plate area as shown by the author is a step in the right direction and certainly shows a method whereby a saving can be made in capital cost and labour together with economy in mercury consumption.

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* This Society's Proceedings, Vol. III., June, 1[illegible], p [illegible]

† See this *Journal*, Vol. XI., February, 1911, p [illegible].

Though in most cases on the Rand large quantities of amalgam are "locked up" on the plates, it is very questionable whether this is necessary. Some mines clean up all the plates each month leaving sufficient amalgam on them so as not to interfere with the extraction. The application of 'elbow grease' to the amalgam left allows the plate to be made in good condition with quite a small quantity of gold amalgam.

The copper amalgamated plate in one piece generally in use could advantageously be cut in three pieces 3 ft. long to form a series of steps. Each part should be detachable and laid on the table in order of condition, the poorest piece being placed at the head. One spare piece of amalgamated plate should be kept and substituted for any well set piece of plate which can then be carried to the clean up room. There the amalgam can be mechanically peeled off, the plate scoured and put back on the table, thus avoiding a monthly event known on the mines as "clean up day." Mr. H. W. MacFarren[*] in writing of the drop plate method says "the only objection to drops is that they interfere to a slight extent with the quick dressing of plates" but I cannot see even this objection. A small mine belonging to the Manhatten Milling and Leasing Co. Nevada obtain a 92% extraction by "careful plate amalgamation" by the method above described. The advantages alone of facilitating the clean up by this method make it worthy of consideration on the Rand.

As regards the erosion of amalgam by coarse crushing we seem to be much divided in opinion. With the great efficiency of the tube-mill circuit as a mercury trap it is reasonable to suppose that the amalgam eroded from the stamp-mill plates would be caught on the tube-mill plates. Considerable difficulty has been occasioned in some quarters by amalgam erosion from coarse crushing, at others full amalgamation recovery is obtained when crushing through 64 and 36 mesh screens, and even these it is claimed do not mark the limit of coarse crushing screens in the stamp mill with full amalgamation results. The eroded amalgam in many cases is recovered through the medium of the much criticised and as often condemned spitzlutten. As the result of experimental work conducted by J. W. Elmer[†] at Tuolumne, California, the stamp-mill is now crushing through a 4 mesh screen, the pulp passing over an amalgamated plate, followed by a conical pebble mill and secondary amalgamation. These are but a few instances where it is claimed that what amalgam is lost in the stamp mill is caught on the tube mill plates.

It is doubtful if a saving in time in dressing plates is effected by having the plates out of the stamp mill. On the East and West Rand some years ago and when an ample supply of ore was available for crushing, the pulp from one battery was by-passed to the next plate there being no stoppages for dressing plates. The method may have been crude but if the plates had been a few feet from the mortar box the method would have been perfect. Similar devices are in use elsewhere.[*]

The question now is can we get a satisfactory extraction by cyaniding without previous amalgamation of the free fine gold or is it necessary to recover all the amalgamable gold before cyaniding With fine crushing and efficient cyanide treatment the author has shown that it is not necessary to recover the fine metallic gold by amalgamation, but the question here is what is the cost in cyanide consumption against mercury consumption for the recovery of this gold. So far the author has not shown that there is anything to be gained by only amalgamating part of the gold recoverable by this process, except perhaps in the case of fine crushing to 77·6% or 81% – 90 mesh product. In fact the author goes on to show that "where the ratio is lower" as at the Simmer and Jack the residues are "normal" "and a trace of fine gold can be detected by panning the final pulp," this when dealing with the residue from a "freely leachable sand." It goes without saying then that traces of fine gold would be detected by panning the final pulp with "slimy semi-permeable sand charges," that is of course provided the fine gold is in the sand charges before treatment. In the case of the Simmer Deep the extraction goes up to 93·5% on the fine crushing stated. While I admit that fine crushing costs are rapidly coming down and have not by a long way reached the limit, nor has the ultimate fineness been reached, nevertheless, in view of the recent light thrown upon the subject of fine crushing by one who ought by actual investigation to know,[†] I wonder if this fine grinding is profitable. No doubt under certain conditions it is so. It is particularly noticeable, however, that owing to the recent increased efficiency of the tube-mill as the fruitful outcome of previous wholesale condemnations, owing also to the increased capacity of stamps and to native labour troubles, there are large numbers of idle stamps on the Rand which the mines are quite unable to feed. This fine crushing is obviously not profitable to those mines which have an ample supply of ore in the mill bins. Crushing to 81% – 90 mesh in order to ensure a high extraction could not be effected on many mines to-day without a considerable reduction in the tonnage of ore milled and consequent reduction in profits.

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[*] H. W. MacFarren. "Practical Stamp Milling and Amalgamation, p. 96.

[†] "Pebble Mill Amalgamation, *Mining and Scientific Press*, April 30, 1910.

[*] H. W. MacFarren, "Practical Stamp Milling and Amalgamation, p. 94.

[†] See Reply to Discussion, "Grading Analyses and Their Application," this *Journal*, Vol. XI., Dec., 1910.

The considerations which enter into the problem of double stage crushing and getting, say, a 93% extraction with or without full amalgamation results are simply mercury consumption against cyanide consumption for the recovery of the fine metallic gold. It would seem as if the author over states his case when he puts down the disadvantages of recovering the fine metallic gold by amalgamation to the necessity for "extra capital outlay and buildings, the capital locked up as amalgam as setting to the plates, the increased area of amalgamated surface to guard, the consumption of mercury, the labour of dressing and the additional elevation of pulp necessitated by its fall during this amalgamation process." I admit the extra mercury consumption, but nothing else, and then take as a basis the layout in the proposed method "E" as an illustration in the saving of "extra capital outlay," etc. This design possesses the advantages: (1) The pulp is fairly well comminuted before being pumped, thereby economising in pump liners which become more rapidly abraded by coarse pulp than by fine pulp. (2) Part of the fine sand, slime and a proportion of water as these come from the mortar box gravitate direct to the cyanide plant. The disadvantages of the design are the unnecessary capital expenditure for excavation work, else double pumping is a necessity, and a questionable proposition as to whether a single classifier for classifying both stamp mill pulp and tube mill pulp gives the best results. In a recent criticism on diaphragm classification I confronted Mr. G. O. Smart* with two difficulties which went unchallenged, viz., (1) the uneven distribution of pulp to each cone, and (2) the uneven delivery of pulp flow from the pumps. Both these questions had a direct bearing on the final results of diaphragm classification as outlined by him. These questions, however, being no longer matters of difficulty, I have no hesitation in stating that the good work done by diaphragm classification for tube milling is quite undisputed, save under certain conditions. There is such a thing as overloading a single classifier for each tube mill, and I am of opinion that stage classification would do better work, utilising one classifier for the stamp mill pulp and the other for the tube outlet flow plus overflow from the primary classifier. My reasons are that if 50 stamps and three tube mills, where the stamps have a 15 ton duty through a nine mesh screen, are started with an empty tube mill circuit, there will be an all slime overflow for half an hour. During this time all the sand which comes goes back again to the tube mill until it is ground up. This goes on until the circuit is full, and not till then does the overflow grading begin to be normal. A diaphragm cone 9 ft. deep is larger than necessary, at the start up of the mill but two hours afterwards, there is the possibility of overloading such a classifier. If the tube mills get out of sorts, for their digestive organs do fail them sometimes, the same single cones have got to deal with these troubles. We have command over the tonnage of ore from the stamp mill, the proper method then is to classify the stamp mill pulp in one classifier and the tube mill pulp in a separate classifier.

Within certain limits the classification of a 9 mesh product is extremely simple. It is just like dropping stones into a bucket of water when they immediately sink; so a small classifier is able to classify the stamp mill pulp so as to give a clean underflow and a larger classifier may be used for the comminuted pulp. The Benoni Consolidated Mine is the first to set this example* where the pulp gravitates from the mortar boxes to the primary classifiers which are of the Stadler type, 4 ft. deep. By this method of primary and secondary classification less excavation work has to be done, and by periodically testing the underflow from the secondary classifiers an intelligent shiftsman can detect any symptoms of trouble in the crushing work of the tube mills. This method requires no greater height of pulp elevation than single unit classification. There is a slight disadvantdge that all the pulp has to be elevated, but that is offset by the advantages accruing from stage classification, which on the other hand again allows full amalgamable gold to be recovered as set forth in the accompanying diagram.

It will be seen that two elevations of pulp are only incidental to the situation of the amalgamated plates, there being the following alternatives: (1) The placing of the plates in excavated ground below the tube outlets. (2) The placing of the plates over the tube mills so that the pulp after being elevated will pass over the plates *en route* to the secondary classifiers. (3) The elevating of the pulp on to natural ground where the plates can be situated away from the crushing plant, the gold being amalgamated there prior to the pulp being pumped to the secondary classifiers. The engineers of the Benoni Consolidated mine have adopted the latter system. It will further be seen that if necessary the underflow from the secondary classifiers can be delivered direct to the tube inlets and thereby neutralise any excess of moisture in the underflow of the primary classifiers. According to the plan of that plant the grades of the launders are such that the pulp from any mortar box can gravitate through a distributor to any tube mill or a certain number of stamps and a tube mill can act as an independent unit for the reduction of the stamp mill product.

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* See Discussion, "The Tube Mill Circuit and Classification," this *Journal*, Vol. X., May, 1910, p. 400.

* See "Benoni Mill Flow Sheet," *South African Mining Journal*, March 11, 1911.

I have to thank Mr. E. J. Way, consulting engineer to the Anglo-French Exploration Co., Ltd., who on my application to him for permission to show the principles of the classification system adopted at the Benoni Consolidated mill was good enough to send me a blue print of the crushing plant from which I had prepared diagram Fig. I. Fig. II. represents the class of classifier adopted at that plant.

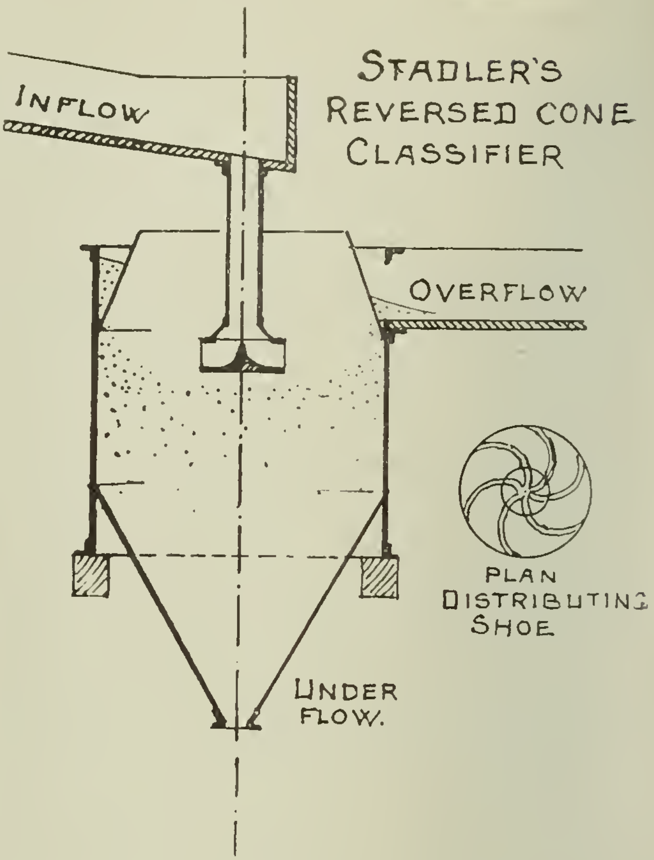

FIG. II.

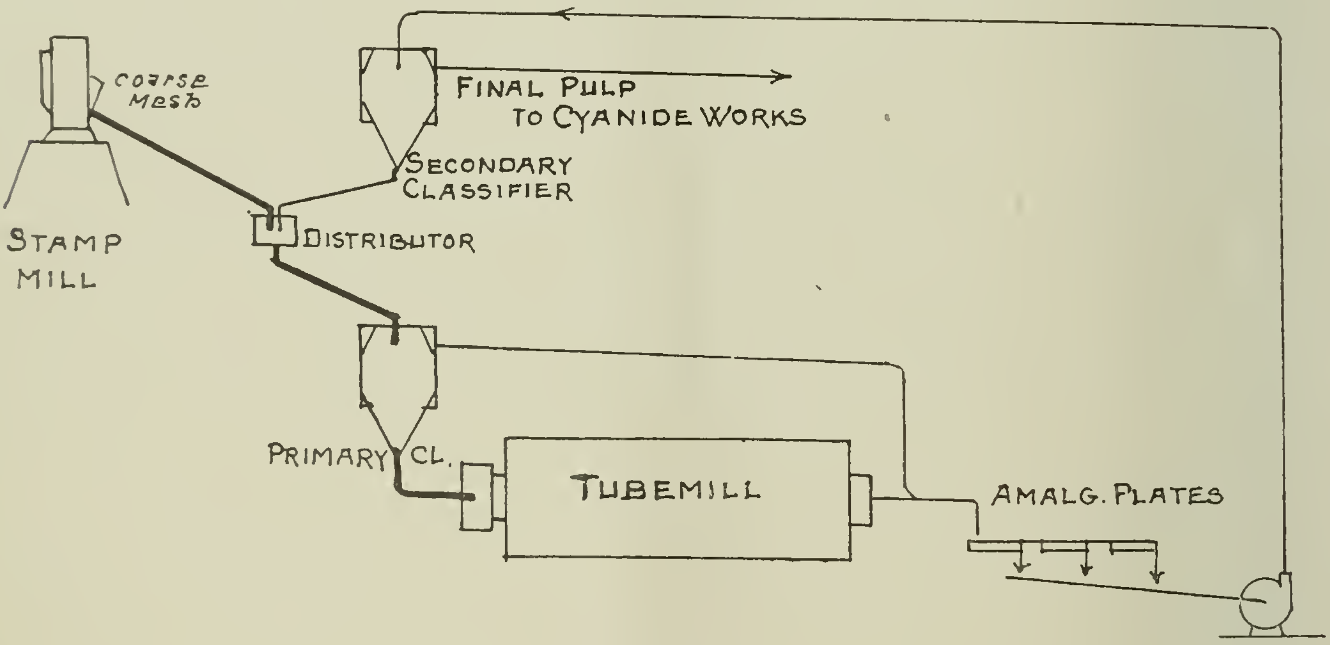

FIG. I.

Before we had tube-mills all the amalgamable gold, except a small quantity which remained in the stamp mill die sand was recovered through a 1,600 mesh screen when required. Now stones are run over plates in the stamp-mill or large tonnages of coarse sand over the tube mill plates. If only the fine gold is required in the cyanide works the underflow from the secondary classifiers contains the coarse gold. Assuming this sand is too coarse for amalgamation on plates surely it is not a difficult matter to apply a suitable sizer and get all the coarse gold in a few tons of fairly fine sand. If it is deleterious to progressive mill practice to run a 9 mesh product over a stamp mill plate, is the only alternative to remove the plate? Many different types of sizers are in successful use on base metal ores in different parts of the world including the simple Callow Screen in front of the mortar box. Has this screen been tried on the Rand to size off the coarse sand? Although the author's arguments against the recovery of all the free gold by amalgamation only had reference to existing designs it will stimulate discussion by my assuming as I have done that "E proposed arrangement" was the case in point. The author is to be congratulated for having brought forward such an interesting subject for discussion, and I think our mining members have a good plan of thrashing out a subject, a method which our metallurgical members really ought to copy. It gives those members a chance to reply who have contributed at length to a discussion on a paper of so great importance to the industry.

AIR LIFT AGITATION OF SLIME PULP.

(*Read at March Meeting, 1911.*)

By Robert Allen, M.A., B.Sc., M.I.M.M.
(Member of Council).

DISCUSSION.

Mr. C. B. Kingston (*Corresponding Member of Council*): Mr. Allen's paper on Air Lift Agitation is of special interest to me, because some months ago in consultation with Dr. W. A. Caldecott and the author I decided to instal Air Lift Agitation Vats to treat the ore from The Lonely Reef Gold Mining Company's property in Southern Rhodesia.

Experiment showed that a satisfactory extraction could be obtained in a single operation by agitating a mixture of fine sand and slime in this manner. Tests made with a small experimental vat to determine the time of agitation and density of pulp from which the most profitable extraction might be expected gave the following results:—

The experimental vat used was 48 in. x 12 in. with conical bottom, and central pipe 2 in. diameter.

Effect of Pulp Density on Air Lift Agitation.

Sample Assay	Pulp Density.		Solution after Agitation dwt. per ton.		Residue after Agitation dwt. per ton.	
dwt.	Solution	Dry Slime.	5 Hours	8 Hours	5 Hours	8 Hours
8·3	2	1	3·9	4·1	0·5	0·1
8·3	1·5	1	5·0	5·3	0·8	0·35
8·3	1	1	7·6	7·9	0·7	0·4

The denser pulps yield lower extractions, but concentrate the dissolved gold in a smaller volume of solution. The difference in the results obtained with pulps of 1·5 to 1, and 1 to 1 is slight. The saving in cost of plant, consumption of power and loss of water makes it profitable to work with a thick pulp. The Air Lift Vats have been designed to work with a pulp of 1·5 to 1, but it is probable that a pulp of 1 to 1 will be used in practice.

Most of the gold is dissolved in four hours. The vats installed provide for 18 hours agitation if necessary. The point of maximum economy will be determined in practice. These experimental residues indicate a good percentage extraction from the ore which goes to the mill at 18 dwts. to 22 dwts. per ton. Improved results will no doubt be obtained with experience. As the complete plant possesses some features that may be of interest to members, a description of it will be contributed later.

Effect of Time of Agitation.

Charge	Grading Analysis Percentages.				Residue after Agitation dwt. per ton.			
dwt.	+90	−90	+150	−150	Two hours	Four hours	Six hours	Eight hours
8·0	10	90			1·2	2·0	2·0	1·9
8·0		100			3·2	1·9	1·9	1·8
8·0		100			3·2	2·5	2·0	—
8·0		100			2·5	2·2	2·1	1·8
9·2	2·25		7·5	90·25	1·6	1·4	1·2	—
8·6	2·25		7·5	90·25	2·2	1·7	1·3	—
9·3	7·	93			3·3	—	2·3	2·3
12·8	1·		7·0	89·00	2·9	2·7	2·0	—
8·0	1·		7·0	89·00	0·9	1·0	1·0	—

FUTURE ECONOMIES IN RAND REDUCTION PLANTS.

(*Read at April Meeting, 1911.*)

By C. O. Schmitt (Member.)

DISCUSSION.

Mr. H. Stadler (*Associate*): Owing to the increase of mining costs with the increase of depth any suggestion which may result in a lowering of the working costs is of vital importance to the mining industry.

The subject of Mr. Schmitt's lengthy contribution is one, therefore, eminently suited for discussion by technical societies, and for this reason the thanks of all members interested in the mining industry are due to the author.

The paper itself, however, has been so hurriedly compiled, and is consequently so full of obvious and serious mistakes that, *per se*, it hardly merits serious consideration. As, however, a few people may be misled by the mass of words and figures used by the author, and as the discussion of even an inaccurate paper may throw some light upon the questions it deals with, I propose to subject the author's paper to a little unbiassed criticism.

1. *Lowering of Grades and Profits with Increasing Depth.* — In order to emphasise the importance of his suggested schemes for the reduction of costs, the author gives some data on the falling off of the grade of the ore in the deeper levels of the Rand. An article in the *S.A. Mining Journal* (April 29th, 1911), scruti-

nising the author's points, concludes with the remark : —"That they are based upon entirely misapplied premises, and are, consequently, incorrect and misleading to the highest degree."

2. *Savings Resulting from Redistribution and Closer Classification of Ore in Breaking and Sorting Plant.* (*a*) *General.*—In the comparative statements of operating costs (Table IV. and IVa.) it unfortunately happens that the author (in the item "Sorting"), multiplied the unit price given per ton of waste *sorted* by the total tons sent to the sorting plant. This serious error makes the sorting costs, in many cases, as high as either the stamp milling or the tube milling costs, and affects the figures throughout the paper. The consequences I have not troubled to calculate, because in another important particular also, the tonnages are entirely wrong.

The suggested improvements in the crushing and sorting plant are based on the principle of relieving the preliminary crushing units from the "fines" in the feed by deviating them direct to the secondary crushing appliances. As shown, for instance, in diagram Fig. 6, only the + 2 in. rock of the ore hoisted is run through the breaker after sorting, whilst the − 2 in. portion is deviated to a second screening and sorting plant, whence the $+\frac{3}{8}$ in. grade is sent to the stamp mill and the $-\frac{3}{8}$ in. grade (together with the $-\frac{3}{8}$ in. portion deviated from the ore breaker discharge) to the tube mills. By such elimination of "fines" the tonnages fed to the preliminary crushing appliances are brought down, but since the feed becomes much coarser more energy is required for reduction ; in other words, the gain in the tonnage is practically wiped out by a corresponding loss in the duty. The presence of "fines" in the feed has no detrimental effect on the efficiency of an ore-breaker of sufficient capacity, if the volume of the "fines" is not far in excess of the volume of the interstices in the coarse rock. The same reasoning is applicable also to stamp crushing. This can be best demonstrated by an assumed example in which not only the tonnage crushed but also the work done is taken into account by representing the gradings by their mechanical value (E.U.)* Assuming that a mixed feed of 70 tons (to a 5 stamp mortar box with 4 mesh screen) contains, as per Fig. 6, 16 tons $+\frac{3}{8}$ in. rock of a mechanical value 0 (equal to equivalents of 1 in. cubes) and 24 tons $-\frac{3}{8}$ in. "fines" with a mechanical value of 16 E.U., then the mechanical value of the mixed feed works out at 5·5 E.U. The work done by the stamps is therefore in the two cases :—

1. *Mixed Feed*—

Mech. value of 4 mesh discharge	17·0 E.U.
Less mech. value of feed	5·5 E.U.
	11·5 E.U. × 70 tons
	= 805 E.U.

2. "*Fines*" *Discarded*—

Mech. value of 4 mesh discharge	17·0 E.U.
Less mech. value of feed	0·0 E.U.
	17·0 E.U. × 46 tons
	= 782 E.U.

This shows that the work done is about the same in both cases, and that no appreciable saving in working costs and capital expenditure can therefore be obtained by the deviation of the "fines" and the consequent tonnage reduction. The battery screen being of ample capacity the "fines" readily find their way through it and if they are caught—well, they are crushed ; exactly what is wanted. The discharge pulp becomes perhaps a little finer than the discharge through the same mesh in the case of a clean coarse feed. A reasonable percentage of "fines" present in the battery feed does not affect the size-composition of the mortar box content, which is exclusively determined by the battery mesh. The working conditions of the stamps are therefore exactly the same whether the "fines" are eliminated from the feed or not. Experiments made at the New Kleinfontein mine confirm these facts and clearly prove that the tonnage crushed of the separately measured coarse rock portion of the battery feed is not lessened by an addition of "fines," on the contrary, the addition of "fines" within well defined limits apparently improves the efficiency of the stamp. The stamps will therefore hardly derive advantage from the deviation of the $-\frac{3}{8}$ in. portion to the tube mills but, on the other hand, the tube mills will doubtless benefit from this addition owing to the consequent increase in coarseness of their feed. However, an equally efficient coarse feed, showing a similar grading to this combined feed, can just as well be obtained by using a suitably coarser battery screen. Since a 4 mesh screen does not seem, so far, to mark the limit of highest efficiency, there is a possibility of success in the new Giesecke mill (a combination of ball and tube mill revolving on the same trunnions) now under examination by the Mines Trials Committee. This mill takes its feed direct from the ore breakers.

(*b*) *Saving in Working Costs.*—Inaccurate as they are, the figures in Table IV., which compare the operating costs in an existing plant (16% waste discarded) with the operating costs in a proposed new plant (30% waste discarded), show practically no other advantage in favour of the suggested modifications than that resulting from the discarding of a larger percentage of waste. Sorting being cheaper than milling, it is quite evident that the working costs, if distributed over the total ore hoisted, and therefore over the waste discarded as well as over the ore milled must fall approximately in proportion to the percentage of the ore sorted.

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* See this *Journal*, May, 1910.

(*c*) *Saving in Capital Expenditure.*—Here again the reduction in capital expenditure per ton hoisted is the natural consequence of the larger amount of waste sorted and therefore not the result of the suggested improvements. These improvements requiring, as they do, more complicated arrangements than does the present practice—bring the cost up again.

By debiting interest, depreciation and redemption of capital expenditure (at 12% per annum) to working costs, the author obtained for his improved scheme, with 30% waste discarded (as per diagram Fig. 6, picked out at random), the following savings compared with an existing plant with 16% waste discarded (diagram Fig. 2):—

Saving in operating costs per ton hoisted	10·32d.
Saving in capital charges per ton hoisted	1·47d.
Grand total per ton hoisted...	11·79d. = say 1s.

It appears from the above figures that the possible saving resulting from capital charges is insignificant in comparison with the influence of other more important factors affecting the working costs, such as, for instance, amount of waste discarded, etc.

(*a*) *Economical Limit of Sorting.*—We do not require lengthy papers to tell us that working costs per ton hoisted can be reduced by an increase in the percentage of waste discarded. What is really of importance is to determine, by taking into account the operating costs of sorting and the loss of the unrecovered gold in the waste, what are actually the economic limits of sorting.

In compiling the following table for the purpose of comparing the reduction costs in two cases, namely, no waste sorted and 30% waste sorted, I have taken over part of the author's figures as they stand, and any inaccuracy that may be present will not affect the purpose of this comparison.

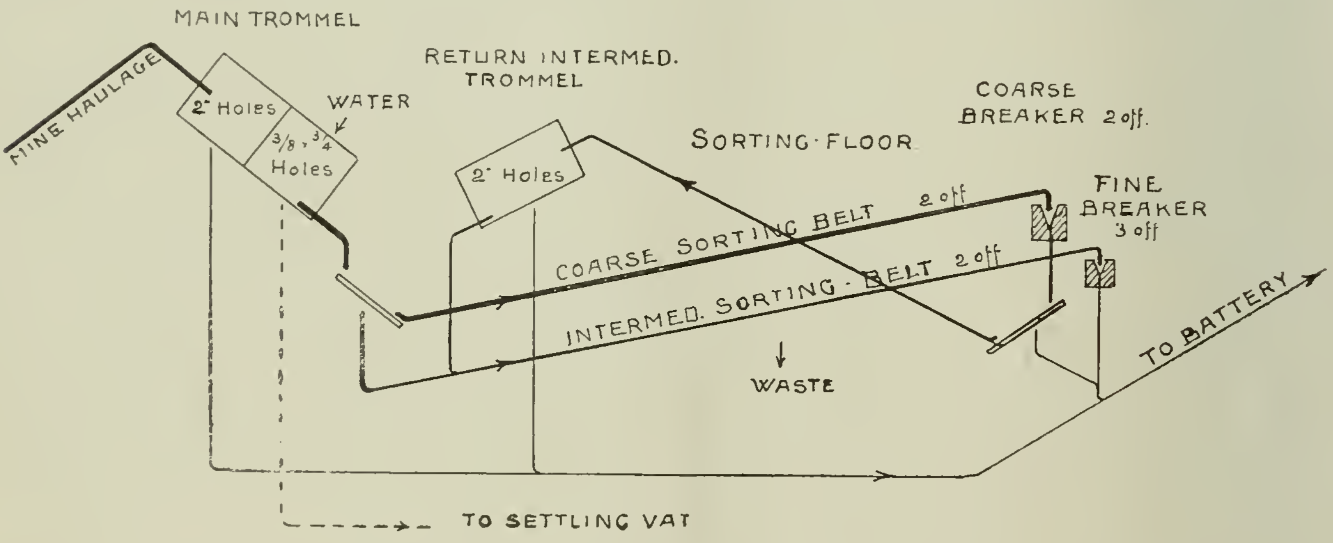
From these figures it appears that with 30% waste (0·8 dwt.) sorted, the economical limit is already passed. In practice the differences might even be greater as the value of the unrecovered gold in the waste is almost sure to increase with the increase in the percentage of waste sorted and as the operating costs with a more complicated sorting plant will also in all probability be higher than those given in the table. On the other hand, if no waste is sorted, the costs will—especially in the case of stamps hung up in consequence of shortage of labour—probably be lower, owing to a greater tonnage being available for crushing. This proves how easily sound economy can be overlooked by those in a hurry for lightning "progress."

As the author claims to introduce improvements on present day practice, he will appreciate my putting a skeleton sketch of the breaking

	Ratio per Ton.	Reduction Costs per Ton Hoisted. New Proposal Fig. 6.			
		No Waste Discarded		30% Waste Discarded.	
	Shillings	Tons.	Shillings.	Tons.	Shillings.
Milling	1·5	70	105·00	46	69·00
Tube milling	·75	82·5	61·88	58·5	43·75
Grinding	1·75	100·	175·00	70·	122·50
Screening	·0416	100·	4·16	180·	7·49
Sorting	1·0	—	—	30·	30·00
Coarse breaking	·167	—	—	—	—
Fine breaking	·334	50·	16·70	30·	10·00
Reduction costs per 100 tons hoisted ...			362·74		282·74
Reduction costs per ton hoisted			3·63		2·83
Adding loss of unrecovered gold 0·8 dwt. (3·33s.) per ton sorted				30	1·00
Total per ton hoisted			3·63		3·83

and sorting plant of the New Kleinfontein mine—so frequently referred to in his paper—before him. This plant has now been working for seven years and its perfect arrangements will hardly be beaten by any of the author's "new proposals."

3. *Savings Resulting from Improvements in Individual Crushing Units.*—At the end of the year 1908, coarse crushing tests on a larger scale were executed on the initiative of Dr. Caldecott by The Mines Trials Committee at the Simmer East Mine under the direction of Mr. W. R. Dowling. In a report, dated 19th January, 1910, dealing with the results of these tests I clearly laid down and showed graphically, that single stage crushing appears to reach the limit of economy at about 1,200 to 1,400 mesh, corresponding to a final pulp of about 24·5 E.U. (say 17% of the + 60 grade). For the production of fine final pulps stage crushing proves to be more efficient, provided that the additional expenditure is paid for by the higher extraction to be expected. The economical advantages of recrushing by tube-mills are only pronounced when the very coarsest battery screens are used, whilst the intermediate stages all show—if any—small advantages, or, putting this statement in the words of one of my conclusions: "Not only the mechanical, but also the economical efficiency increases with the coarseness of the battery screen, and, *even a 4 mesh screen (·404 in.) appears not yet to mark the limit of the highest efficiency.*" Dr. Caldecott, who had pioneered the coarse crushing policy, guided by practical considerations stopped at about 9 mesh, whilst Messrs. Johnson and Way went still further in an extreme coarse-crushing policy.



Breaking and Sorting Plant, New Kleinfontein Co., Ltd.

For the calculation of the "savings" to be realised by improvements in the mill plant as an independent crushing unit, the author falls back upon the figures given by Mr. Johnson on the results of the experimental tests carried out at the Cason and Angelo Mines on the East Rand*, but he completely ignores my strong comment upon these figures in my reply† to that discussion.

Convincing arguments have already been brought forward‡ against the system of creating a "Nominal Crushing Unit," the establishment of an arbitrary ratio between the power consumption in tube-mills to these N.C.U., and the fallacy of using the production of the – 90 grade as a standard for efficiency measurement, but as Mr. Schmitt does not refute these plain statements, I stigmatise his figures, so far as deduced therefrom, as misleading, incorrect and worthless.

4. *Savings Resulting from Higher Extraction by Finer Grinding.*—The author reminds members of the following paragraph contained in my reply to the discussion on my Paper on Grading Analyses§ :—

"By the statement made by Mr. E. H. Johnson in his contribution that the stamp and tube-mill combination produced 9·5% more surfaces than the stamps alone, the impression is—perhaps unintentionally—produced that the *useful work* had been increased in this proportion. The assay value of the ore being given by weight is, however, in both cases the same, and the only gain to be derived from the additional energy spent for the further reduction of the pulp, is to be looked for in the direction of higher extraction, which *in the present case, under fair assumption, amounts to about* 1%."

The pulps then under consideration were pulps Nos. I. and II. (not I. and III.) between which, according to the author's own statement, the gain in extraction is only 1·56%. In reducing the above 9·5% to "about 1%," I was therefore not very wide of the mark, or, as I have shown to be the fact, this is about the only one of the author's figures which happens to be approximately correct.

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* See this *Journal*, July, 1910.
† See this *Journal*, December, 1910.
‡ See this *Journal*, May, 1910.
§ See this *Journal*, December, 1910.

The data published by the author, on the gain in extraction with finer grinding, are quite insufficient to allow the establishing of a regular ratio of increase, and—if anything—they only prove that this question cannot be generalised. One of the five Mines quoted, shows, for the coarsest pulp No. III. (17% + 60 grade), an extraction of 94·8%, *i.e.*, very nearly as high as obtained at each of the other four mines with the finest pulp No. I. (2·5% + 60 grade). For this example, the gain in extraction between pulps I. and II. is only 0·4% and therefore even less than "*about* 1%."

5. *Actual Experiences.*—The results of a longer run on a practical working scale at the East Rand Proprietary Mines are awaited with great interest, as they will show whether the author's predictions will be fulfilled, or whether the advantages realised will remain within the limits of my calculations, which, though they are based on only elementary mathematical and physical laws, are, on the author's own confession, "mysterious formulae" to him.

In face of the fact that the New Kleinfontein Mine, run on conservative lines with single stage crushing, without tube-mills according to the Annual Reports for 1910, still holds the record in reduction costs (4s. 6·7d. with 94·50% extraction on 466,882 tons per year), against two of the largest progressive mines (East Rand Proprietary Mines: 5s. 5·8d. with 92·85% extraction on 2,126,324 tons and Crown Mines 5s. 9·0d. with 96·2% extraction on 1,511,658 tons), I cannot help feeling that metallurgical business on the Rand has become too refined and that, from the introduction of the Cyanide process to the present date, very little—if any—real economical progress has been made. A year ago* Mr. Way published a tabular statement showing that the reduction costs of the New Kleinfontein Mine are lower than on any one of about 30 Rand Mines, including some crushing over 100,000 tons per month. This statement has been ignored and it is high time that attention were compelled to the reason for these low costs. Although a fresh flood of figures may again swamp this question, it will never be solved until it is taken up by either the heads of the mining groups, the company directors or shareholders.

A point which might be raised in this connection is that the peculiar nature of the Kleinfontein ore allows a high extraction without recourse to very fine grinding. This might be true; but it is much more likely that this high extraction is exclusively due to the fact that a pulp passed through a 1,200 mesh battery screen is much more amenable to amalgamation than pulps coming from coarser meshes or discharge pulps from tube mills containg a large amount of slime. It is, therefore, very probable that, if the ore of the East Rand Proprietary Mine, which actually gives 50% extraction by amalgamation, were crushed through 1,200 mesh, it would yield the same high amalgamation extraction of 63% as the ore of the New Kleinfontein Mine. Should this prove to be the case, the extraction from the East Rand Proprietary Mine's ore would work out as follows:—

		Total Extraction
Assay value of mill feed ...	7·01 dwt.	
63% recovered by amalgamation	4·42 dwt.	4·42 dwt
Final pulp	2·59 dwt.	
Extraction by cyanide (say, 85%)	2·20 dwt.	2·20 dwt
		6·62 dwt
		or 94·1% of mill feed

as against 92·85% actually obtained with very fine grinding.

CONCLUSION.

The final results of the possible savings worked out by the author are briefly as follows:—

1. *Savings resulting from suggested improvements in breaking and sorting plant compared with present day practice.*—

Operating costs	1/- per ton milled
Gain in higher extraction by finer grinding... ...	6·72d ,,
Grand total ...	1/6·72d ,,

2. *Savings due to crushing by a double stage plant, as compared with single stage by stamps only.*—

Operating costs	8·2d ,,
Gain in higher extraction by finer grinding... ...	1/-
Grand total ...	1/8·2d ,,

Although the savings to be realised by the suggested improvements of the breaker and sorting plant, which, as the author bashfully admits, "may or may not be adopted in practice," cannot merely be added to those resulting from double stage crushing, "either result (the author states) can be improved by a judicious combination of the two." The very considerable increase in the total profits to be obtained in either case, reminds one involuntarily of the little story of the man who went to buy a stove. The merchant (in this case Mr. Schmitt), as the agent of a well-

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*See this *Journal*, April 1910.

known manufacturing firm, recommended him a new pattern, by the use of which 50% of the coal could be saved, "Well," said the would be buyer, "give me two of them and I'll save the lot."

Whilst I am at perfect liberty to criticise and go as far as I take responsibility, I might be reproached for the form and wording of my criticism, as not carefully balanced, and as unfair from a professional point of view. My answer to this is that the attack has not been made in the dark but in the full light of the world-wide scientific reputation of this Society, in whose *Journal*—if caught blundering—I, too, may one day be roasted.

Mr. Tom Johnson (*Member of Council*): The author is to be congratulated on that part of his paper dealing with the surface plants. There is no doubt that many of us will often refer to it with benefit to ourselves, and, I hope, to the industry in which most of us are engaged. In a paper of the length of the one under discussion, there are sure to be some points where others hold different opinions to those of the author. There is no doubt that the author points out a way of reducing costs, even if the reductions do not amount to as much as he calculates.

It is certain that on mines working thin reefs more sorting can be done than at present, even with single breaking; but with double breaking and more screening one hardly knows what percentage could be sorted out, although it might be better to break the stope over or under the reef first and pack, then take the reef as clean as possible after. If breaking, screening and sorting costs 1s. 6d. and total reduction 4s., it is seen at once that 2s. 6d. per ton would be saved on every ton sorted out. The cost of sorting a ton of waste is greater however when sorting from − 2″ than from + 2″ ore. There are one or two things to keep in mind. Sorting should not be so close that the value of the waste rock increases too much; with 30% sorting and the gold in the waste rock 0·2 dwt. higher, about 5d. per ton milled would be lost. Another thing to examine, is whether a certain labour force is sufficient to handle a greater tonnage than is being delivered then, for this difference, the saving by sorting closer is not 2s. 6d. per ton, as the wage bill is already paid in the reduction works.

In the latter part of his paper the author mentions the large amount of inertia in custom and thought on the Rand which is opposed to change. This is true, but we must acknowledge that in surface work there are changes taking place, also there are some indications that changes will take place in underground work, and not before they are needed. For these reasons I absolutely differ from the author in regard to his statement that costs will increase in depth for the underground work.

I will now examine the items of cost in the order set forth by the author on page 466 of the *Journal*.

(*a.*) *Increased cost of skilled and unskilled labour.*—Both skilled and unskilled labour will cost less per ton, even with increased wages due to better methods of working, which the mines will have to adopt. They will be forced to make a virtue of necessity. I have several times pointed out, in papers before this Society, some of the changes that will have to be made, not because I say so, but because they are necessary. As to scarcity of native labour for drilling, it is being found that machine work can be done cheaper than hammer work and can be done in narrower stopes. It will also be found that a large amount of native labour will be replaced by white, in what is termed "native work" and at no greater cost. Of course this is heresy, still it will come. It is only the idleness of the mine officials that prevents it. Look at the benefit to both masters and men, if we would only do the right thing and spread our native force properly and work about 50% greater tonnage, either in the present mines or others to be started.

(*b.*) *Explosives.*—"More required due to increased hardness." Increased depth does not necessarily mean increased hardness. I have seen harder rock at 1,000 ft. than at 3,000 ft. Even granted that the rock is harder at 5,000 ft. than at 1,000 ft., this does not necessarily mean that it will take more explosives to break it. The probability is that it will take much less, owing to the very pressure the author assumes will cause an increase. The use of smaller machines will also contribute to the decrease in quantity, as will stoping without leaving pillars, and the employment of rock breaking experts. Explosives will certainly be less per ton with equal stoping widths than at the present time.

(*c.*) *Ventilation.*—"Outcrop and shallow deeps have not provided any artificial ventilation. Cost liable to be 2d. to 3d. per ton hoisted." Mr. Penlerick said in his paper recently read before this Society, that the cost on the East Rand Proprietary Mines was 1d. per ton, and that they had *got* back their capital on the plant three times over. The Village Deep and Village Main Reef bear this out, so it is probable that the deeper mines will have an advantage in having to provide artificial ventilation, for most of the shallower mines of the present are losing money because of the lack of artifical ventilation.

(*d.*) *Mine Drainage.*—"Greater depth, greater cost." This looks logical, but fortunately it is not usual to get much increase of water in depth. So as to keep down the costs, the water will be collected in most cases at lesser depths than at present, or power will be taken from the water and used for different purposes such as ventilation. Then the quantity does not increase in ratio to the claim area, so there will be a smaller quantity of water per ton of rock hoisted. It is assumed that the deeper shafts will command much greater tonnages than most of the shallower shafts, and much larger tonnages will be hoisted per shift, so that drainage costs will be spread over the larger tonnages in this way keeping the costs down.

(*e.*) *Hoisting to Surface.*—The author says the cost of hoisting will undoubtedly increase. I am inclined to believe there will be very little increase if any. Mr. Sawyer in his paper "Profit per Fathom," fixes the hoisting charges at 5d. to 6d per ton. As Mr. Sawyer is connected with one of our deepest mines this figure is most likely taken from that mine. One of the chief reasons for high costs in hoisting, is the large number of engines used to hoist the rock. There seems a tendency towards improvement in this matter. It is certain that labour costs should not be above say 30% of what they are at present. The power charges can be greatly reduced by running the engines up to something like their capacity. The larger units and the large tonnages, with balanced hoisting of the future, will tend to keep the cost from rising much. If Kimberley can do over 7,000 tons in less than 12 hours with one engine and a pair of skips, using a labour force for this engine and skips only—whilst on the Rand probably ten times the number of engines, skips and labour force is used—there is a chance of keeping down the cost here.

(*f.*)—*Underground Transport.*—The author inclines to the belief that because of the longer distances the rock will have to be transported, that the cost will increase. If the mines were condemned to use only Kaffir labour this would be true, but when it is realized that it is just as easy to handle the transport mechanically in a mine properly laid out, the cost will be decreased instead of increased, also a few more mines can be supplied with the native labour set free.

(*g and h.*) *Power and Capital Charges.* I think the author is mistaken in saying these charges will increase when compared on the tonnage basis. Take the matter of hoisting engines, this charge will not increase, although boilers will. More air will not be used in drilling the rock. Ventilating machinery will quickly return its cost, and so with the other items when calculated on the ton basis. If a shaft be put down twice the depth of another and twice the tonnage be hoisted in a shorter time the cost per ton will decrease for the shaft.

Personally I think costs will decrease: if the inertia the author speaks of can be overcome on the surface, we underground men must put our backs into it and see that it is overcome underground, even if some of us get somewhat crushed with the weight of it. The whole fact of the matter is that underground work is too little studied. We are only just now beginning to understand that mining consists of something more than getting holes drilled, filling them with explosives and letting them go. The underground work is just as worthy of scientific study as the surface work. The author truly states that two-thirds of the costs are in connection with underground work, and yet it is carried on by rule of thumb. A good miner must, of necessity, have a fair knowledge of mechanics: one cannot get on without it, but with most underground men this knowledge is x in the equation. Although costs have come down greatly in the underground work, there is still room for further reductions without touching wages. I think the author did quite wrong to publish a table like his Table I. He is liable to do harm to the mines for the sake of emphasizing this contention that working costs must be reduced on the surface. I do not think it was necessary to go so far: his case is good enough without this emphasis. Where does the author get his figures from to cause him to put the recovery at 20s. per ton for mines of 5,000 ft. depth. Take the mines of over 4,000 ft. deep. The Cinderella Deep is recovering over 20s., the City Deep the same, the Turf mine is opening up better ground than the Village Deep which is over 20s. and the South Rand, I believe, is over 20s. Taken on the whole I do not see much difference between 2,000 ft. mines and those of 4,000 ft., there are good and bad patches in any horizon.

In the years that are gone mining was more or less selective, costs were high and profits had to be made to get a standing for the industry and to find money to open up new mines. Now-a-days thousands of tons are being mined that were left years ago. We are now getting down to the sensible idea that it pays to mine anything that will return a profit. The rich rock is not lost by so doing, but many tons are obtained that could not be got otherwise. Even at the present time I know mines where it would not reduce monthly profits if the value of the rock milled was allowed to drop 2s. per ton; working costs would come down and a greater number of payable tons would be got, also a greater total profit. There are many people who think something is wrong on the Rand because the profits are not increasing so fast as the increase of tonnage. Granted the profit per ton is lower, the ultimate profit will be greater. The larger tonnage available through the policy of milling all rock that will show a profit, multiplied by its profit per ton will be much greater than the tonnage of the older system multiplied by its profit per ton.

I do not know how the author reconciles his Table I with what he says after quoting figures from Ross E. Browne's paper. He says "At this period (Sept. 1905) the mines were working at considerably less depth than at present, yet the profit, if we ignore the capital charges of 5s. was not much greater than to-day as appears below," then he shows that the profit has dropped 0·18s. and the yield 7·7s. Outsiders, as well as ourselves, should take very particular notice of this, the yield has dropped 7·7s. whilst the profits have only dropped 0·18s. Is it not plain that the yield has not dropped because the rock got poorer, but has been allowed to drop through the inclusion of poorer rock that used to be left in times gone by. The yield per ton has been allowed to drop 21%, whilst the profit per ton has only been allowed to drop 3%, after allowing for redemption of capital. If we have dropped working costs 7s. per ton since 1905 by working at a considerably greater depth, by what reasoning does the author assume that costs increase with depth? Also, because the yield per ton has been allowed to drop, why does the author assume that it will continue to drop?

The author has laid an obligation on us by showing us how the surface costs can be reduced, but he is in error in offering this saving to the people underground. Instead of providing a margin for the underground people, they should be made to provide one for themselves, which should not be a heavy task considering the possibilities.

NATIVE FOOD SUPPLIES AND THEIR QUALITY.

(Read at January Meeting, 1911.)

By F. W. Watson, B.Sc. (Lond.), F.I.C., F.C.S. (Member).

REPLY TO DISCUSSION.

Mr. F. W. Watson (*Member*): In writing this paper I had expected that more data would have been brought forward during the discussion than has been the case, and I had hoped that my "analytical brethren"—as Mr. Gray calls them—would have contributed a larger quota.

Dr. McCrae, in the opening paragraph of his valuable contribution, says that had I been more conversant with actual conditions I would have modifies my opinion that little had been done to test the foods supplied to the compounds, and he also remarks on the work done by the Native Affairs Department. I have no desire to underrate the valuable work done by this Department, but it is a curious fact that in this Society, in which the scope for papers is so wide no one has, up to the present, brought forward a paper giving any data of work done on the foods supplied to the native labourers on the mines. It seems to me that it is a matter of great regret that the staff of the Native Affairs Department, have in this matter, been so prone to hide their light under a bushel at any rate as far as our Society is concerned, as it is in this *Journal* that we look for much information on the progress of chemical investigation on mine supplies.

In criticising my remarks on mealie meal, Dr. McCrae says that my experience of this article, as supplied to the compounds, cannot have been extensive as I had found no cause for complaint. This, I think, is another instance of the two great principles of "cause and effect." In my paper I stated the effect, *i.e.* that the samples of mealie meal submitted were of good quality. Dr. McCrae has attributed the cause to my limited experience, but I would suggest another reason and that is that the suppliers know that each consignment sent to the Consolidated Goldfields' compounds may be submitted for analysis and therefore they are careful to keep their article up to the standard. This is due, to a great extent, to the interest taken by our medical staff and the compound managers of the group (and it is only by co-operation between the various departments that efficiency can be obtained). While feeling a certain amount of sympathy for the native in other compounds who has to put up with weevil-eaten mealie meal, I am pleased that my critic has brought this matter up, since the obvious moral to other mining authorities is to go and do likewise in the matter of testing supplies.

In referring to the section of my paper on peanuts, Dr. McCrae quotes an analysis of the ground nut cake, and remarks on its high nutritive value as a foodstuff. This high nutritive value no one will deny, but to come down to the commercial aspect which is that usually taken up by the seller, I would take this opportunity of showing the difference in value between the oil and the cake after pressing. In the wet grinding of maize for the manufacture of starch, corn-flour, etc.– of which I can speak from actual experience—it is customary to separate the embryo or germ as it is commonly called—from the endosperm and these germs are ground in roller mills and pressed. The oil after refining used to fetch about 27s. 6d. per long cwt. (112 lbs.) the pressed cake 5s. 3d. per long cwt. This latter price was considerably below some of the commoner grades of starch, so that if starch is a cheap adulterant to pea-nuts much more so is pressed cake. It is for this reason that I would recommend that nuts should be tested chiefly for fat content and a certain percentage of that constituent specified.

In the Kafir beer section I am glad to note that Mr. Gray has, like myself, been more fortunate than Dr. McCrae. The samples submitted to the Goldfields Laboratory are sent by the compound managers who have control of the issue of this beverage and they are tested for alcoholic content on receipt at the laboratory, as this is the condition in which the beer is issued to the native. In the sample tested which did not go beyond 3·84% of alcohol, I am of opinion that the reason there was no further increase was the low diastasic power of the Kafir corn malt and therefore saccharification of the starch did not take place as it might otherwise have done with more efficient malt. In one instance the percentage of alcohol did not rise above 1·14 and this was entirely due to the starch not being converted to sugar. The quality of some of the malt supplied to the compounds leaves much to be desired. While on the subject of Kafir beer I would like to mention another alcoholic beverage, namely Kali. This is a favourite of the East Coast native, many of whom do not care for Kafir beer. Kali is made by taking the root of a plant (I believe the native name is Ezema), cutting it up and boiling it in water. The water and root are kept warm for three days and the water is then poured off. If this water is not discarded the liquor is too pungent and I believe has injurious if not poisonous effects. Golden syrup and sugar are then dissolved in warm water (not boiling) and the steeped root added, when a violent fermentation quickly sets up. As an experiment some of this beverage was made and was allowed to stand in an open tin for 24 hours. The percentage of alcohol by weight was then found to be 3·27. The whole sample was then well mixed and portions bottled in (a) screw stoppered bottles and (b) in open bottles. The alcoholic content was tested at various times with the following results:—

	In screw stoppered bottle.	In open bottle.
After 24 hours	4·02% alcohol	4·18% alcohol
,, 72 ,,	4·16 ,, ,,	4·98 ,, ,,
,, 96 ,,	4·51 ,, ,,	5·14 ,, ,,
,, 144 ,,	4·51 ,, ,,	5·14 ,, ,,

It will thus be seen that after 96 hours or 120 hours after brewing the increase in alcoholic content was practically negligible in both cases and the effect of pressure was to lower the percentage slightly. All the percentages given above are by weight. Sometimes the natives add sago or other grain, but this does not seem to have much effect on the alcohalic content of the beverage.

I must thank Dr. McCrae for his valuable contribution as he has raised many interesting points, but only regret that he has not opened his laboratory note-book a little more.

Mr. Pooler is to be congratulated on his well written criticism, but most of his remarks are on the physiological aspect of food which I felt, when writing this paper, was ground sacred to the medical profession. I can assure Mr. Pooler that several medical men have told me that the results I have given have been of value to them. Mr. Gray has shown that the figures quoted by Mr Pooler are misleading, and in regard to maize I have had experience of grains from all parts of the world, and 14% of proteid is an exceptionally high figure. Surely Mr. Pooler must have picked out the analysis giving the highest nitrogen value.

In quoting the figures on cocoa Mr. Pooler has evidently not read the text of the paper carefully, otherwise he would have seen it stated that samples A to D were of the variety known as soluble cocoa in which a certain percentage of fat had been removed, also he might have noticed the remark on sample E relative to the small portion of the cocoa fat having been extracted. If Mr. Pooler had noticed these statements, I am sure he would not have compared an analysis of a raw cocoa with that of the manufactured article.

I have to thank Mr. Gray for his remarks, figures and the interesting deductions he draws from the use of the Government Laboratory in Court cases.

I have, in addition to the discussion published in the *Journal*, two verbal communications to remark on. I was asked if I had ever noticed borax used as a preservative compound in supplies. I may say that I have not; but, even if I had, I would have been afraid to include it in my paper as it might have sounded a call to arms. I am sure, however, that the advocates of no borax would, in this instance, have a much larger following than when the discussion raged round the assay crucible. The second query was about the sugar supplied and the possibility of adulteration with sand. The samples of sugar submitted have been few and have been of fairly good quality. If, however, sand is used as an adulterant I would recommend the mines to specify that the particles of such sand shall be rounded and not angular, as otherwise we may be threatened with a new disease, namely silicosis of the alimentary canal. This might give an excellent field for medical research, display of lantern slides and perhaps the invention of a food classifier, but it would be rather rough on that valuable asset to the Rand gold mining industry—the native labourer.

THE ASSAY OF USED PLUMBAGO CRUCIBLES FOR GOLD.

(*Read at May Meeting*. 1911.)

By JOHN WATSON, F.I.C (Member).

DISCUSSION.

Mr. Alfred M. Thomas (*Associate*): I have carefully read through this paper, and am at a loss to understand why the author should make the assay so complicated, and tedious, as to require a combination of roasting, and crucible fusion. The oxidation of plumbago either by litharge, or nitre, presents no difficulties whatever, when carried out with the ordinary precautions. Hitherto, I have been accustomed to assay this class of material by the nitre method, taking ·2 assay ton for each charge, but by using a smaller quantity such as ·1 assay ton, just as reliable results can be obtained by the direct fusion method. Having a sample in the office almost identical in composition to the one operated on by the author, I got it ground to pass a 60-mesh sieve, and then made some assays of it, the results of which are tabulated below.

After well mixing the sample, I first of all ran through a preliminary assay, in order to find out the amount of silver required for parting purposes, and to be on the safe side, I put into each of the following assays three times the weight of silver to that of the gold found in this preliminary assay.

1. *Roasting, followed by fusion* (the author's method).—In order to prevent any risk of clotting, I mixed ·2 assay ton of powdered silica with the other contents of the roasting dish previous to placing it in the muffle, the fluxing, etc., was closely followed as directed in the author's paper.

Results :—

	(1)	(2) oz. per ton	(3)
Gold from the assay =	78·5	77·5	76·4
Gold from the slag =	·2	·3	·4
Total gold =	78·7	77·8	76·8

Average = 77·76 oz. gold per ton.

2. *Direct fusion method.*—

Used plumbago pot mixture ...	·1 assay ton.
Routine flux	2·5 assay tons.

After well mixing, the charge was put into a No. 1 link, or F crucible. As soon as the charge was completely fused a small wash composed of litharge and charcoal was added in order to clean the slag. On pouring, the slag was quite fluid, and the button of lead freely malleable. The slags were put back into their respective pots, and when fused, were given sufficient litharge wash to reduce a 10 to 12 gm. lead button. A small piece of silver was also added to act as a gold collector.

The routine flux is composed of the following by volume :—

Litharge	1 part.
Borax (20 mule-team brand) ...	1 part.
Soda (exsiccated)	5 parts.

Results :—

		oz. per ton.	
Gold from the assay =	76.4	78·5	79·0
Gold from the slag =	·1	·2	·2
Total gold =	76·5	78·7	79·2

Average = 78·13 oz. gold per ton.

3. *Nitre method.*—

Used plumbago pot mixture ...	·2 assay ton.
Routine flux	2·5 assay tons.
Powdered nitre	8 grams.

Well mixed, and put in a No. 3 link, or G crucible. In order to obtain good results by this method, it is necessary for the charge to be in a state of tranquil fusion before adding the litharge wash, and it is well to take the pot out of the furnace when seemingly fused, and rotate it to see if any effervescence occurs, if so, the pot must be returned for a longer time and more heat. The fusions soon became quite fluid, and gave lead buttons which broke away easily from the slags, and were very malleable. The slags were run as described above.

Results :—

		oz. per ton.	
Gold from the assay =	77·25	76·8	79·40
Gold from the slag =	·30	·3	·25
Total gold =	77·55	77·1	79·65

Average = 78·10 oz. gold per ton.

It is obvious in assaying this class of material either by direct fusion, or by the nitre method, quite a considerable saving in the aggregate of fluxes, crucibles, roasting dishes, and valuable time, would be effected when compared with the practice as described, and carried out by the author. I trust that the methods, and the results obtained therefrom, may be of interest to members and others connected with this Society.

Before concluding this contribution, I desire to express my thanks to Mr. A. W. Stockett, the general manager of the Simmer and Jack Proprietary Mines, for permitting me to publish the above records.

The meeting then closed.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

WHAT IS CYANAMIDE?—"The following, in view of some recent discussion relative to improvements in the cyanide process, is of interest to all metallurgists. It is abstracted from *Metallurgical and Chemical Engineering*:

At the present time there are three different substances that go under the name of 'cyanamide.' But only one of these has a legitimate right to the name. The substance with the right to the name is one whose chemical formula is $H_2N.CN$, and the name of cyanamide indicates to the chemist that an amide(NH_2) group is combined with a cyanogen (CN) group forming cyanamide. It is produced in the following manners:

Cyanogen chloride is introduced into a solution of ammonia in ether, and cyanamide is formed according to the following formula:

$$CNCl + 2H_3N = CN.NH_2 + NH_4Cl$$

If thio-urea is treated with the oxides of lead or mercury in an aqueous solution, cyanamide is formed, and the reaction can be written in the following manner:

$$NH_2.CS.NH_2 = CN.NH_2 + H_2S$$

Also if a cyanamide metal-derivative be treated with an acid under special conditions cyanamide will be produced. Pure cyanamide is a white crystalline, deliquescent mass, soluble in water, alcohol, and ether. It melts at 40° C. One characteristic feature of cyanamide is its tendency to polymerise, forming dicyandiamide and even higher products of polymerisation, such as melam. The polymerisation of cyanamide takes place when a solution of cyanamide is being concentrated or is kept for any length of time. If cyanamide be heated to 150° C. it polymerises with great violence. Chemically cyanamide can be either classed as weak acid or as a weak base. As an acid it will form salts with metals, such as silver, calcium and others. Of these salts the calcium salt is the most important. As a base, cyanamide will form a number of salts: with acids these salts as well as most of the metal salts are very unstable. Cyanamide as well as a number of its salts, is an unstable chemical complex that very readily transforms if conditions are right in two different directions, polymerising forming dicyandiamide or hydrolysing forming urea. The substance that is most generally known by the name of cyanamide is the product obtained in the process of fixing atmospheric nitrogen with calcium carbide. Cyanamide is the name most generally used for this compound, but a trade name of 'lime-nitrogen' for this product is being used. The only right lime-nitrogen has to the name of cyanamide is that one of the components of the product is a calcium salt of cyanamide.

The chemical formula for 'lime-nitrogen cyanamide' can only be expressed in the following manner:

$$xCaCN_2 + yCaO + zC + uCaC_2$$

From this formula it will be readily seen that this compound is a heterogeneous substance and of no definite chemical composition, nor do its components appear in the same ratio all the time. To explain the composition it is necessary to give, in brief, the chemistry of the formation. If calcium carbide is heated in an atmosphere of nitrogen, the carbide combines with the nitrogen with great avidity, according to the following formula:

$$CaC_2 + 2N = CaCN_2 + C$$

This reaction explains the existence of free carbon in the graphitic modification in the cyanamide, while the free CaO is an unavoidable admixture. The carbide used for the fixation of the nitrogen contains a considerable amount of free CaO as diluent, and this naturally will appear in the finished cyanamide. A small amount of carbide is unavoidable, as it is impossible with the present state of the art to totally bind all the carbide within reasonable time. The chemical nature of cyanamide or lime-nitrogen is a very complex question, and even more so than for the pure cyanamide $NH_2.CN$, on account of the unavoidable CaO. An Italian agricultural chemist by the name of Ulpiani has made some extensive researches on lime-nitrogen, and has found that lime-nitrogen is permanently changing its chemical structure, even only by storing same and under the influence of ordinary atmospheric conditions. What part the air-slacking of the lime may play in this chemical rearrangement of the chemical structure of lime-nitrogen, or, more correctly the calcium salt of cyanamide, is hard to determine. The calcium cyanamide is of an unstable structure and forms with great ease polymerised as well as hydrolysed compounds. In treating the lime-nitrogen with water, even with small amounts, still more complex phenomena take place, basic as well as acid calcium cyanamide salts are formed, with the free lime present; urea is formed; calcium cyanamide carbonate may be formed, and a quantity of others, including dicyandiamide and free ammonia. Ulpiani has made extensive research regarding the action of lime-nitrogen with various acids and in various solutions, and these are able to spread a good deal of light over an exceedingly complex chemical problem, which is growing more and more important as new uses for cyanamide are developed.

These researches of Ulpiani have also shed light on the fertilising value of cyanamide, and he proves conclusively that it is not cyanamide or dicyandiamide that is the direct fertilising ingredient in lime-nitrogen, but it is the result of hydrolysis, the urea, which is acted upon by the bacteria, and ultimately transformed into a nitrogen compound that can be absorbed by the plant.

The third product that claims the name of cyanamide bears it only as a trade name. Lime nitrogen is mixed with a certain percentage of sodium nitrate and with a low-grade nitrogeneous material as a diluent to regulate the nitrogen contents of the finished product, and is then called 'cyanamide' This material is compounded exclusively for the fertilizer trade and is of no chemical importance. It is apparent from the above that the name of cyanamide is one that has lost its original meaning and has come to indicate altogether different substances to different groups of people. The name 'cyanamide' should stand for the chemical compound known by the formula, H_2CN_2.

'Lime-nitrogen,' although a meaningless name, as meaningless as if gypsum were called 'lime-sulphur,' may remain the name for the product now generally known as 'cyanamide,' while for the third product a name like 'cyanamide compound' or 'lime-nitrogen compound' would probably answer, thus clearing up a question of names that is becoming more and more complex the wider the field stretches in which 'lime-nitrogen' can be used.—J. W. BECKMAN.—*Mining and Scientific Press*, Jan. 28, 1911, p. 184. (K. L. G.)

ESTIMATION OF TIN IN AN ORE.—(R. J. Morgan, Proc. Aus. Ins. Min. Eng., Jan., 1911.) The author states that the main object of his paper is to draw attention to the unsatisfactory. and by no means up-to-date, methods employed in estimating the tin contents of an ore. He then briefly reviews the dry methods. The wet methods, especially volumetric, are advocated. the most useful and accurate being, it is contended. the titration of stannous chloride with iodine. The method advocated is a modification of that generally known as the Pearce-Low, and appears to be a combination of the latter and that recommended by Thibault in 'Metallurgy of Tin.' Fusion is effected by means of caustic soda in an iron crucible, solution in dilute hydrochloric acid, reduction with metallic iron followed by metallic antimony, cooling the solution in an atmosphere of carbon dioxide and titration with iodine.

The author also reviews two other methods of reducing the tin ore, viz., reduction with zinc fume or clippings (J. J. Berringer, *Mining and Engineering Review*, Aug. 5, 1910) and reduction by hydrogen (Assay of Tin and Antimony, L. Parry). Methods of reducing stannic solutions are also reviewed. He considers reduction with antimony the best method (Ibbotson and Brearley. See Thibault's Metallurgy of Tin.) A table showing the comparative efficiency of antimony, iron and nickel as reducers is given. A short reference is also made to the method of titration with standard ferric chloride."—*Mining and Engineering Review*, March 6, 1911, p. 232. (A. R.)

CELTIUM.—"The discovery of a new element by Urbain in the gadolinite earths, where scandium and lutecium occur, was announced at the meeting of the Paris Academy of Sciences on the 16th January. A quantity of gadolinite had been worked up for the purpose of obtaining larger amounts of lutecium, and in the mother liquor resulting from several fractional crystallisations from nitric acid a new element was found to be present by spectroscopic examination. To this element it is proposed to assign the name celtium. Its spectrum exhibits a large number of bright lines."—*S. A. Journal of Scicnec*, April, 1911, p. 253. (H. A. W.)

NITON.—"In a paper by Dr. R. W. Gray and Sir William Ramsay, read before the Royal Society on January 12, the disintegration theory of Rutherford and Soddy received further confirmation by the announcement that, as a result of several determinations of its density, the atomic weight of the radium emanation—for which the authors propose the name of niton—was found to average 223. This agrees well with the view that radium (at. wt. 226·4), on changing into niton, loses one alpha particle (at. wt. 4). With regard to the name proposed by them, the authors point out the desirability of indicating, by similarity of name, that the gas belongs to the argon family. This is regarded as preferable to emphasising its radio-active relations. If the latter were to be indicated in the name, a similar principle would have to be applied to radium itself, which should then be named in some way that would indicate its derivation from uranium."—*S.A. Journal of Science*, April, 1911, p. 268. (H. A. W.)

HEAT OF COMBUSTION OF EXPLOSIVES.—"In order to obtain the true explosion temperature and heat combustion, it is necessary to know the composition of the explosion gases at the moment of completion of the explosion in the closed vessel. This differs considerably from that found on analysis, on account of the reactions taking place during the cooling. If A = CO_2, B = the hydrogen, C = the CO × three times the methane, D = the water less the methane, all expressed from the analysis as gram molecules per gram of explosive fired, then

$$\frac{(C+x)\ \ (D+x)}{(A-x)\ \ (B-x)}=K$$

For temperatures about 2,000°, K = 6·6 (Langen), or 5·5 (Mallard and Le Chatelier). The composition of the gases before cooling is thus obtained by deducting x from the gram molecules of CO_2 and H_2, determined analytically per gram of explosive, and the addition of x to those of CO and H_2.

The true heat of combustion is calculated from that determined experimentally by deducting 50,600 calories for each gram molecule of methane found, and 10,000 (Oswald) or 10,100 (Berthelot) calories for each gram molecule of CO_2 formed during cooling.

In order to obtain more accurate results, gravimetric methods are substituted for the usual gas analysis by absorption and measurement of volume. The sample is placed in the bomb usually 1 to 1·5 grams of dry powder in a bomb 10·1 cubic centimeters content, or 10 grams of another explosive in a bomb of 750 cubic centimeters. After sweeping out free air with dry CO_2 and closing the valves, the charge is fired, the bomb allowed to cool, and the gases—consisting of H_2O, CO_2, CO, CH_4, H_2 and N_2—passed slowly through a weighed $CaCl_2$ tube and potash bulbs. The mixture of CO, CH_4, H_2, and N_2 is then led over hot copper oxide in a hard glass tube 25 to 50 centimeters long, connected to a $CaCl_2$ tube and potash bulbs. The CH_4 and (by difference) the CO are determined in a fresh analysis by leading the gases issuing from the first potash bulbs into an absorption apparatus containing cuprous chloride solution, which is renewed several times during an analysis from a vessel at a higher level, through towers containing dilute sulphuric acid (to absorb ammonia), concentrated sulphuric acid (to keep back moisture), and thence to the combustion tube. The CO_2, weighed in the second potash bulbs, will now be due to the CH_4 alone. If the two firings are carried out under exactly similar conditions, the results of the two analyses will be directly comparable. The gases are fully swept out of the bombs, $CaCl_2$ tube, and potash bulbs by a stream of dry CO_2 free air, which, to avoid oxidation of the CO absorbed in the cuprous chloride, has been passed through alkaline pyrogallol, and the rubber connection leading to the cuprous chloride apparatus closed with a pinch cock. By raising the level of the cuprous chloride solution the gases are driven over into the sulphuric acid towers, and thence, with a stream of dry CO_2 free air, into the combustion tube and bulbs. By means of a pump with a drying tube interposed, the $CaCl_2$ tube and bomb, which is warmed to 50° C., are evacuated, and the water remaining in the bomb thus transferred. The nitrogen is determined as described in the preceding abstract."—O. POPPENBERG and E. STEPHAN.—*Mines and Minerals*, Feb., 1911. (A. R.)

DETECTION OF SMALL QUANTITIES OF GOLD AND SILVER BY MEANS OF FORMALDEHYDE.—"It is found that formaldehyde is far more sensitive than hydrochloric acid as a reagent for the precipitation of silver. Even at a dilution of 1 in 240,000, silver may be detected by formaldehyde, which produces a distinct violet colour, becoming more intense on warming or allowing time for its development in the cold. In more concentrated solutions the colour is red-violet, and at the comparatively high concentration of 1 in 10,000 violet-brown. Since copper is similarly reduced by formaldehyde, special precautions must be taken to remove it before making the test.

Formaldehyde may also be used to detect gold in solutions which give no visible reaction with oxalic acid, sulphur dioxide, or stannous chloride. In the cold 1 part of gold in 160,000 may be detected, and the sensitiveness of the test can be increased by warming the mixture.

In making the test a few drops of the indicator are added to the solution to be tested. The indicator is prepared by mixing two volumes of 30% formaldehyde with one volume of 20% potash."—G. ARMANI and J. BARBONI, *Chem. Zeit.*, Vol. 34, 1910, p. 142.—*Pacific Miner*, Feb., 1911, p. 61. (H. A. W.)

A NEW REACTION FOR COPPER.—"A new and extremely sharp reaction for the detection of copper has been discovered by observing the effect of a solution of 1, 2-diamidoanthraquinone-3-sulphonic acid, in dilute alkali with copper salts. In the presence of copper the reagent will cause the solution to take on an intense blue colour, which is distinguishable even when highly diluted. The reaction is especially sharp if one uses an alkaline solution of the diamidoanthraquinone sulphonic acid as a comparison.

The solution is prepared by dissolving 0·5 gm of 1, 2-diamidoanthraquinone sulphonic acid in 500 cc. of water with the addition of 40 cc. of concentrated sodium hydroxide solution of 40° B.

It is possible to detect copper with this indicator when everything else fails. The D solution is allowed to run into the solution to be tested and the blue colour forms instantly. The colour is noticeable with 0·0000019 gm. of copper in 1 cc. of solution."—Dr. RUDOLPH UHLENHUTH, *Chem. Zeit.*—*Pacific Miner*, Feb., 1911, p. 70. (H. A. W.)

METALLURGY.

PRECIPITATION OF GOLD AND SILVER IN CYANIDE SOLUTIONS BY CARBON. "(R. K. Cowes, Proc., Aus. Ins. Min. Eng., Jan., 1911): The paper deals with some difficulties, and their cause, that were encountered at the Waihi-Paeroa Gold Extraction Co.'s tailings plant at Waihi, New Zealand. The comparatively high values of the residues in gold gave a great deal of trouble. This was caused by the presence of carbon in the tailings. Experiments were conducted, and it was found that the silver residues, just after grinding, were normal, while the gold was slightly high. Samples were then taken every day and assayed, with the result that the value of gold residues unmistakably increased, while the value of silver residues decreased until such time when the gold was all practically precipitated; then the value of the silver residues started to rise, thus showing that the carbon had no apparent effect on the silver until nearly all the gold was precipitated. Some tabulated results of these experiments and others are given. It was found also that fresh carbon precipitated gold and silver more freely than when the carbon had become saturated with water."—*Mining and Engineering Review*, March 6, 1911, p. 232 (A. R.)

STANDARDISATION OF BULLION AND ASSAY VALUES.—"The following definitions have been adopted by the Institution:—

(1) Assay reports shall state the exact condition of the sample as to dryness when assayed.

(2) Assay values of gold and silver ore and products shall be represented in pennyweights and decimals or in ounces and decimals, and not in ounces, pennyweights and grains. They shall be expressed in terms of fine gold and fine silver respectively, not as 'bullion.'

(3) Assay values of alluvials shall be reported in grains and decimals of a grain of 'fine' gold, or in pence (at 2d. per grain of 'fine' gold), or cents per cubic yard.

It is recommended that, in the absence of specific information as to boulders, etc., one cubic yard of ordinary alluvial be taken as equivalent to 3,000 lb. (1½ short tons).

(4) In reporting assay values of cyanide and other solutions, the results shall be given in parts by weight in a stated volume of the solution.

In the case of cyanide solutions, the use of the 'fluid ton of 32 cub. ft.' is recommended. It closely approximates to 2,000 lb. and is in common use.

(5) When it is necessary to state or estimate the money value of an ore, etc. (other than of gold), it shall be accompanied by the assay value, and the basis on which the former has been calculated from the latter shall be stated.

(6) Laboratory sieving tests shall be made with the I.M.M. Standard sieves, or, when other sieves are used, the widths of the apertures shall be stated.

Note.—Reports of assay results, extraction tests, metallurgical and ore-dressing trials, etc., are sometimes so worded as to permit a wrong interpretation, and the Council urge the necessity for definite statements as to the nature and condition of the sample assayed, and, where advisable, an indication of the method of assay. Thus, in the case of such metals as copper, tin, lead, antimony, etc., where both 'wet' and 'dry' methods of assay are in use, results should be reported as obtained by 'fire' assay or by 'wet' assay, with further details as to method, if judged advisable. Similarly, it should be stated whether the sample was assayed in the condition in which it was received, or 'air-dried,' or dried at C.

In reporting that an assay for such a constituent as tin, arsenic, tungstic acid, etc., corresponds to so much 'black tin,' 'white arsenic,' 'wolfram' or other recognised saleable product, the report should give the assay result for such constituent, together with a definite statement as to the figures used in calculating such saleable product. Thus, an assay showing 1% of metallic tin might be reported as 1% of metallic tin, equivalent, if all could be extracted as such, to 1·5% of 'black tin' containing 66·6% (or two-thirds) of its weight of metallic tin. Similarly, in reporting results, where assay values might be mistaken as representing available produce, it should be stated that such are merely 'gross' assay values, a statement or estimate of the actual 'commercial' values being added whenever possible.

In such matters as ore-dressing, ore treating processes, furnace work, etc., where so much depends on the way in which results are reported, on the nature of the sample and on the method of taking it and its relation to other samples, the Council emphasise the necessity for careful phraseology and for the inclusion of exact data, as also the necessity for care in preparing, describing, labelling, sealing and otherwise ensuring the correctness and authenticity of samples submitted for tests or assays."—*Bulletin No. 76, Institution of Mining and Metallurgy*, Jan., 1911, p. 31. (H. A. W.)

AUTOMATIC PULP DISTRIBUTOR.—"The question of the regular and equal division of a main body of pulp for feeding nests of grinding pans, concentrators or tube mills has recently been discussed in a contemporary journal

Up to the present an all-round satisfactory machine for this purpose has not been described. In Broken Hill, New South Wales, automatic distributors are in use which accurately divide the main product into as many parts as required. As a case in point, a bucket elevator raises the pulp which gravitates to the distributor where it is equally divided and discharged into fifteen launders, which feed as many grinding pans.

A similar distributor is now being constructed for the Sons of Gwalia mine. The underflow from the sand separators will gravitate to the distributor, which will then equally feed ten grinding pans.

The feed hopper is set on ball bearings to minimise friction, a rotary motion being brought about by the reaction due to the centrifugal force of the pulp as it leaves the four discharge pipes. These discharge pipes distribute the pulp evenly over the bottom of the receiver. The receiver is divided by means of radial webs into as many compartments as there are grinding pans to feed. In the event of one grinding pan being temporarily out of commission, the hinged launder at the discharge from the receiver is deflected, so that the product falls into a pipe which delivers back to the tailings wheel. This increment to the amount passing over the sand separators will then be evenly divided over the nine remaining compartments and so equally feeding the nine grinding pans in commission.

This return scheme when one or more pans are idle, is essential to the successful working of the distributor, as, were the outlet from the compartment simply blocked, those compartments on either side would receive more than their fair share of the product.

A similar distributor is being made for equally feeding from a main launder, 13 secondary Wilfley tables at the Lake View and Star mill."—W. R. DEGENHARDT.—*Journal of the Chamber of Mines of Western Australia*, Jan. 31, 1911, p. 42. (H. A. W.)

ZINC DUST.—"*Recovery.*—The zinc vapours escaping during reduction process from the muffles are never completely condensed in the receiver to liquid zinc, but a considerable portion is caught in the form of metallic dust by special collectors connected as closely as possible with the receivers. In order to avoid the loss of any zinc vapours which are not precipitated in these collectors, the latter are further connected with collecting channels and similar appliances. The proportion of the zinc dust recovery to the spelter production depends on the zinc contents of the furnace charge, as well as on the temperature of the distilling furnaces. With a rich charge and a hot furnace, which means a high tension of the zinc vapours, the zinc dust recovery may reach from 10% to 15% of the spelter production. The best quality of zinc dust, so far as the percentage of the metal contents is concerned, is derived from the special collectors, because the stuff precipitated in them is least exposed to the oxidising influence of the air. This dust is composed of from 88% to 93% metallic Zn, and contains from 6% to 10% ZnO, as well as small quantities of Pb, Cd, As and Sb. The initial dust, a product which is formed three or four hours after the distilling process has been started, possesses a comparatively high percentage of oxide of zinc, but it also holds nearly the whole cadmium contents of the furnace charge. It is, therefore, used by preference for the production of the latter metal. Merchantable zinc dust must contain about 90% of metallic zinc.

Owing to the fact that the zinc dust precipitated in the special collectors is not throughout very finely divided, but contains a sprinkling of somewhat coarser particles, it is now customary to subject the stuff to an automatic sifting process in specially constructed zinc dust sifting apparatus, having about 1,024 meshes per square centimeter, or 6,700 meshes per square inch (sifting apparatus No. 80).

An important use has been found for zinc dust in the cyanide process of gold extraction. At the Homestake mine, one of the largest concerns in the world, very favourable results have been obtained. In the more old-fashioned works zinc shavings are used, but they are considerably dearer than dust. The material must contain at least 2% of lead; it is even desirable that it should contain a small quantity of nitrate of lead. The zinc dust is automatically introduced into the receptacles—large wooden vats—while the solution runs into the filter presses. The extraction of the gold contents and the retention of the precious metal in the filter presses are almost absolute. A notable feature of the process is the rapidity with which the zinc dust acts.

Analysis.—Among the many methods of analysing zinc dust, the Drewson method is one of the most convenient and most reliable. It is based upon the fact that finely divided zinc reduces a solution of chromic acid (bichromate of potassium and sulphuric acid) without generating hydrogen. The process is carried out as follows: Dilute 40 gm. of molten bichromate of potassium to make one litre, and adjust the titre with a strongly aciduated ferrous sulphate solution by means of the ferricyanide of potassium volumetric test. Put 1 gm. of zinc dust into a glass beaker and pour over it 100 cc. of the bichromate solution. Add, while gently shaking the beaker, 25 cc. of diluted sulphuric acid in portions of 5 cc. each, in intervals of from 8 to 10 minutes. Let solution stand for an hour, and then add another 25 cc. of diluted sulphuric acid. Then put the whole mixture into a volumetric apparatus and dilute to make about 500 cc. Then gradually add, while strongly stirring, ferrous sulphate solution (200 g. $FeSO_4 + 7H_2O$ + 25 cc. H_2SO_4 concentrated to 1,000 cc.) until one drop with ferricyanide of potassium solution produces a weak but distinct blue colouring.

In order to determine the quantity of the metallic zinc contents the cubic centimetres corresponding to the iron solution which has been used have to be deducted from the bichromate of potassium solution, and the bichromate of potassium contained in the remainder has to be multiplied by 0·66113."—PAUL SPEIER, *London Mining Journal.—Mining Science*, Jan. 12, 1911, p. 38. (H. A. W.)

AMERICAN PROGRESS IN CYANIDING.—"Each year sees a marked increase in the percentage of the world's precious metals recovered through the agency of cyanogen. In the early days of the cyanide process, the opinion was prevalent among metallurgists that the successful use of cyanide would be confined to a few special instances where all other means had proved to be futile and where the physical and chemical natures of the ore were particularly favourable. To-day practically the first questions asked when attacking the problem of treating gold-silver ore, are, what extraction can be attained by cyanide, and the cost of making it. It is only after these conditions are shown to be unfavourable that the possibilities of other processes are considered. As in the history of most of the other arts, the first years of cyaniding saw improvement follow improvement in such quick succession that there was great trouble in keeping pace with the march of progress, but as time went on, these improvements became less spectacular until to-day changes are more refinements of existing features than anything positively new. There are now plants that have attained a remarkably high degree of metallurgical perfection.

As has been the case for several years past, Mexico has led this year, though mostly through the efforts and ability of Anglo-Saxon engineers. Several remarkable new mills have been put into service there during the past twelve months, and several more are to follow in the first months of 1911. The flow-sheets of these new mills show that the 'Pachuca tank' will be used in almost every one. It is rather curious that although this was the invention of F. C. Brown, of New Zealand, it is used scarcely at all in Australia. A striking innovation has been made by the introduction of the continuous system of agitating which has proved an unqualified success. In the old method the pulp was agitated in separate vats which had no communication with each other, each constituting a self-contained unit. By the continuous system the 'Pachucas' are arranged in series or in batteries, and the pulp overflows from one into the next and passes through the whole series before its discharge to the filters. The method of communication used is a pipe set at an angle of 60°. The upper end of the pipe is placed in the discharging tank, half way between the central airlift and the side of the tank, while the lower end is placed in the same relative position in the receiving tank. The upper end of the pipe is distant about one-third of the height of the tank from the top, while the lower end is one-third from the bottom. A. Grothe states that both M. H. Kuryla and J. E. Mennell report increased extraction as a result of using this method, but this would seem probably to be due more to the fact that the old system of agitation was not carried to the point of maximum dissolution, than to any other reason. There is no reason to doubt, however, that there is a considerable saving of cyanide where the continuous process is used. Other advantages are, that the tank is used continuously for agitation, and no time is lost in filling and discharging. Also, the discharge being from the side, and not from the bottom, there is an economy of gravity pressure, which in most plants is important. Furthermore, the discharge being continuous, the pressure is constant; which is essential in many cases where the pulp is filtered in pressure-filters.

The use of the silica-sponge diaphragm has caused a good deal of discussion during the year, but so far has not proved of great value. The idea, as well as the application, is so new that further service may suggest changes that will bring it into more extended use. The reported disadvantage is that the volume of air which can be passed through the sponge is too small to prevent classification in the pulp, and that the interstitial spaces become choked easily and are hard to clean. Other operators report very successful results. C. A. Fulton says that in the San Matias mill at Guanajuato the process has been very successful and has reduced the time of agitation from 32 to 12 hours. It would seem that for successful use, the pulp agitated should be ground very fine and contain a very high percentage of flocculent. The Clancy process was announced late in the fall, but up to the present it has not been tried in enough places to give sure basis for judgment as to its practical success. At one well known property the experiments were brought to rather an abrupt termination at the instance of the management, as experimental work lasting over a considerable length of time proved that, so far as this particular property was concerned, there was little hope of its being used to advantage. Further application of the process will be watched closely, as the problem of treating ores containing combined gold is one of much interest. The wonderful increase in the use of the Dorr thickeners and classifiers shows that they have come to stay, for a time at any rate, in slime plants. Some new things are promised soon from South Africa along these lines in addition to the Caldecott diaphragm-cone, but a discussion of these will come better from the pen of some South African.

The various forms of filters, both vacuum and pressure, are still merrily warring for the favour of the profession. Among the former the Butters and the Oliver are going into many of the mills that are being designed by engineers favouring vacuum filters. The Butters Company is providing a filter of 1,000 ton capacity for the Dos Estrellas which will handle the slime from the No. 1 and No. 2 plants, and it is understood that the Real del Monte is to use the Butters in its enlarged mill. Another new Butters installation is at the Palmilla, at Parral, while the La Blanca and others have put in Moore filters.

Among the pressure filters the Merrill seems to be gaining great headway. The Santa Gertrudis engineers decided on this type for their new 600 ton mill after competitive tests (held at the mine on ore of the usual character), and in which a number of manufacturers of different types competed. The Merrill filters, it is also given out, are doing good work at Esperanza, where the management has recently given an order for one more, making six in all. The New York Honduras & Rosario Co. is also installing two of the large 90-leaf type in Honduras. At first when the Merrill press was used it was found that a certain amount of granular material was necessary in the slime. By a very ingenious idea, known as 'centre washing,' this objection has been largely overcome. The 'centre washing' system consists simply of filling the press to a point where there is left an opening through the centre of the cake about ½ in. wide. Solution or wash water is introduced into this opening through the nozzles on the automatic sluicing bar or through the sludge feed channel, and leaching begins. The flow of liquid commences at the centre and proceeds outward in both directions. The result is that instead of a solid 4 in. cake (which if very talcose, was formerly impermeable), there is now in the same leaf two cakes, each 1¾ in. thick, which are readily permeable, even when formed of the most talcose slime.

Among other filters, the Kelly seems to be achieving an increasing measure of popular favour. The great value of the Kelly lies in the small amount of water used in its operation, which is no small factor in many places where the water supply is limited and evaporation heavy. At the new mill of the Veta Colorada, at Parral, eight Kelly presses working in two batteries of four each are being installed, and two are in use in the Carmen mill in Guanajuato. The Tigre mill in Sonora is also to be equipped with Kelly filters. This last mill is rather remarkable among the Mexican plants from the fact that its stamps are to weigh 1,250 lb. It would rather seem that there is a tendency toward heavier stamps in Mexico as well as elsewhere. Certainly the trend is all that way farther north, although as yet designers in the United States have not dared to go as far in the matter as have engineers in South Africa, where some of the latest plants include stamps of 2,000 lb weight. A number of mills are being designed for work on gold ores containing stamps weighing 1,250 lb., and the probabilities are that, if these show good economic results, the fashionable stamp-mills of next year will include still heavier heads. Tube-milling is also gaining in favour steadily, as it has been shown that fine grinding results almost invariably in increased amalgamation as well as added extraction in the subsequent cyaniding. The distinctive feature, however, of the fine grinding of gold ore seems to be in a release of the gold in the sulphides, thereby shortening the necessary time of contact with cyanide solution. This makes possible in many cases the direct cyaniding of the sulphides in the same flow with the slime and thereby does away with losses due to concentration.

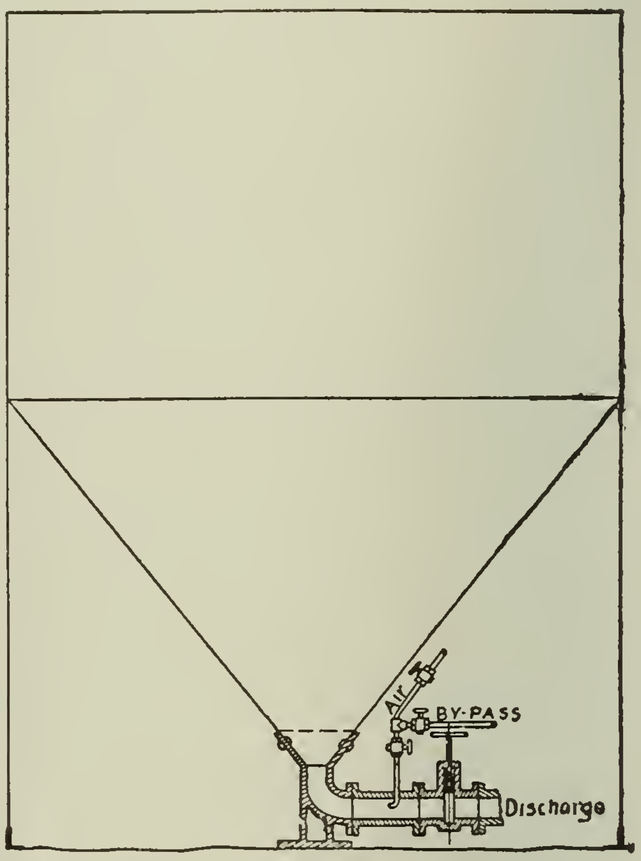
At Cripple Creek the subject of the economical handling of the low-grade sulpho-telluride ores remains still the important one. The metallurgists of this district have worked untiringly on this problem, and this year brings high hopes that the solution is near at hand. At the Stratton's Independence a second mill-unit has been started by Philip Argall. The process used here was described by P. H. Argall last year. In the middle of the summer the new mill of the Portland G. M. Co. was started. The company has been loath to give out information as to results until success had been demonstrated beyond doubt, hence the incompleteness of published descriptions of the new mill. In a general way, it is understood that low-grade sulpho-telluride ore assaying \$4 to \$5 per ton is slimed by standard methods and agitated for about six hours with cyanide solution in Pachuca vats. This is followed by another agitation with bromo-cyanide and the pulp filtered and washed. It is said the extraction is about 80%, and the cost reasonably low, considering the complex nature of the ore."—*Mining and Scientific Press*, Jan. 7, 1911, p. 67. (K. L. G.)

DURALUMIN.—" This mixture has recently created much interest in metallurgical circles, as having a tensile strength and hardness akin to mild steel, with a specific gravity of about 2·8. An analysis of this metal gave Al, 94·60%; Cu, 3·90; Fe, 0·45; Mg, 0·75; Si, 0·21: total 99·91. While the tensile strength as given for duralumin is 39 tons per sq. in.; an elastic limit of 38 tons per sq. in.; an elongation of 3·5%; and a reduction of area of 10%; tests by the Alloys Research Committee on other Al-Cu alloys approximating the composition given above did not approach these figures. It is suggested, therefore, by the *Foundry Trade Journal*, that the high strength must be the result of alloying pure materials under almost ideal conditions, one of which is a most thorough deoxidation."—*Engineering and Mining Journal*. March 18, 1911, p. 559. (M. T. M.)

THE "AIR POCKET" AGITATOR. — " Recently, while travelling through Mexico, I came across a simple and efficient system of pneumatic agitation, the details of which may be of interest to others engaged in cyaniding.

The tanks used had a cone bottom with a 50° slope. The total outside height was 20 ft. and the diameter 16 ft.

Before the introduction of the method ultimately employed, the air was introduced into the tank just above the apex. The introduction of the air at this point was not a success for the reason that considerable pulp settled on the sides of the cone bottom, and this 'dead' pulp received little if any agitation. To correct this fault a 6 in. nipple, 18 in. long, was placed between the apex of the bottom and the discharge valve (see Fig. I.), and air was then introduced into this nipple, which acted as an air chamber.



When the pressure became sufficiently great the air arose with explosive violence, carrying before it the sludge accumulated on the bottom and along the sides of the cone. Although intermittent in action, the pulsations were sufficiently close together to keep the pulp in constant agitation.

The successful operation of this simple form of agitation depends on the introduction of the air into an auxiliary chamber below the apex of the bottom.

Numerous tests by the writer and others, by prodding the bottom of the tank with a pole and also with a weighted line, failed to show the presence of any 'dead' pulp.

The fact that the tailing averaged but 35 gm. of silver per ton, bears out the statement that the agitation is highly efficient. The pulp settled, and only a few hours were required for decantation.

It is worthy of note that coarse pulp was agitated as easily as slime.

When the tube mill was shut down for renewing the liners, or for other cause, the No. 4 screens on the battery were changed to 30 mesh, and the pulp was led around the tube-mill and discharged into the tanks, where it was handled without any trouble whatever.

Shortly after the installation of this system, another engineer who had had considerable experience with Pachucas, was put in charge of the plant. Without stopping to investigate the workings of the system, he condemned it and converted one of the tanks into a Pachuca to demonstrate his personal belief in the inefficiency of the new system. After a period of several weeks, during which the results obtained by the two systems were closely compared, this engineer frankly admitted his error, and removed the air-lift from the Pachuca, and went back to the first system."—H. T. Zembler.—*The Pacific Miner*, Feb., 1911, p. 58. (H. A. W.)

A New Type of Tube Mill.—"A new type of tube mill was recently tried in Germany, which undertakes to reduce the ore from 'run of mine' size to slime at practically one operation.

The mill consists of a huge wrought-iron cylinder, about 23 ft. long, over 8 ft. in diameter for the first third of its length, and about 6 ft. for the rest. The reduction of the ore is effected by means of steel balls, which both crush and grind it. The cylinder is divided into three compartments, and the balls in each are graded in size according to the stage of grinding, beginning with balls of about 4 in. in diameter in the first section, and ending with about 1½ in. in the last. Within the cylinder are placed some ten rows of so-called 'fall-plates' of heavy steel; these lift the balls and the ore about half-way up the cylinder as it revolves, when the whole mass drops upon the next plate. Water is fed in with the ore at the rate of about one-third of a ton to each ton of ore, which means a very heavy reduction in water consumption as compared with stamps. The cylinder is kept half filled with water; hence the mineral matter within can only pass out when held in suspension by the water. The mill product is, therefore, in the form of slime.

It is held that the mill can be used in place of stamps and that the cost of installation and operation is considerably less than in the case where stamps are used in conjunction with ordinary tube mills."—*The Pacific Miner*, Feb., 1911, p. 58. (H. A. W.)

Relation of the Chemical Industry to Metallurgy.—"For many years chemical industry and metallurgy were widely separated, but at the present time their boundaries may be regarded as having met and to a large extent overlapped. So much is this the case that one often scarcely knows whether a process is chemical or metallurgical, while undoubtedly there are very many processes that are a combination of both.

Metallurgy is often regarded as an art which has progressed but slowly during our time, except as regards increase of output and economies in working costs, but those who are engaged in it know that its advances have been marked by changes as drastic as any which have occurred in chemical industry, and that in many cases processes have been so entirely altered or modified that they are almost beyond recognition. Again, materials which were once useless are employed by one or the other industry. By-products, often now almost as important as the material which once was the sole object of a process, in some cases contribute a large proportion of the total profit. From a metallurgical point of view it cannot be denied that, without the aid of chemical industry, metallurgy would not be so complete a science as it is now, the successful treatment of many ores and products often depending on both of them.

It must also be remembered that both sciences are largely dependent on a third—mechanical engineering—as in many cases the greatest difficulty incurred in working out a process is of a mechanical nature. Special plant has often to be designed, and the success of the process is made or marred by the efficiency or inefficiency of the machinery installed, for example (1) the delayed utilisation of electrolytic methods (now so important) pending the development of the dynamo; (2) the whole new science of electro-metallurgy recently opened up by the same, and by mechanical harnessing of water-power, etc.; (3) the extended metallurgical applications due to the gas producer (in the metallurgy of iron and zinc, etc.); to mention three out of many instances that might be quoted.

This leads one to suggest that anyone wishing to be successful in the economical treatment of ores or products should receive sound training, not only in chemistry and metallurgy, but also in mechanical engineering. It is not for one moment suggested that the student should be capable of erecting machinery as well as evolving chemical or metallurgical processes; but it is necessary that he should possess sufficient knowledge to enable him to realise what can or cannot be done in regard to what he starts out to do. He should know, theoretically and practically, a considerable number of standard types of machinery, common to both the chemical and metallurgical industries, and at least the chief applications of thermo and electro-dynamics, power distribution, heat production and measurement, etc.

It seems only right and proper to draw attention to the fact that practical men need practical assistants. To this end, then, a student should be given opportunities to prolong the technical portion of his education, so that he may reap the benefit of it afterwards. How often, much more often than one realises, does the machinery erected for a special purpose fail because the chemist or the metallurgist has been unable from lack of knowledge to help the mechanical engineer to understand what is required of him before it is too late. Cases have occurred where a process has absolutely failed, not from any demerits of its own as a process, but owing to the mechanical part of it failing to do the work required. It is not too much to assert that several ingenious and metallurgically sound processes have failed heretofore for want of suitable appliances now available. That the solution of several such problems has now been reached by other means does not invalidate the argument that the mechanical furnace, the vacuum filter, the converter, the contact process for sulphuric acid, and many other appliances of recent introduction, would have achieved lasting success for methods now abandoned and only remembered as failures.

The utilisation of by-products from smelting processes occupies the attention of the chemist much more than formerly. Selenium, tellurium, bismuth, gold and silver, which are often present in copper ores, follow the copper through the smelting processes and are finally collected as valuable residues during the electrolytic refining of the blister copper: they are then subjected to chemical treatment for their separation and are thus made valuable. Arsenic is collected in large quantities in the form of white arsenic from the flues of smelting furnaces, and is a valuable by-product, especially in the calcination of Cornish tin ores. Coke-oven and blast furnace gases yield by-products of great value to the chemist, of which ammonium sulphate is an example. The success of the shale-oil industry depends largely upon the profits derivable from the ammonium sulphate by-product, and the utilisation of blast furnace gases both as a source of energy and for the manufacture of sulphuric acid. Basic phosphatic slag from the iron furnaces is a valuable fertiliser, and in certain cases is worth as much as the iron itself. Some iron slags are utilised to make Portland cement.

Electric smelting seems likely to considerably alter the position of many metallurgical industries, as in the case of the work now being carried out in Norway and Sweden, where power is cheap, on zinc ores and concentrates.

The following seven were selected as amplifying the points which the author emphasised above, namely, (1) the cementation process as carried out on cupriferous pyrites in South Spain, at Rio Tinto, Tharsis, etc.; (2) the treatment of Spanish cuprous iron pyrites imported to Great Britain for sulphur and copper extraction; (3) pyritic smelting of low grade sulphide copper ores; (4) treatment of the Broken Hill lead-zinc-silver sulphide ores; (5) manufacture of copper sulphate; (6) the cyanide process in gold extraction; (7) the Pearce process of treating auriferous copper; (8) the arsenical cobalt and nickel ores of Ontario."—ARTHUR C. CLAUDET, *Journal of the Society of Chemical Industry.—Mining World*, March 4, 1911, p. 491. (A. R.)

MINING.

PULMONARY DISEASES AMONG MINERS.—"Early in 1910 a Royal Commission was appointed by the Governor of Western Australia to investigate and report upon (*a*) the prevalence among miners of pulmonary diseases, (*b*) the nature of such diseases, and (*c*) the extent to which they are associated with or consequent on mining. The Commission consisted of only a single Commissioner, the Hon. John Howard Lidgett Cumpston, M.D., B.S., D.P.H., who under date of October 4, 1910, rendered his report, which constitutes a valuable contribution to the limited literature of miners' lung diseases whether of tubercular or non-tubercular origin. The report, in an admirably condensed form, contains a considerable amount of entirely new information on the subject, and all the essential facts and conditions are discussed with rare lucidity and ability to bring the results within range of a lay-man's point of view.

Medical Data of Value to Mine Managers.—The report makes no specific recommendations, since under the terms of reference none were requested. For American mine managers, physicians, mine inspectors and others interested in the welfare of the mining population, the results of this investigation are of decidedly practical value and the method pursued in the inquiry would seem to be suitable for our own urgent needs to ascertain the actual conditions affecting life and health in metal mining, with particular reference to the probable excessive occurrence of pulmonary disease. No voluntary evidence was brought to the Commissioner's attention, which merely confirms the experience elsewhere that it is difficult indeed to arouse miners and other wage earners to a vivid realization of their own interest in matters pertaining to health and mortality. It requires to be considered, however, that personal opinion in a subject of this kind is of very limited value, and the excessive amount of verbal evidence often presented in government investigations frequently precludes the adequate consideration of the actual facts secured by scientific methods of inquiry.

The report of the Commissioner is divided into three sections: (1) Statistical, consisting of a study of the prevalence of lung disease among miners during the past ten years; (2) the results obtained in an examination of 2,050 men engaged in the mining industry of Western Australia; of these 2,050 men, 1,805 were actually at work and were examined under conditions as nearly as possible identical with the actual conditions under which they carried on their work; (3) experimental. This last section describes the results obtained in a very limited number of experiments performed with the object of elucidating the pathological changes in the lungs of small animals produced by the continued inhalation of mineral dust.

Complete Data Difficult to Obtain.—This outline of the scope of the investigation is admirable in its simplicity and comprehensiveness. Of all the information needed to establish the true incidence on health of injurious trades probably the most conclusive is the actual physical condition of the workers, obtained by a medical examination, and a truthful record of personal sickness experienced with a due regard, of course, to the factors of age, race and nativity, mine experience and mine employment as followed in one or more localities or in different mining fields. The importance of strict attention to these facts was well brought out in the investigation into the health of Cornish miners and the present inquiry fully confirms this view. Of course many other facts must be taken into account, and chief among these are post-mortem examinations, but unfortunately it is often very difficult to obtain the lungs of deceased miners other than those killed by accident. The present investigation failed almost completely in this respect and the lungs of only one miner were available for the purpose, which, of course, precluded definite conclusions.

Importance of Adequate Data.—A large amount of statistical data were collected for a period of 10 years, and the conclusions from the facts obtained are briefly set forth as follows: (*a*) Lung diseases, in the 10 years under review, have been, considered as a group, distinctly more prevalent among miners than among males over 15 years of age generally; and while this is true of lung diseases considered as a group, it is also true for the principal individual diseases of this group. (*b*) An important factor in producing an excess for miners in the group of other respiratory diseases, has been fibrosis of the lungs. (*c*) At all ages between 25 and 60 the percentage of deaths from lung diseases in each age-group is higher among miners than among the general population. (*d*) Tuberculosis of the lungs has been steadily on the increase among miners during the period, and pneumonia has been steadily on the decline. (*e*) The miner has considerably less chance of surviving until the age of 60 years than has the average male over 15 years of age. (*f*) Considering the average ages at death for miners and for all males over 15 years of age, the miner has 3½ years less of life than the average male in the general population.

Merely as an economic question the foregoing conclusions derived from a sufficiently large statistical basis are entitled to serious consideration. Mine experience slowly acquired is of too high a degree of economic utility to permit of the needless waste of the miner's health, physical strength and years of life, but most of all in a country like Australia, where there is normally a deficient skilled labour supply. If the net economic gain of miners' work for a year is placed at $300, then the difference of 3½ years in the average age at death (granting the accuracy of this comparison) means a net economic loss to the mining industry of $1,000 more or less of unrealized return on the human machine, which is, of course, the true productive force in every industry.

Tuberculosis and Fibrosis Defined.—The difference in the duration of mine life is however probably greater than the 3½ years deficiency in the average at death. The difference is primarily if not exclusively the result of an excessive mortality from lung diseases, but in this connection it is pointed out in the report, and the conclusions will be read with decided interest by mine physicians, that "the term *miners' phthisis* is used in this paragraph for the first and last time in this report. That term is an inexact one, and its use has led to a great deal of confusion in the past. The term *phthisis* has for centuries been understood to mean tuberculosis of the lungs, which is a condition having its own distinctive pathological features; but when abnormal conditions in the lungs of miners have been under discussion, the term *miners' phthisis* has been used with the intention of including indiscriminately both the fibrotic changes and those due to the tubercle bacillus. Such confusion of distinct pathological conditions should not be perpetuated; and therefore throughout this report, wherever the term *fibrosis* has been used it is to be clearly understood that the term includes only the changes produced by dust, and does not in any way refer to changes produced by the tubercle bacillus. Similarly, when the term *tuberculosis* is used it means tuberculosis, and does not include fibrosis.

It is not for a layman, of course, to take issue in a matter of this kind, but it may be said that the discussion has its practical application to the vexed question of industrial disease definition in all workmen's compensation laws. The time is probably not far distant when all industrial diseases, or in other words, diseases directly traceable to work exposure, will be brought within the scope of employers' liability and workmen's compensation legislation, and in any future consideration of the subject qualified medical opinion will be of the foremost importance.

Fibrosis More Prevalent among Machine Drillers.—Cases occur, of course, in which both fibrosis and tuberculosis occur together, but it is said in the report that this is rare in Western Australia. A complete physical examination was made of 2,050 men engaged in mine work, the men examined being divided into four groups: (*a*) Those actually at work and examined as nearly as possible under working conditions. (*b*) Those brought to notice by officials of various labour organizations; chiefly advanced cases who felt themselves unable to work by reason of affections of the lungs from which they suffered. (*c*) Miners examined at the sanatorium. (*d*) A few men who, for various reasons, desired to submit themselves for examination. Of the foregoing only the first group was considered a safe basis for final conclusions, being considered a "fair average mining population." The physical examination disclosed the following facts: (*a*) Early fibrosis was present among machine miners in the proportion of 33·16%; among non-machine miners only 7·23%; among truckers 3·1%; and among dry treatment hands 24·5%. (*b*) Intermediate fibrosis was found among machine miners and non-machine miners; late fibrosis was found only among machine miners. (It is unnecessary to specify the percentage of intermediate and late fibrosis respectively; the fact of their existence is of sufficient significance.) (*c*) Pleurisy was found to be present in men engaged in several different classes of mining. (*d*) Tuberculosis of the lungs was present in a total of 28 cases, *i.e.*, 1·5%, or, if the whole of the men examined (2,050, including the selected cases) be considered, this condition was present in 65 cases, which is 3·2%. (*e*) Early fibrosis was most commonly present among machine miners and dry treatment men. (*f*) Early fibrosis seems to make its appearance most frequently about the second year on machine work.

The Spread of Lung Disease Laid to Mining.—Lung diseases, and particularly tuberculosis, being of rather rare occurrence in Western Australia the preceding facts and conclusions prove that these diseases are being spread through the mining industry, largely however by methods of mining more or less subject to improvement and company control. The Commissioner properly calls attention to the seriousness of the present situation and remarks: "This existence of a high percentage of fibrosis among working miners cannot be attributed to the importation of such cases from places outside Western Australia. This aspect has been dealt with, and it is quite clear that the mines of Western Australia can and do produce fibrosis to an important extent; for among machine miners fibrosis is present in 25% of the men, and among the dry treatment men in 19%, these percentages are calculated for those men only who have never worked in mines elsewhere than in Western Australia. Fibrosis of the lungs has been shown in various ways in this report to be due to the action of dust; and as a general statement based on the results obtained by the experiments—and which may be taken as an indication of the lines upon which remedial measures must be instituted—it may be said that a man suffers from fibrosis to the extent to which he is exposed to the continued inhalation of the fine material, dust, in other words; if there be no dust there will be no fibrosis. The converse is equally true that the continued inhalation of dust certainly produces some fibrosis.

As a drastic measure toward the eradication of tuberculosis among miners, the exclusion of affected miners from the mines or mining communities is suggested, but also the segregation of tubercular persons, who are not themselves miners, but who come into more or less close association with them. In explanation of the point of view that tuberculosis of the lungs must be considered more in the light of an infectious than an industrial disease, it is said that: "The fact that tuberculosis is most common in men engaged on machine work has been interpreted as indicating primarily that this class of work, by reason of the close association of the two men working upon one machine, offers the greatest facility for personal infection. It has, however, been generally accepted that lung tissue which has been impaired by the action of dust is hereby rendered more susceptible to the action of tubercle bacilli. So far as I have been able to ascertain, there is no direct experimental evidence upon this point, but the fact that tuberculosis is found in all countries to be excessively prevalent among men engaged in other dusty occupations—for example, knife grinders, pottery makers, etc., must be accepted as indicating the correctness of this commonly accepted belief. Therefore, while tuberculosis among machine men is more an infectious than an industrial disease, it appears certain that the action of dust does predispose to the action of the tubercle bacillus. In this connection, the significance of the existence of a large number of cases of fibrosis must not be lost sight of."

Dust Predisposes to Lung Diseases.—The importance of dust as a predisposing factor to lung diseases is recognized by all who have considered the subject. The report contains some very interesting observations upon the effect of dust on the lungs, or the changes in lung tissue produced by continuous dust inhalation, but these do not fall within the present discussion. It may be said, however, that the effect created by the presence within the lungs of dust is stated to be inflammatory change throughout the lung substance, resulting in the production of fibrosis, and that the ultimate effects are, (1) diminished area of lung tissue available for the purification of the blood, (2) diminution in the amount of air which can reach the lungs, (3) diminution in the amount of blood which can reach the lungs.

The experimental and pathological evidence obtained by the Commissioner, therefore, seems to sustain the conclusion that: "Dust, and dust alone, is responsible for fibrosis; that the causation of pleurisy is somewhat obscure, but that here dust and cold both probably play some part; that there is not forthcoming any evidence to show that there are any other factors producing abnormal conditions in the lungs (except, of course, the tubercle bacillus) though the possibility of the rapid alterations of air pressure having some effect has been considered. It may be that these alterations of air pressure affect some in the direction of producing *emphysema*, but this is pure supposition."

High Mortality of Miners from Lung Diseases.—The medical considerations aside, the statistical evidence presented in the report is entirely conclusive that lung diseases are extremely common among the miners of Western Austrlia. While of the mortality from all causes of all males over 15 years, 19·8% was from respiratory diseases, the corresponding proportion for miners was 53·9%. Considering the mortality from diseases in detail, it is brought out in the report that the death rate from tuberculosis of the lungs was 9·6 per 10,000 for all males over 15 years of age, while the rate for miners was 19·2. From bronchitis the respective rates were 1·7 and 4·2; from pneumonia 6·7 and 22·7, from pleurisy 0·7 and 1·1, asthma and emphysema 0·5 and 1·3, and other respiratory diseases 0·5 for all males and 52·4 per 10,000 for miners. The foregoing rates prove conclusively that respiratory diseases, as a whole, are much more prevalent as causes of death among miners than among males over 15 years of age generally. Tuberculosis has steadily increased among miners from a rate of 10 per 10,000 in 1901 to 30 in 1909. There has been a decline in pneumonia, which may possibly be accounted for by more accurate medical diagnosis.

The death rates were not calculated by divisional periods of life, probably because the necessary information could not be secured. While calculations of the average age at death are apt to be misleading, it is, nevertheless, suggestive that the average age for all males, 15 and over, should have been 47·8 years against 44·2 for miners, a difference of 3½ years in favour of the general population. A comparative table is included in the report for Queensland and Western Australia, according to which the death rate from lung diseases among miners per 10,000 living was 50·03 in Queensland and 53·0 for Western Australia. The rates differ somewhat for the various forms of lung disease, but, on the whole, they confirm each other and sustain the view that the rates are at least approximately correct.

The Physical Examination of Miners.—The most important portion of the report relates to the actual examination of working miners. Of the 2,050 men examined all but 156 were gold miners actually at work, or at a sanatorium, or at union offices. There appears to have been no difficulty in making the physical examination, to which the men seem to have submitted without hesitation. It is stated that in 26 gold mines there were employed a total of 6,846 men and of these 1,805, or 20·5% were physically examined. Considered by individual employments, it is stated that 94% of the surface labourers were found normal, of the wet treatment hands 91%, of the platemen 81·9%, of non-machine miners 85·01%, of truckers 82·9%, of dry treatment hands 65·6%, and of machine miners only 52·6%. In other words the surface labourers and wet treatment hands exhibited the best physical condition, while dry treatment hands and machine miners exhibited the worst.

Pleurisy, as more or less complicated by fibrosis, was found to exist among machine men to the extent of 44·9% of those examined; among dry treatment hands 34·3%, among non-machine men 13·2%, among wet treatment hands 8·3%, and among surface hands only 5·2%. Early fibrosis was found among the men in the following order: Machine miners, 33·2%; dry treatment hands, 24·5%; non-machine miners, 7·2%, and truckers, 3·10%. Intermediate fibrosis was only found among machine miners to the extent of 3·6% and among non-machine miners to the extent of 1·6%. Advanced fibrosis was met with only among machine miners, but to the extent of less than 1%. Of course, men really seriously affected would not have been at work.

Each Trade Has Its Special Disease.—Tuberculosis was found among machine miners to the extent 2·2% and among non-machine miners to the extent of 1·0%. The conclusions are summarised as follows: "Machine men suffer from the three grades of fibrosis, from emphysema, pleurisy, and tuberculosis—the disease most prevalent among this class of workers is fibrosis. Non-machine miners suffer from the first two grades of fibrosis, from emphysema, pleurisy, and tuberculosis. Trackers suffer only from early fibrosis, and pleurisy, and considerably more from pleurisy than from fibrosis. Dry trearment hands suffer from early fibrosis, but not from the more advanced stages of the disease. There may be two reasons for the freedom from the advanced stages—the fact that they work in the open air, and therefore would not be likely to get so much dust in a given time as men working below on machines in a confined space: or, on the other hand, the explanation may be that men on the dry treatment plants have been working for a shorter period at that class of mining than the machine men, and are also much less exposed to rapid changes of temperature. Surface labourers were affected only with pleurisy, to the extent of one in twenty. Wet treatment hands were affected only with pleurisy to the extent of 8%."

While the present investigation is probably of greater interest to mine physicians than to mine managers, the following brief account of the main features of each of the stages of fibrosis will prove useful in diagnosing the disease in its early stages. It is said in the report that, "That the early case is the man who finds that he is continually getting "colds," associated with cough, which he finds great difficulty in throwing off; he finds that his wind is not as good as it used to be, and that he cannot run so far nor work so hard as formerly, and in whom on examination the effort of expansion of the chest wall reveals a rigidity—a rigidity which has characteristics entirely its own, and which is not so far as I am aware, seen in any other disease. At this early stage the man is apt to smile indulgently if it is suggested that there is anything wrong with his lungs, although he recognises clearly that his wind is not so good as formly and that he gets colds more often than he used to and cannot throw them off so easily."

Diagnosis of Fibrosis.—"The next stage in the disease is marked by the fact that the intervals between the attacks of "cold on the chest" are less frequent and less lengthy—the bronchitis has become an established fact; the wind is now always short; occasional attacks of "asthma" occur, although hardly ever at night; expectoration is frequent, and a certain blueness of the lips and the face indicate that the heart is beginning to feel the strain, on examination the rigidity is more marked, roughening of the breath sounds confirm the hypothesis that the bronchitis has become established; emphysematous changes may be present, and the heart signs show the reason for the blueness of the lips and face. But the subject can still do an average day's work.

"When the subject finally has to cease work the advanced stage has been reached and then all the features above detailed are emphasized; the heart and lungs becoming progressively less able to carry out their functions."

In addition to the foregoing, it is pointed out that fibrosis makes its appearance, as a rule, about the second year of machine work. The cases of intermediate fibrosis were all discovered among men who had been engaged on machine mining for not longer than 16 years. With pleurisy the greatest incidence occurred before the fifteenth year after the commencement of machine work. Of the total of 424 normal men, 215, or more than half, had been working on machines for less than 4 years only, while 86 % of them had not yet reached their tenth year of machine work. In explaining the causes responsible for the occurrence of fibrosis among machine miners, it is pointed out that dust is a common factor operating directly or indirectly in the production of both fibrosis and pleurisy. The former conclusion is advanced that the mines of Western Australia can and do "produce fibrosis to a considerable extent, both among machine miners and dry treatment men; and it is to be expected that as the years go by there will be an increasing number of deaths from this cause and an increasing number of cases of fibrosis, if the conditions obtaining in the past remain unaltered in the future."

Dust Prevention Essential. Thus, in its final analysis, the problem of health in metal mining is largely a question of dust prevention. It hardly requires to be said that machine miners are practically rock drillers, working under almost identical conditions in the quartz mines of Western Australia, on the Rand, in Cornwall, or in the western states of North America.

While not all of the conclusions arrived at by the Commissioner can be accepted as final, the report unquestionably constitutes one of the most valuable contributions to the literature of health in mining. The results of the physical examination of the men alone makes the enquiry a most important one and suggests the utility of a similar investigation in this country. It is certainly significant to find that out of 805 machine miners 267, or 33·2 %, were found to be effected with early fibrosis and 29, or 3·6 %, with intermediate fibrosis, and 4, or 0·5 %, with advanced fibrosis. Of the remainder of the machine miners, one was affected with emphysema, 62 with pleurisy, and 18 with tuberculosis. In other words, only 424 machine miners, or 52·6 %, were found to be in normal health."—Frederick L. Hoffman. Discussion on Western Australian Royal Commission on above Diseases.—*Engineering and Mining Journal*, March 11, 1911, pp. 523-526. (M. T. M.)

The Adaptability of Reinforced Concrete to Mining Engineering.—"*Introduction.*—Whilst advocating the use of reinforced concrete in mining engineering, with a view to increased safety both above and below ground, the writer regrets that he is unable to give actual examples of work carried out in this country. He is of opinion, however, that this material is eminently suitable in lessening some of the dangers met with in everyday mining work.

For the benefit of those members who are not familiar with the functions performed by the concrete and the steel respectively, a short *précis* of the theory of reinforced concrete may not be out of place. The theory upon which a reinforced-concrete beam or column is designed is extremely simple as regards first principles. Concrete in compression is considerably stronger than in tension, and the best authorities, such as the Royal Institute of British Architects, recommend as a safe stress 600 lb. per sq. in., whilst they consider the tensile stress to be almost *nil*. There seems, however, to be considerable difference of opinion amongst engineers, both in Great Britain and abroad, as to whether or not, in calculating designs, the resistance of concrete in tension should altogether be neglected. Prof. Hart, in America, and Mr. Considère, in France, advocate that an allowance should be made for this resistance, whilst the majority of other experts neglect such an allowance, and the writer thinks that it is better to do so at the present time. The problem is, therefore, to strengthen the structure in some manner in the portion subject to tensile stresses, leaving the concrete to resist the compressive stresses by its own strength, so long as the latter are within the limits of the safe value allowed—that is 600 lb. per sq. in.—as, if not, a certain amount of reinforcement will also be required for compression. Steel being very strong in tension, it is eminently suitable to take up all tensile stresses, so that a combination of steel and concrete makes the structure strong in tension and compression.

It is fortunate that, so far as elongation or contraction due to changes of temperature is concerned, both materials may be taken as acting together. Prof. W. D. Pence, of Purdue University, U.S.A., has made considerable research in the attempt to discover the co-efficient of expansion of Portland cement-concrete, and has found it to be, on an average, 0.0000055, the co-efficient of expansion of steel being taken as 0·0000067. Taking the highest values, the difference of deformation of the concrete and steel for a change of temperature of 70° F. (21° C.) would be only 0·12 in. for a length of 100 ft. Mr. Coignet gives for every 1·8° F. (1° C.) rise in temperature the co-efficient of expansion for steel as about 0·0000148, and for concrete, 0·0000135. From these figures, it is evident that nothing much is to be feared so far as temperature stresses are concerned, except in walls of considerable length.

The simplest form of efficient re-inforcement for resisting tensile stresses appears to be ordinary round steel rods. The shearing stresses of a structure have, however, to be taken into consideration, and the general method of resisting these is by the use of what are known as 'stirrups,' which are bars or strips fixed to the main reinforcing bar at intervals, and bent to certain angles. These stirrups (or shear-members) are generally bent at an angle of 45° to the main tension rod; they thus pass through the planes of rupture at right angles, and therefore are of maximum efficiency in preventing such planes of concrete from tearing asunder. It has also been found desirable that the stirrups should be rigidly attached to the main tension rod, as the curve giving the results of the French Government's official tests on reinforced concrete beams, show that beams thus reinforced are about 20 % better than those reinforced with loose stirrups. Loose stirrups have also the disadvantage of being liable to be displaced by the workmen during the process of ramming the concrete, an operation most essential in order to obtain good workmanship.

Systems.—With a view of explaining more thoroughly the theory and general practice of reinforced concrete, a brief *resumé* of a few of the principal systems and methods of reinforcement will be given. Owing, however, to the almost innumerable patents now on the market, such an account can only embrace a few of the leading systems.

Kahn System.—This system consists of special diamond-sectioned rolled bars, with projecting wings on each side. The wings are sheared, and bent up to any angle desired to form the reinforcement against shear, the desirable feature being that such shear-members or stirrups are rigidly attached to the main tension-bar.

Indented-bar System.—The exponents of this system claim that if smooth bars are used as reinforcement, the adhesion between the concrete and steel is uncertain, and liable to be further reduced by moisture, vibration, etc. They consequently advocate steel bars having an elastic limit of more than 50,000 lb. per sq. in. and indented throughout their length, the area of their cross-section being the same at any point.

Expanded-metal System.—Expanded metal is made from sheets of rolled steel in varying thicknesses, which are cut and expanded by special machinery into meshes of a diamond shape. For reinforced-concrete work, it affords a splendid bond, and is excellent for foundations, floors, roofs, etc.

Coignet System.—In this system bars are also introduced in the compressed portion of the concrete, thus making it possible to reduce the cross-sectional area of the beam, thereby lightening the structure. The compression and tension members are connected by light rods, which also take up the shear stresses.

Lock Woven-mesh System.—The reinforcement in this system consists of a woven wire mesh, the wires (with an elastic limit of 64.000 lb. per sq. in.) which form the mesh being locked at their intersections. This form of reinforcement is extensively used for floors, walls, etc.

Hennebique System.—This system consists of the use of ordinary plain bars, no special sections being required. The bars are either bent to take the shear stresses, or scrap-pieces or stirrups are hooked on the main rods, these stirrups being generally at right angles to the main rods.

Considere System.—Mr. Considere, the inventor, favours the use of spiral binding for columns, piles, etc., and for securing the continuity of the beams.

British Reinforced Concrete Engineering Company's System.—This firm favours the use of high-carbon steel for their stirrups and carbon-hoops. The former are sprung on the main tension-rods at any point desired, whilst the latter are threaded on the verticals, and are exceedingly efficient for columns, struts, piles, etc.

The writer does not advocate any one system in particular, but merely wishes to point out the characteristics of each. The intending user may therefore adopt that which is most suitable for the class of work required, as in some structures a combination of several may tend to greater economy and efficiency.

Notwithstanding the number of systems, a colliery manager can often utilize old steel ropes, rails, chains, hoops, etc., which provide fairly efficient reinforcement. A little rust is in no way detrimental, as steel embedded in concrete does not oxidize.

Aggregates.—All aggregates for reinforced concrete should be clean, hard, and free from dust, and should be such as will pass through a $\frac{3}{4}$-in. sq. mesh, and refuse one of $\frac{1}{8}$-in. mesh. Granite chippings and good clean ballast give excellent results, and make a good, tough, fire-resisting concrete. To engineers connected with mining the use of slag, coke-breeze, clinkers, burnt spoil-heaps, etc., is particularly interesting, as such material can often be obtained very cheaply in the locality of collieries. Many collieries have considerable quantities (considered more or less as being of no particular value), which can be safely used with a little care. A great deal depends, however, upon the chemical combination in which sulphur (which is generally found in some form in these aggregates) is present, for, if in the form of sulphide, the chemical equilibrium is unstable, whilst in the form of sulphate it is practically harmless.

Coke-breeze, although making a good fireproof concrete, is not advisable for use in workings where water is likely to be encountered, for, being porous, there is always a risk of the moisture percolating into the concrete, which may after a long period result in the corrosion of the reinforcement.

In connection with aggregates, it is advisable to mention that broken brick and pumice should be as far as possible avoided, and, if used at all, should be chosen with great discretion.

Sand.—It is necessary that all sand for use in reinforced concrete should be fairly coarse and clean, although a slight percentage of loam is in no way detrimental. A very find sand tends to weaken the concrete, so that the use of soft 'floury' sand should be carefully avoided. Seashore sand is perfectly suitable for concrete, the only drawback being that the presence of salt retards the setting of the cement in the earlier stages, although it does not affect the ultimate strength of the concrete.

Cement.—All cements complying with the British Standards Specification can be safely used. Cheap foreign natural cements are certainly not to be recommended, whilst in the case of slag-cements, there is always the risk of the presence of sulphur.

Great care should be exercised in mixing the concrete, which should be made plastic so that it can be rammed thoroughly into all parts of the forms, and especially around the reinforcing bars. The water used in mixing should be clean and free from organic matter, acids, etc.

For reinforced-concrete work the following will be found to make a good mixture, which is in the proportion of 4, 2 and 1, in the order named :—

Ballast or chippings ...	27 cub. ft.
Sand	13½ ,,
Cement	5 cwt.

Centreing.—Great care is necessary in the erection of the centreing, as it must be well stayed and braced, so that neither the weight nor the ramming of the concrete will make it bulge or sag. It should also be closely jointed, to prevent the liquid cement from oozing out before the initial set..

The absence of deterioration in reinforced concrete is a very strong argument in favour of its use in mines, for, although perhaps a little more expensive in initial cost than timber, it is far more economical in the end, as there is no cost of upkeep ; in fact, the structure actually increases in strength with age.

Use in connection with Electricity.—In view of the fact that electricity is at present used extensively in mines in order to increase the output, the writer would like to take here the opportunity of pointing out the suitability of reinforced concrete for constructions such as switchboards, shelves for protecting cables, etc.

To mining engineers who may have found some difficulty in protecting bunches of cables underground, the following suggestion may be interesting.

Wind first a sisal rope, ¾ in. in diameter, spirally around the cable with a pitch of about ½ in., then apply by means of a pad and brush a mixture of 1 part of cement and 2 parts of sand. This not only affords a protection against mechanical injury, such as accidental damage by tools, rough handling by inexperienced workmen, etc., but serves to locate a short circuit in the cable by the bursting of the cement—in fact, it is even superior to asbestos.

Mining Applications.—A few of the many applications of the use of reinforced concrete in mines are briefly stated below :—

Pit-props.—These being fireproof, are very reliable, and although slightly heavier, are very much stronger than timber.

Lining of Underground Roadways.—In this connexion the writer would draw attention to the adaptability of reinforced-concrete slabs, say 3 in. thick, reinforced to stand heavy breaking loads, and prepared and matured above ground. These can be utilised for supporting the roof of the roadway with beams at, say, 6 ft. centres.

Reinforced-concrete lining for main roads was adopted about 4 years ago at the Béthune Mines, in the Pas-de-Calais coal-field, for the protection of a certain section of the road originally supported by steel girders resting upon masonry walls about 18 in. thick. It is interesting to note that the present lining of reinforced concrete is only about 8 in. thick, and that no skilled labour was used in its construction.

There are four distinct advantages in lining roads with reinforced concrete, namely—(1) resistance to explosions and shocks ; (2) undue accumulation of dust prevented, owing to smoothness of surface : such a road can be swept or watered ; (3) less resistance is offered to the air-current ; and (4) there is no depreciation in strength and efficiency concurrently with age.

At the Providence Pit, Charleroi (Belgium), which is the deepest colliery in the world, a scheme has been devised whereby reinforced concrete will be used to support the whole length of a level subject to great pressures. The Philadelphia & Reading Coal & Iron Company, Pennsylvania, use reinforced concrete extensively in their roads, where, owing to double tracks, long spans and arches supported on columns are necessary. When first employed, these columns were made 24 in. sq., but the later ones have been reduced in size to 18 in. Old rails, conveniently bent, wire ropes, etc , were used as reinforcement, and are easily and readily obtained on the spot.

Sleepers.—For sleepers in mines where the wear and tear is very considerable, reinforced concrete is an ideal substitute for timber, and although slightly in excess in first-cost over the latter, such sleepers are far more economical in the end, as they are practically indestructible.

Engine Foundations.—For foundations of all descriptions, reinforced concrete is incomparable, being much better than mass-concrete, besides saving a considerable amount of excavation.

Bins.—Many coal and ore-bins have now been constructed of this material, the saving in first cost and annual upkeep for scrapping and painting over steel being very marked.

Airways.—An excellent use of reinforced concrete pipes (which can be obtained in lengths up to 18 ft.) may be suggested here—that is, for the lining of airways, thus giving a good smooth passage, as there is always a little difficulty in keeping small airways thoroughly clean and free from falling *débris* when constructed in the ordinary way.

Bulkheads.—The writer believes that it is fairly common practice nowadays to build bulkheads in the galleries for resisting large quantities of water, and reinforced concrete is very efficient for their construction, being very strong and capable of resisting great bursting pressure.

Sanitary Uses.—In addition to the few uses of reinforced concrete above mentioned, and the many others too numerous to be described, the writer would draw the attention of the members to the value of this material for raising the standard of mining hygiene. He fears that this matter does not meet with the attention which it merits from those responsible for the care and safety of the colliery. Excellent sanitary stabling arrangements can be made by the use of reinforced-concrete slab-work, made as before mentioned, the reinforcement necessary being light steel meshwork. The great advantage of using this material is that it can be thoroughly cleansed by swilling with a hose, and, in addition, organic acids have no deleterious effect.

It is impossible to prophesy at the present period what will be the ultimate uses to which reinforced concrete will be put ; but there is no doubt that the material has now passed the experimental stage, and offers a vast field of investigation and enterprise to those interested in mining engineering, whether from the standpoint of economy or from that of improved hygiene and efficiency of the employees."

In the course of a short discussion on the foregoing paper, the President (Mr. Houfton) said that while he had seen a good deal of reinforced concrete on the surface of collieries, he had not yet had any experience of its use underground. He recently had a quotation for a complete headgear of reinforced concrete, which the makers were prepared to guarantee would be indestructible and stronger than timber or lattice steel work. The price came out at about 30% to 40% above that of timber, and more than 50% less than lattice steel headgear. He hesitated to adopt it because of the difference, slight though it might be, between the co-efficiency of the expansion of cement and iron, and he could not see what was to prevent the wet getting into the cracks. If it would be cheaper for use underground than bars and props, and cheaper than arching, many of them would be glad to adopt reinforced concrete."—R. N. MIRZA.—Trans. Institution Mining Engineers (Mid and Counties West, Eng.), Jan. 25th, 1911, pp. 12—19. (A. R.)

EARTHQUAKES AND MINING.—"Professor J. W. Gregory, Glasgow, contributed a paper on "The Glasgow Earthquake of December 14, 1910, in Relation to Mining."

At the outset he observed that earthquake phenomena present several problems of special interest to miners, although the actual shocks may be less felt underground than on the surface. Thus it has often happened that an earthquake which has done serious damage has not been felt by men at work underground. For example, the disastrous earthquake in Northern India of June 12, 1897, passed unnoticed in the Bengal coal mines. Nevertheless earthquakes may produce several important mining effects. They may lead to slight dislocations which may cause falls of roof ; or they may produce strains which are relieved by sudden rock explosions or "air blasts," such as have long been known in Cornwall, and are especially frequent and fatal in the Kolar goldfield in Mysore, where they have been carefully investigated by Dr Smeeth.

The suggestion has been repeatedly made that careful seismometric observations should be taken in the neighbourhood of dangerous coal mines, and although such work has been undertaken in Japan and France, and for a short period at the Marsden (now Whitburn) Colliery, near South Shields, the writer is not aware that any adequate observations on this subject have been made in the British Isles. As an opportunity for again calling attention to the need for fresh evidence regarding the relation of seismic activity and some kinds of mining accidents, he (Professor Gregory) gladly accepted the invitation from the secretary of the institute to forward a short account of the recent Glasgow earthquake.

The general results may be summarised as follow:—The earthquake was one of a series, several of which were very powerful and wide-reaching. The series began on December 14 with a severe earthquake along the coast of South-eastern Africa at 11.40 a.m., Greenwich mean time. The first earthquake of the series recorded in Scotland was a slight shock felt at about 7 p.m. at Ayr. It was followed at 8.54 p.m. by a shock which was felt throughout Glasgow and in the adjacent districts. The epicentre of this shock was clearly to the north-west of Glasgow about Possil and Ruchill whence the earthquake wave travelled northward as far as the Blane Valley, and southward under Glasgow to Barrhead and Cambuslang, south-eastward to East Kilbride and south-westward to Lochwinnoch. Eastward it was felt as far as Glenboig and westward at least to Bowling, while either the main shock or a secondary after-shock at 9 o'clock was felt at Dumbarton. Secondary shocks were recorded from the line of the Highland boundary fault across the Roseneath peninsula.

The area in which the shock was appreciable included about 300 square miles. The area of sensible disturbance was practically confined to the rocks of the carboniferous limestone series. It was felt on the outcrop of the under-lying lavas of the calciferous sandstone series near Barrhead, Strathblane, and Mugdock; but in these cases the earthquake was very slight, and observed only by a few of the residents. The shock was practically confined to the sedimentary rocks of the carboniferous limestone series. In the neighbourhood of Possil the intensity was as high as 5·5 on the Rossi-Forel scale, and the earthquake may therefore be described as between moderate and fairly strong.

The Glasgow earthquake exhibited the characters of a fault movement; and the major fault nearest to its probable centre of origin passes to the south of Possil, and brings the millstone grit against the carboniferous limestone grit near Garnkirk and Gartsherrie to the north-east of Glasgow. This fault has not been traced far west of Possil. The suggestion has been made that the earthquake was due to the collapse of some old mine workings, and it is advisable to consider the possibility of the cause having been subsidences in the abandoned mines of the Possil coalfield. This explanation is not likely to find favour with mining engineers, with whom the subsidence of mine workings is a familiar experience. Such occurrences produce shocks which may be violent, but are very local in range. They do not disturb areas of hundreds of square miles. Many pseudo earthquakes due to such causes have been investigated. A local example was the earthquake reported at Kilsyth on February 16, 1898. The shock was accompanied by sounds like the fall of a heavy body or of a load of coals; it caused a sharp vibration, which lasted for about a second, and was strong enough to rattle windows, overthrow crockery, and ring the bells in one house. Unlike ordinary earthquakes, the shock appears to have been more alarming underground than on the surface. The area affected was, however, very small. Dr. Davison investigated the reports and concluded that the effects described were due to the sinking of the rock above a worked-out coal seam, where the roof was especially weak from the immediate neighbourhood of a fault. The movement was a fault slip, and the phenomena were therefore like those of an ordinary earthquake; but the cause was probably of artificial origin. The range of the disturbance caused by the recent Glasgow earthquake indicates that it was not due to mining operations, and the depth at which it doubtless originated supports the same conclusions. The determination of the depth at which earthquakes originate is a problem at present insoluble. They probably always originate within a few miles of the surface, but none of the methods proposed can determine the depth even approximately, except in such cases as that of Ischeá. It is, however, generally accepted that the damage caused by an earthquake of shallow origin is exceptionally high in proportion to the area shaken. We may safely conclude that the recent Glasgow earthquake must have been due to a movement at the depth of many thousands of feet. If it had been caused by a subsidence only 2,000 ft. beneath the surface, it could not have affected so large an area without having done more serious damage immediately above its place of origin. The movement which caused it was probably in the deep roots of the Possil fault, and far below the level of any mining operations, which did not there extend much below 1,500 ft. The writer, however, is bound to remark that a different explanation has been arrived at by Mr. G. W. Walker, the director of the Estate Mines Observatory. He is disposed to regard the earthquake as antipodal in origin, and having reached Glasgow through the centre of the earth."—Mining Institute of Scotland.—*Iron and Coal Trades Review*, Feb. 17, 1911, p. 319. (A. R.)

THE ASSAY PLAN FACTOR.—"In our January impression, some reference was made to the ratio of the assay plan indication to the actual revenue, and it had been urged previously that 'even with present day metallurgical treatment to go over 80 per cent. might involve a strain on the mine.' The London *Daily Mail* in the course of an article which should have been read by all interested in mining shares, indicated from 80 to 85 per cent. We now have these ratios confirmed by Mr. W. McC. Cameron, consulting engineer of the Goerz group, who says that 'a yield of 82 per cent. of the mine assay value of the reserves may be expected' in the case of the May Consolidated. This company is not equipped with tube mills, but against this the reef width is above the average. Last year the recovery was 89 per cent. of the screen samples, or say 6 per cent. below that of concerns possessed of most up-to-date metallurgical equipment. Consideration of these facts would point to a factor of about 85 per cent. But, of course, the more scientific method is to examine each proposition upon its merits. We notice that the Barnato mines have adopted the excellent practice of giving the 'actual recovery value' of their reserves."—*The South African Mining Review*, March 1911, p. 44. (A. R.)

ROCK DRILL TESTS IN FRANCE.—"M. Deschamps, mining engineer, has published in the *Annales des Mines* an important article on the results of the comparative trials made at the iron-pyrites mines of Sain-Bel with different types of drills, which, having been employed there during a long period, have made evident the advantages and defects of each pattern. Those tested were :—

François, hand rotation, of 8·5 and 9·1 kgm.
François-Bolido, automatic, of 11·0, 12·6, & 19·0 kgm.
Ingersoll-Rand, hand rotation, of 9·0 kgm
Ingersoll-Rand, valveless, hand rotation, of 10·5 kgm
Bornet, hand rotation, of 9·3 kgm.
Flottmann, automatic, of 14·7 and 12·0 kgm.

These borers have been employed in the stopes of pyrites and also in rock. The pyrites of Sain-Bel possesses a granular texture and is intimately commingled with a small proportion of quartz, forming a very compact rock. The drivages have been made in rather easy slate, and in hard slates somewhat resembling quartzite.

The Ingersoll-Rand drills of 9 kgm., which entailed frequent repairs, are no longer used. The François of 8·5 kgm. gave a better result, but were considered too light, while some of the parts became injured. With the François 9·1 kgm. and the Valveless of 10·5 the effect was more satisfactory, as they are powerful, work regularly and effectively. The Flottmann drill of 14·7 kgm. has manifested its incontestable strength by operating several months without requiring any repairs, and notwithstanding that the miners are habituated to the use of lighter drills, they quickly appreciated its good qualities. The Bornet drill has the advantage over the other types of working as well in wet as in dry ground, on account of the injection of water which enables it to advance rapidly, but it requires frequent cleaning and repairs. The François-Bolido of 19 kgm. has been deemed too heavy for the customary work of exploitation, although it is fitted with a support which advances automatically, and which rather complicates it. It is not so well adapted for horizontal borings as for shafts with vertical holes, where it works satisfactorily. The François-bolido of 11 kgm. and the Flottmann of 12 kgm. have been successful, though inferior in strength and operation to the Flottmann of 14·7 or the Bolido of 12·6 kgm., but the latter are not so easily managed. The cutters of the drills employed have been either a star with six radiants or a Z, the last being preferable because it breaks away larger fragments and reduces the dust, and because the Z progresses more speedily than the star.

In addition to the drilling experiments, there have been tests made of the consumption of compressed air and of the rate of advance of the different forms of drills used. These tests were holes bored in Vittlebois limestone with drills of 35 mm. diameter. Each drill was submitted to three trials of five minutes each, with an air pressure of 4,200 kgm. The results appear in the following table :—

Drills.	Weight kilogrammes.	Aver. litres of compressed air per minute.	Millimetres advance per minute.	Litres of air consumed per cm. of advance.
Hand rotation—				
François	8·500	212	19	107
,, ...	9·100	161	22	72
Ingersoll	9·000	165	17	97
,, Valveless	10·500	120	21	57
Automatic rotation—				
François-bolido ...	19·000	214	87	24
,, ...	11·000	193	71	28
,, ..	12·600	186	86	21
Flottmann ...	14·700	194	78	25
,, ...	12·000	197	70	28

The above-cited figures demonstrate that the employment of pneumatic drills in the above mining works has sensibly augmented the output of the miners and reduced the cost of breaking ore. The estimation of the cost indicated should be somewhat raised to allow for the cost of compressed air, expenses of maintenance, and the wear and tear of drills. A question inseparable from the use of the pneumatic drill is that of the production of dust, since, with the object of increasing the advance, it has been attempted to clear the drills of the sand by injecting air through a channel going down the centre of the bar to its head ; could this purpose be attained progress would be much faster. The only means which has given relative satisfaction is the substitution for the usual hollowed bar of a drill with an outer screw running down its whole length ; the

M. Deschamps gives the following comparative results obtained from the working of the Flottmann drill of 12 and the François of 12·6 kgm., operating in the same cuts or stopes :—

Machine Drills.

Workings.	Extraction per day. Cubic Metres.	Cost per cubic metre. Frs.	Explosives per cubic metre. Frs.	Labour. Frs.
Rise in the slates ...	0·810	12·50	2·95	7 82
Winze in easy slates	0·249	30·35	7·06	5·80
Pyrites—				
Cut A	1·78	5.25	1·70	6·80
,, B	2·16	6·00	2·07	8·63
,, C	2·42	6·60	2·19	10·69
,, D	2·29	6·00	2·50	8·00
,, E	2 22	6·00	2·15	8·93
,, F	1·61	5·47	2·56	4·72
Average ...	2·08	5·89	2·20	7·88
Pyrites—				
Cut G	2 27	3·50	0·52	6·95
,, H	1·84	4·00	0·76	6·23
,, I	2·72	4·00	0 65	5·42
,, K	2·11	3 75	0·82	6·37
Average ...	2·23	3·81	0·69	7·24

Hand Drills.

Pyrites—				
Cut A	0 80	8·96	3·32	4·50
,, B	1·59	7·54	1·84	9·05
,, C	1·03	8·60	2·35	6·42
,, D	0·87	10·13	2·68	6·52
,, E	1·58	8·08	2·18	9·34
,, F	1·02	10·14	2·72	7·54
Average ...	1·15	8·91	2 51	7·23
Pyrites—				
Cut G	2 00	4·04	0 61	6 84
,, H ...	1·51	4·35	0·57	5 67
,, I .	1·15	4·90	0·[illegible]7	4·57
,, K ...	1·81	4·40	0 50	7·03
Average ...	1·62	4·42	0·66	6 03

rotation of the drill would determine the separation of the débris in the same manner as does a carpenter's gimlet. The difficulty lies in the augmentation of the size of the boreholes. Drills having an injection are 26 to 32 mm. in diameter, which produce holes of from 30 to 36 mm. As the strength of the drill rests in the solid centre, it is obligatory to take the same section of 20 mm., and this with a helice of five would give a total of 30 mm., and with a cutting face of 32 mm. the hole would attain 36 or 37 mm. But practice has proved that increase of 5 or 6 mm. in the width of the bore nearly doubles the duration of the time, besides diminishing the density of the explosive charge. The helicoidal drills, with a solid centre of 16, or a total diameter of 26 mm., have been tried: but the breakages, especially of the cutter, were much more common; they were rather quicker than ordinary drills.

The speed of advance in the stopes of iron-pyrites, with different forms of cutting heads, have been as under:—

- Hand rotation—
  - Hollowed drills, with star cutters, 4 cm. per minute.
  - Hollowed drills, with Z cutters, 4 to 5 cm. per minute.
- Automatic rotation—
  - Hollowed drills, with star or Z heads, 5 to 7 cm. per minute.
  - Helicoidal drills, with Z cutters, 3 to 4 cm. per minute.

London Mining Journal, Feb. 18, 1911, p. 160. (A. R.)

Ropes with Signal Wires.—"A new kind of rope used in mine hoists in German mines is described in *Berghau*, which has electric conductors woven into the iron ropes. The centre core of the rope is formed by insulated copper wires which are to connect men in the cages to engineer at hoisting engine or with the power house. One pair of wires carries telephone current, another pair carries current for electric bell signals to engine room. The third pair has the electric light current. One set of wires is often used to connect the hoist direct with a switch at hoisting engines, so that, in case of an accident, the cage can be stopped by the men by turning a switch in the cage, which will stop the engine up in the engine room. Each copper conductor is enclosed in a good strong vulcanized rubber insulation. To protect this from the effect of acids and greases, a special impregnated covering is the next layer. The conductors are then woven into one strand, and encased again by a thick layer of specially treated hemp rope. This complex winding prevents conductors coming in contact with each other and short circuiting. Leakage of current in to outside iron and steel wires is also eliminated. Two ropes of this type, each 330 ft. long and 2 in. in diameter, have been used at the Königsgrube mines in Germany with satisfaction."—*Mining World*, Jan. 28th, 1911, p. 248. (A. R.)

Concrete Shaft Work in Mines.—One of the greatest pieces of concreting undertaken in the Lake Superior region is that in progress at the Cleveland Cliffs Iron Co.'s Maas mine at Negaunee, Marquette range. The shaft at this property was sunk a number of years ago. The overburden was of quicksand, 167 ft. in depth, and to penetrate to the ledge required many months' hard work and the expenditure of a large sum of money. Millions of gallons of water and sand were pumped out of the opening, and long before the rock was reached, there had been created a huge excavation, projecting from the centre of which was the shaft timbering. There was much twisting and buckling, as was to be expected, and when the shaft had attained the ledge it was badly out of alignment. With all the settlings and draws, and upboilings and slides, it was an utter impossibility to keep it in line. To hold it anywhere near alignment was great luck, and to keep it open, was the thing hardest of accomplishment. Hoisting operations have not been satisfactory, due to the shaft's irregularities, and with the mine now well opened and prepared to furnish a large tonnage of ore, it was decided to put the property in the best possible condition by making the outlet safe and stable. This is being done by giving the shaft a heavy lining of concrete from the surface to a point some 30 ft. below the ledge. The work is being performed by the Foundation Co. of New York.

The concreting will be dropped from the surface, the weight of the material forcing the structure downward. It will be of very massive construction and will weigh enormously per running foot of completed circle. The outer portion of the shaft will be round, while the dimensions of the interior will be 14 ft. 10 in. by 10 ft. 10 in. The walls will not be of the same thickness. They will be 7 ft. 11 in. at the bottom, and for a considerable distance above it, and from the latter point will gradually narrow, until at the surface the width will be 2 ft. 6 in. The greatest thickness will be where the pressure exerted against the concrete from the surrounding formations will be the heaviest, which will be at the bottom of the shaft, the width growing proportionately less as the surface is approached. The walls will be reinforced with steel. The concrete will be carried well into the ledge. The latter rises up in one side of the shaft at an angle of 70°. A portion of this rock has already been cut away and the balance will be taken out as the task requires. When finished the concreted portion of the shaft will be about 200 ft. from surface to bottom. Concreting this shaft is a huge and at the same time a most delicate piece of work. The sand will run like water, and in almost inexhaustible volume if once it is given a chance, and the utmost caution must be exercised. Failure to take proper care would easily wreck the entire shaft. The old timbering is being removed slowly and cautiously. It is in bad condition, notwithstanding its comparatively short term of service, and is being replaced none too soon. Underground, the Maas is connected with the Cleveland Cliffs Negaunee mine, by a long drift, so that in the event of an accident at either property a safe outlet is available. When completed the Maas will be equipped with a shaft that will stand as long as the mine has ore to take out, and this will be a great many years.

The Cleveland Cliffs Co. is also concreting its Negaunee mine shaft. This work, while by no means easy of accomplishment, is less difficult, not so hazardous, and much less expensive per lineal foot than at the adjoining property. The Negaunee's overburden is quite as unstable as that of the Maas, There were several bad runs of ground prior to the Cleveland Cliffs taking hold, lives were sacrificed and the wrecking of the property was threatened. The concreting of the shaft, which has been in progress a number of months, is now nearing completion. It will not be many weeks before it will be possible to begin installing the shaft equipment. The concrete has reached the 9th level, the principal working point in the property, and will be continued downward until the 10th level is attained. Concrete pockets are to be put in at the 9th level. These will be the first underground pockets of that nature on the Marquette range They will be reinforced with steel and will be lined with the same material. The shaft will be of concrete from top to bottom. In practically all of the concrete shafts in the Lake Superior district the concrete reaches from the surface only to the ledge, but in the Negaunee shaft the ledge has been cut away in order to make room for the concrete walls.

The original shaft at the Negaunee mine, which was sunk under the direction of the late Captain Samuel Mitchell, who was one of the best miners who ever came from England to this country, was one of the most difficult ever put down in any field. The sand was very wet, and for that reason was very 'quick,' boiling upward with terrific force as pressures were placed upon it. Many schemes were devised to overcome the various obstacles, and finally, after a long time and the expenditure of a great amount of money, the desired object was achieved. The original shaft at the Steel Corporation's Aragon mine at Norway, on the Menominee range, was another tough proposition that was conquered only by perseverance and much capital. The Steel Corporation's Chapin mine at Iron Mountain, the biggest of the Menominee range producers, has troublesome ground also, and in sinking one shaft it was necessary to freeze the earth in its vicinity by a foreign patented process. This plan worked all right, but it has long since given way to the caisson and concrete system, in which the Foundation Co. has a monopoly in the Lake Superior iron region. A dozen or more concrete shafts have been put down the past few years in ground difficult of penetration in the ordinary way, because of a quicksand overburden."—GEO. E. EDWARDS, *Mining World*, March 11, 1911. p. 535. (A. R.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

MINING AND METALLURGICAL ABSTRACTS. Vol. II., 1910. London: International Institute of Technical Bibliography. (57-58, Chancery-lane, E.C.)

These abstracts form a section of the work of the above institution, and are published monthly. Other sections cover mechanical, electrical, and civil engineering, applied chemistry etc., and the whole forms a valuable index to current technical science. Each item gives the full title and name of author, title and date of publication, and a brief descriptive *résumé* of the article where it is not sufficiently described by the title. In the case of books the name of publisher, size and price are also stated. To non members of the institute an annual subscription of 24s. entitles to one section, and for 80s. the total edition may be obtained. Much excellent work has already been done in Germany by this institute, and an examination of the abstracts leads us to hope that the extension of its activities to England may be justified by the support which it deserves. Great care and infinite pains are evidently expended in rendering the abstracts what they purport to be—an epitome of current mining and metallurgical literature, and it is scarcely necessary in these days to dilate upon the advantages which such a work presents to the engineer and the student.—*Colliery Guardian*, Mar. 3, 1911. (A. R.)

Anschutz Gyro Compass. The History, Description, Theory and Practice. 8vo., *H. Rees.*

Barnard, J. E. Practical Photo-Micrography. Illustrated. 8vo., pp. 334. *E. Arnold.* Net 15s.

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Dunn, E. J. Pebbles. 76 Plates 8vo, pp. 122. *Robertson* (Melbourne.)

Deinhardt-Schloman. Technical Dictionaries. Vol. II. Metallurgy of Iron. Compiled by M. Venator and C. Ross. 12mo. *Constable.* Leather, Net 14s.

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Turner, Dawson. Radium. Its Physics and Therapeutics. 12mo, pp. 86. *Bailliere* Net 5s.

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Wiley, H. W. Foods and their Adulteration Roy 8vo, pp. 654. *Churchill.* Net 21s

Abstracts of Patent Applications.

(C.) 71/10. E. J. Way (1), A. J. Arbuckle (2), Improvements in apparatus for the treatment of crushed ore products. 25.2.10

This refers to an apparatus for the continuous treatment of ore pulp by agitation with cyanide solution. A rectangular tank is divided up into squares by double partitions, after the manner of a zinc extractor box; these compartments have an inverted pyramid beneath them and are provided with an air lift system. The pulp may be made to travel all compartments successively, or through a single row only as desired.

(C.) 92/10. Charles Edward Meyer. Improvements in and relating to the treatment of zinc-gold slime and in apparatus for use in connection therewith. 10.3.10.

This application refers to an apparatus for the treatment of zinc-gold slime by oxidation with steam and subsequent solution of base metals by means of carbonate of ammonia followed by acetic acid.

(C.) 94/10. Edward John Way (1), Alexander John Arbuckle (2). Improvements in vats or vessels for the treatment of ores. 11.3.10.

This patent application refers to improvements in vats for the treatment of ores, the principal features of which are the means and apparatus employed for circulating and diverting the pulp into the same vat, or partly into the same vat and an adjoining vat.

The apparatus and its working is fully described in the claims, and the main features are a device forming a chamber round the upper end of the circulating means, a launder in communication with said chamber, the means for closing or partially closing said chamber to the launder, and an intake of pulp and air into the main elevating column of a special design.

(C.) 252/10. James Edward Thomas. Improvements relating to feeding means for tube mills and like grinding apparatus. 1.6.10.

The above application relates to means for feeding pebbles or rock to tube mills.

A previous specification by the same applicant (No. 23/10) is referred to in conjunction with the present application, which describes an apparatus consisting of a hopper, with chute, to which is suitably attached a fluted roller, to act as a feeder to the previously described hopper and chute applied to a spiral feeder. The fluted roller is arranged with pins attached working in slots made in the chute to prevent jamming of rock or pebbles and a hinged upper portion of chute is shown to allow large rock to pass. The fluted roller is intermittently driven, similarly to a battery feeder of the same design and the motive force derived from the mill itself.

(C.) 283/10. Herman Stadler. Improvements in pulp classifying, concentrating and dewatering apparatus. 14.6.10.

This application relates to an improved form of hydraulic classifier which provides means for departing from the usually accepted practice in two particulars. The first consists of imparting a rotary motion to the mass within the classifier by means of a distributor, placed at the bottom of the feed pipe, which is fitted with tangential vanes. The second particular consists in increasing the velocity of the upward current by adopting an inverted truncated cone shape for the upper part of the classifier.

The advantages claimed for this classifier are that it provides more favourable conditions for separating pulp into the required grades.

(C.) 292/10. Robert James Galt. Improvements relating to shaking tables for ore treatment. 21.6.10.

This application is in respect of a method of operating a series of shaking tables from one driving eccentric or crank by a rod passing along the series of tables to each of which it is connected by a detachable clip, so as to allow any table to be stopped.

(C.) 306/10. James Lester Clark. Improvements in pipes and launders suitable for conveying abrasive or corrosive material. 30.6.10.

This application has reference to pipes and launders for conveying abrasive or corrosive materials such as sand pulp, etc. This particular application is for the use of solidified wood pulp as the main body of the pipe or launder and held together by an agglutinant. The pipes or launders may be reinforced with wire or furnished with a metal casing.

(C.) 434/10. Arthur Martin Ludwig Dammrich (1), Charles Hansen (2). Improvements in means for conveying and distributing water or other liquids in mines and the like. 3.9.10.

This invention relates to means for conveying and distributing water or other liquids in mines or any other place where the conditions met with are more or less similar to those in underground workings.

The special object that the inventors have had in view is to lay out or provide a water service in a mine, or any other place where the conditions are analogous, in such a way that the pipe system shall not be subjected to excessive hydraulic pressure whatever may be the vertical distance between the point of supply and the lowest point of the distributing system.

It is claimed that the improved system is applicable for conveying and distributing water or other liquids to any depth without exceeding a safe pressure, such pressure, if so desired, not to exceed, say, 50 lb. to the square inch or thereabouts.

The invention is fully described in the specification accompanying this application. The upper end of the main supply pipe passing down the shaft is fitted with a funnel, and water is supplied by a stand pipe and tap by means of this funnel. Briefly stated, the method adopted for preventing excessive hydraulic pressure in the pipe system is the formation in the main supply pipe of a single coil opposite each level, from the lower ends of each of which coils a branch pipe is taken to supply a tank in each level. As long as the capacity of the main supply pipe, and of the overflow pipes fitted to the supply tanks in the levels is greater than that of the stand pipe on the surface, it will be obvious that the hydraulic pressure in the section of the main supply pipe between any two levels will never be greater than that due to the vertical head between these levels.

The branch pipes are fitted with ball cocks for shutting off the water when a certain level is reached in the tanks, and the tanks are fitted with floats and electric contacts for signalling to the surface when they are empty.

(C.) 435/10. Arthur Martin Ludwig Dammrich (1), Charles Hansen (2). Improvements in means for conveying and distributing water or other liquids in mines and the like. 3.9.10.

This invention relates to means for conveying and distributing water or other liquids in mines or any other place where the conditions met with are more or less similar to those in underground workings.

The special object that the inventors have had in view is to lay out or provide a water service in a mine, or any other place where the conditions are analogous, in such a way that the pipe system shall not be subjected to excessive hydraulic pressure whatever may be the vertical distance between the points of supply and the lowest point of the distributing system.

It is claimed that the improved system is applicable for conveying and distributing water or other liquids to any depth without exceeding a safe pressure, such pressure, if so desired, not to exceed, say, 50 lb. to the square inch or thereabouts.

The invention is fully described in specifications accompany this application.

Briefly stated, it consists of supply tanks in each level fitted with a float, an electric contact for signalling when the tank is empty, a tap for drawing off the water as required, and an overflow pipe near the top of the tank to act as an air vent and overflow outlet.

The main supply pipe is taken from the surface to the bottom of the first tank, and from a point in this tank, below the level of the overflow pipe, a pipe is taken to the bottom of the second tank which acts as the suppiy pipe to that tank. Similarly the water is taken from level to level throughout the mine. It will be obvious therefore that as long as the capacity of the overflow pipes fitted to the tanks is greater than that of the main supply pipe, the hydraulic pressure in any section will not be greater than that due to the vertical head between the tanks in the levels above and below the section in question.

(C.) 87/11. James Drage. Improvements in filtering machines. 10 2.11.

This refers to improvements in filtering machines of the drum type. Three or four are provided in series, worked continuously, the slime being removed at a certain point on the drum and re-pulped with wash solution or water before passing to the succeeding drum.

(C.) 101/11. Andrew Frederick Crosse. Improvements in apparatus for the fine grinding of ores and other materials. 16.2.11.

The invention consists of a portable cylinder lined the same as tube mills and a muller arranged inside which is mechanically moved reciprocating and parallel to the axis of the cylinder which latter has to rotate and is claimed to grind between its lining and the muller the ore or other material to a very fine state.

(C.) 114/11. Sydney Russell Barrett. Improvements in and relating to pipe joints. 22.2.11.

This invention relates to improvements in and relating to pipe joints, and has for its object to provide means enabling a sound fluid tight joint to be made between ordinry screwed pipes and socket fittings for water, gas or steam, without the necessity of specially preparing either the ends of the pipes or the sockets.

This invention relates particularly to the socket or union type of joints in which an internal ferrule is provided with a projecting collar between the faces of which and the ends of the pipes to be jointed washers of suitable material are placed.

The improvement outlined in the specification accompanying this application consists in having the sides or faces of the collar machined perfectly smooth, so that, when the pipes are screwed home into the socket or union and the rough unmachined ends of the pipes bed into the soft washers, these washers as they only bear upon the smoothly faced sides of the collars, shall rotate with the pipes, and, instead of being cut to bits, shall be gradually pressed outwards until the material of which the washers are composed flows into the threads of the socket or union, and so forms a secure joint.

(C.) 119/11. John Berry. Improvements pertaining to winding and hauling engines. 24 2.11.

This application is for a device intended to show the slipping or surging of a winding rope on a conical drum hoist, in such a manner that the driver will be warned of any dangerous position of the rope when hoisting or lowering.

It is operated by means of levers attached to fine wires which are in turn connected to an indicator on a dial, in such a manner that in the event of the rope surging or winding unevenly, the driver is immediately warned.

(C.) 120/11. Hermann Wolf. Improvements in spraying matter in suspension and in spraying nozzles therefor. 24.2.11.

This patent refers to an improved spraying nozzle in which the spraying matter is supplied tangentially to its axis and provided with the means of observing the work through the nozzle or nozzles by using glass or other sufficient transparent material.

(C.) 122/11. Henry Squarebrig Mackay. Process and apparatus for extracting metals from their ores. 24.2.11.

This specification contains 27 claims, and is essentially applicable to the extraction of copper from its ores by dissolving the copper in an air-lift vat by means of an electrolysed solution of common salt, precipitating the copper electrolytically with iron anodes, and using the residual ferric chloride solution to dissolve more copper.

(C.) 130/11. Carl Nitschke. Telescopic lifting gear. 27.2.11.

This application has reference to lifting gear consisting of telescopic tubes extended and drawn together by means of a steel band inside the tube.

(C.) 138/11. Peter Mommertz (1), Gewerkschaft Deutscher Kaiser (2). Improvements in linings for pipes. 3.3.11.

This invention relates to the lining of pipes with porcelain, stone ware, glass, hard steel, cast iron, particularly in the form of case hardened castings, etc, the object of the inventors being to reduce to a minimum the wear of pipes exposed to the abrasive action of materials containing such sharp constituents as sand, gravel, chippings of stone, ash, etc.

(C.) 139/11. Peter Mommertz (1), Gewerkschaft Deutscher Kaiser (2). Improvements in linings for disintegrating machines. 3.3.11.

This invention relates to the lining of disintegrating machines, such as disintegrators, stamp mills, centrifugal mills, scouring mills, pendulum mills etc, with such materials as porcelain, stone ware, glass, hard steel, cast iron, particularly in the form of case hardened castings, etc, the object of the inventors being to reduce to a minimum the abrasive action of the substances being operated upon by any of the machines above detailed.

(C.) 143/11. Emil Schranz. Improvements in jigging machines for separating minerals. 7.3.11.

This device consists of a jigging machine with a vessel arranged underneath to catch minerals that may fall through the sieve, in the event of the latter becoming damaged

(C.) 144/11. Oscar Horace Schuller. Improvements in vibratory sifting machines. 7.3.11.

This device consists of a vibratory sifting machine mounted on rollers on an incline track with irregularities for the purpose of giving the screen a rising and falling motion as well as a shaking motion in the plane of the track.

(C.) 153 11. Hermann Georg Spengel. Improvements in conveying apparatus. 10.3.11.

This application covers apparatus for conveying material continuously and is especially adapted for overcoming elevations. It consists of a conveyer with self levelling buckets. The improvements claimed are, firstly, the method of attaching and detaching the conveying buckets from the driving chain, secondly, a tooth chain for driving the buckets, and thirdly, a peculiar method of sectionalising the system.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 217/11. John Burdon (1), William Burdon (2) and Matthew Murray Burdon (3). An improved apparatus for producing oil gas. 13.4.11.

(C.) 218/11. Fritz Mueller (1) and Hermann Strothoff (2). Improvements in and relating to grate stokers. 13 4.11.

(P.) 219/11. Harry August Stockman. An improved concrete reinforcement and apparatus for producing the same. 15.4 11.

(P.) 222/11. Edmund Paul Millington Jerrard (1) and Paul Millington Jerrard (2). Improvements in clutches or similar devices for the transmission of power. 18.4.11.

(P.) 224/11. Thomas Graham Martyn. Improvements in or relating to apparatus for grading, sorting and classifying granular matter. 20.4 11.

(P.) 225/11. Xavier du Homme de Ste Croix (1) and Nicolas Emmanuel Bertier (2). Improvements in means for automatically indicating or signalling the approach of tram cars and like vehicles. 20.4.11.

(C.) 227/11. Arthur Francis Berry. Improvements in or relating to electrical transmission and distribution systems. 21.4.11.

(C.) 228/11. John Lysaght, Ltd. (1) and Walter Robert Palmer (2). Improvements in and relating to skylights, sidelights, louvred ventilators and the like in corrugated or plain sheet metal. 2.4.11.

(C.) 229/11. Emil Ullman. Improvements in means for opening and closing the discharge flaps or doors of self-discharging trucks, cars and like vehicles. 24 4.11.

(P.) 231/11. Robert Kyd Coull. An improved method of preventing cattle or other animals from straying on to railway lines at level crossings or other openings. 24.4.11

(P.) 233/11. Thomas Mason Howard. A new and improved means for jointing together the ends of leather belts. 25 4.11.

(P.) 234/11. John William Petrie (1), James Watson. Improvements appertaining to the heads and the stems of the stamps of stamp mills. 26 4 11.

(P.) 335/11. Hans Charles Behr. Improvements in driving pulleys for stamp batteries. 26.4.11.

(P.) 236/11. Hans Charles Behr. Pneumatic percussive apparatus. 26 4.11.

(C.) 237/11. Charles Washington Merrill. Process for precipitating and recovering valuable materials from solutions, and particularly metals from hydrometallurgical solutions. 26.4.11.

(C.) 238/11. Charles Washington Merrill. Process for treating a semi-solid material in the containers of filter presses, and for removing the same therefrom. 26.4.11.

(C.) 239/11 Wilfred Watkins Pitchford. Apparatus and the method of using the same, for titrating solutions of arsenical and other active substances. 27.4.11.

(P.) 240/11. Hans Charles Behr. Improvements in the construction of stamp batteries. 27.4.11.

(C.) 241/11. Edgar Arthur Ashcroft. Improvements in or relating to the treatment of sulphide ores and products. 28.4.11.

(C.) 242/11. Anders Ponten. Centrifugal sluicing apparatus. 28.4.11.

(C.) 243/11. Felix Gain. Improvements in ventilating devices. 29.4.11.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

AIMETTI, S. C. B., *l/o* Cleveland; Transvaal Gold Mining Estates, Ltd., Pilgrim's Rest.

BASKETT, E. G., *l/o* New Zealand; *c/o* J. R. Thurlow, Esq., New Modderfontein G. M. Co., Benoni.

BELL, K. S., *l/o* Johannesburg; The Rhodesian Exploration and Dev. Co., P. O. Box 213, Bulawayo, Rhodesia.

BIDLAKE, A. E., *l/o* Johannesburg; P. O. Box 75, Boksburg.

DAVIES, C. N., *l/o* Johannesburg; 58 Lombard Street, London, E.C.

GILBERT, T. W. *l/o* Cleveland; Rand Collieries, Ltd., P. O. Box 13, Brakpan.

GRIFFIN, N. H., *l/o* Bulawayo; Oakdene, Cowden, Kent, England.

HOLDEN, W. C., *l/o* Randfontein; 14, Radnor Park Road, Folkestone, England.

KRIEKHAUS, K., *l/o* Ketahoen Mining Co., Salida Mining Co.; Salida, Tambang, via Padang, Sumatra.

MINSCH, W. C., *l/o* Colorado; Cortez Associated Mines, Jacala, Hidalgo, Mexico.

QUINCE, W. A., *l/o* Knights: P. O. Box 47, Germiston.

ROBERTS, E., *l/o* Randfontein; Witwatersrand G. M. Co., P. O. Box 1, Knights.

ROSS, A. M., *l/o* Lawlers; Mitchells Creek G. M., Wellington, New South Wales, Australia.

SCHOCH, E. R., *l/o* Boksburg; The Rooiberg Minerals Dev. Co., P. O. Rooiberg, via Warmbaths.

SHORT, A., *l/o* Fordsburg; French Bobs Mine, P. O. Louws Creek, near Barberton.

SCHWABE, S. B., *l/o* Fordsburg; *c/o* Standard Bank of South Africa, Bulawayo, Rhodesia.

STACEY, D. W., *to c/o* B. Smallie, Esq., Private Bag, Bulawayo.

WESTON, E. M., *l/o* Johannesburg; P. O. Box 64, Bulawayo, Rhodesia.

WISDOM, G. E., *l/o* Selukwe; The Hay Mine, Kimberley Reefs, Rhodesia.

THE JOURNAL

OF THE

Chemical, Metallurgical and Mining Society

OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications

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Vol. XI. JUNE, 1911. No. 12.

Proceedings
AT
Annual General Meeting, June 17, 1911.

The Annual General Meeting of the Society was held in the Lecture Theatre of the South African School of Mines, on Saturday evening, June 17th, Dr. James Moir (President) in the chair. There were also present :—

49 Members: Messrs. W. R. Dowling, R. Allen, K. L. Graham, Tom Johnson, E. J. Laschinger, J. E. Thomas, Prof. J. A. Wilkinson, A. Whitby, James Littlejohn, W. A. Caldecott, R. G. Bevington. A. McA. Johnston (Members of Council), S. Beaton, W Broom, M. H. Coombe, A. J. Cotter, W. M. Coulter, J. M. Dixon, A. L. Edwards, R. Gascoyne, James Gray, B. C. Gullachsen, J. H. Harris, J. M. L. Henry, A. Heymann, C. B. Hilliard, A. J. Johnson, H. Lea, L. Marks, T. Graham Martyn, P. T. Morrisby, M. T. Murray, H. S. Potter, E. Roberts, D. Robertson, O. D. Ross, F. W. Rush, A. H. Scarf, S. Shlom, J. Simpson, B. C. T. Solly, A. R. Stacpoole J. A. Taylor, A. Thomas, O. Tonnesen, C. Toombs, J. T. Triggs, and A. Wilkinson.

14 Associates and Students: A. R. Adams, C. J. Crocker, C. A. Damant, M. Dennehy, H. J. Filmer, P. H. Grumitt, B. W. Holman, A. King, G. F. Matthews, F. J. Pooler, H. Stadler, D. J. Thomas, J. Thorlund, and P. A. Tucker.

14 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as printed in the May *Journal*, were confirmed.

NEW MEMBERS.

Messrs. H. A. White and E. J. Laschinger were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that the candidates for membership had been unanimously elected, as follows :—

ALLEN, ARTHUR WATTS, M.A., M.I.M.M., Cuñapirú, *via* Paso del Cerro, Departmento de Rivera, Uraguay, S. America. Metallurgist.

THOMPSON, R. REGINALD, A.I.M.M., M.I.M.E., Portland House, Jesmond Road, Newcastle-on-Tyne.

WARTENWEILER, FRED. A.I.M.M., Messrs. H. Eckstein & Co., P. O. Box 2476, Johannesburg. Metallurgist. *(Transfer from Associate Roll.)*

WUNSCH, ARTHUR FREDERICK WILHELM, Lebong Soelit, Sumatra, Dutch East Indies. Mining Engineering Student.

THE ANNUAL REPORT OF THE COUNCIL.

The Secretary read the Annual Report of the Council, as follows :—

It is with a considerable feeling of satisfaction that your Council brings forward its Annual Report on the scientific and other activities of the Society during the past year.

The financial statement accompanying this Report reveals a very satisfactory position. The revenue, exceptionally high last year, is again practically maintained this year. It will be noticed that for the first time in its history, the Society is enabled to show amongst its receipts a small amount for interest on funds which have been placed on fixed deposit. The expenditure, including depreciation, with the exception of the *Journal*, shows a decrease about £75. The *Journal*, however, despite an increased revenue from advertisements and subscriptions, shows an increased cost of about £310. This is mainly due to an increase of 188 pages, 50 of which were pages of abstracts. Some portion of the increased cost is undoubtedly due to the greater use, by members, of diagrams and photographs for illustrating their contributions (there being an increase of 67 illustrations over the previous year's *Journals*) and to the large amount of tabular matter printed during the past year. It will be seen from the foregoing that the Council, through its Editorial and Publications Committee, has done a great deal towards giving members in the abstract columns, particularly those who have not the opportunity or advantage of consulting the technical literature of the day, the most useful articles on the methods and latest practice in chemistry, metallurgy and mining both here and elsewhere.

The largely increased cost of the *Journal* and the practice, inaugurated last year, of writing off the books over £200 in depreciation of the value of the stock of Transactions and office furniture, makes the statement show an apparent loss on the year's work of £195 6s. This loss is merely a book entry as the stock of transactions remains a large one, and if it became necessary at any time to reprint any *one* of the past volumes, it would entail a cost considerably more than £530, at which sum the whole of the eleven volumes issued to date now stand in the books.

The Balance Sheet does not call for any special comment except that it will be seen that our liabilities are amply covered by our cash assets. In addition to the Bank Balance of £180 9s. 9d., there is £400 on Fixed Deposit with the African Banking Corporation, and £300 in the Post Office Savings Bank, together with the accrued interest. The amount due by Sundry Debtors is principally for advertisements, which will be collected during the current month.

During the year 90 Members, 78 Associates and 42 Students were elected and admitted, 8 of the new Members having been Associates transferred to Membership, whilst 6 Students were transferred to the Associate roll.

The resignations of 15 Members, 5 Associates and 1 Student have been accepted with regret, and the loss by death has been 8 Members and 4 Associates, as follows :—

Messrs. J. S. Fisher, A. H. Hartley, Dr. M. Klein, D. Murdoch, W. D. Morton, F. B. Ogle, C. E. Rusden, J. R. Streeter (Members), and D. W. Greig and C. C. Marshall (Associates). Whilst this report was being written news came to hand of the regrettable death of two Associates, Mr. Horace Weldon, lately Chief Inspector of Mines of the Transvaal, and Mr. H. G. Roake, who was killed on the 14th inst. by an accident on the Geldenhuis Deep Mine.

As adumbrated in the last Annual Report the case of those members whose subscriptions are in arrear for two years and more, has been the subject of special consideration by the Finance Committee. Endeavours to collect the arrears during the past year having for the most part proved useless, it has become necessary to strike off the roll no less than 87 Members, 83 Associates and 38 Students. With regard to the latter it must be explained, that at the end of the financial year, all students have to renew their applications in accordance with the rules, and that of the 60 Students on the roll at the last annual meeting, the 38 Students mentioned above failed to comply with the rule and have consequently been struck off. It is hoped that in future years it will not be necessary to adopt such rigorous measures, but this year your Council felt that it is useless to keep a roll fictitiously inflated with the names of these defaulters and to continue to incur the expense of circulars, postages, etc. Even with the loss of these 208 names it is very satisfactory to find that the total number at present on the roll is 1,177 or 121 more than the previous highest membership of 1,056 in June, 1907. Although the roll to date shows thereby a reduction of 47 in the total membership from that returned last year, the Society is now in a very sound position.

A list of members is now in the printers' hands and will be issued shortly.

The usual tables of membership are given herewith.

During the past year there were held twelve ordinary general meetings, and one extra general meeting to cope with the amount of discussion which had accumulated.

The following is a list of the papers submitted during the year :—

July, 1910.—The President's (Dr. J. Moir) Inaugural Address.
"Notes on Precipitation," by Mr. F. D. Phillips.

August, 1910. — "The Efficiency of Labour Underground," by Mr. T. Johnson.
"Ventilation and Health Conditions of the Mines of the Witwatersrand, with Special Reference to the Ventilation System of the East Rand Proprietary Mines, Ltd.," by Mr. S. Penlerick.

September, 1910.—"The Distribution of Pulp for Tube Milling," by Mr. G. A. Robertson
"Notes on Battery Practice," by Mr. A. R. Stacpoole.

October, 1910.—"New Method of Lighting Fuse" (Note), by Mr. E. M. Weston.
"The Classification of Tailing Pulp prior to Cyaniding," by Mr. E. H. Johnson.
"Notes on the Occurrence and Concentration of Tin Ores," by Mr. C. F. Thomas.

November, 1910.—"Cheesa Sticks" (Note), by Mr. W. Cullen.
"The Shrinkage Method of Stoping at the Ferreira Gold Mine," by Mr. G. Hildick Smith.

December, 1910.—"A Plea for Uniform Time on the Rand" (Note), by Mr. W. M. Butler.
"Danger of Explosions in Amalgam Barrels" (Note), by Mr. W. R. Dowling.

TABLE I.

	June, 1895.	June, 1896.	June, 1897.	June, 1898.	June, 1899.	June, 1902.	June, 1903.	June, 1904.	June, 1905.	June, 1906.	June, 1907.	June, 1908.	June, 1909.	June, 1910.	June, 1911.
Hon members	5	5	5	8	14	9	13	18	18	16	17	18	21	22	20
Life members	—	1	1	1	1	1	1	2	3	4	7	7	8	9	9
Members ...	49	81	101	197	281	291	441	639	680	680	683	618	630	731	711
Associates ...	—	—	—	180	238	243	281	338	321	314	342	357	342	402	380
Students ...	—	—	—	—		—		4	8	4	7	18	20	60	57
	54	87	107	386	534	544	736	1,001	1,030	1,018	1,056	1,018	1,021	1,224	1,177

TABLE II.

DISTRIBUTION.	MEMBERS.			ASSOCIATES.	STUDENTS.	TOTAL.
	Honorary.	Life.	Ordinary.			
Transvaal (Witwatersrand)	3	2	446	190	49	690
Do. (outside districts)	2	—	42	29	2	75
Rhodesia	—	1	82	40	1	124
Cape Colony, O.R.C., and Natal	1	—	8	2	1	12
British East & West, German S.W. & Port. E. Africa ...	—	—	3	6	—	9
British Isles	10	3	47	21	2	83
Australasia ...	—	1	10	29	—	40
Dominion of Canada ...	—	—	1	1	—	2
India and Burma	—	—	1	7	—	8
Europe	—	—	4	5	—	9
United States of America ...	4	1	22	24	—	51
Central and South America	—	—	3	2	—	5
Mexico	—	—	4	8	—	12
Dutch East Indies ...	—	—	20	5	—	25
China, Japan and Korea ...	—	—	3	4	—	7
Federated Malay States ...	—	1	—	—	—	1
Addresses Wanted ..	—	—	15	7	2	24
TOTAL ...	20	9	711	380	57	1,177

"Notes on the Matte Assay," by Mr. L. J. Wilmoth.

January, 1911.—Stationary Amalgam Plates in Tube Mill Plants" (Note), by. Mr. W. R. Dowling.

"World's Glycerine Supply" (Note), by Mr. W. Cullen.

"A Grinding Machine for Zinc Cutting Tools" (Note), by Mr. H. Brazier.

"Native Food Supplies and Their Quality," by Mr. F. W. Watson.

"The Mine Dust Problem," by Dr. L. J. Aymard.

February, 1911.—Address by Sir W. Preece.

"Profit per Fathom," by Mr. R. E. Sawyer

"Practical Notes on Coal," by Mr. M. Dodd.

"Faulting Phenomena in Rand Mines, by Mr. G. H. Beatty.

March, 1911. — "Amalgamation of Gold in Banket Ore," by Mr. W. R. Dowling.

"Air Lift Agitation of Slime Pulp," by Mr. R. Allen.

April, 1911.—"The Assay of Used Plumbago Crucibles for Gold," by Mr. John Watson,

"Future Economies in Rand Reduction Plants," by Mr. C. O. Schmitt.

May, 1911. —"Some Aspects of Mine Ventilation and Earth Temperatures," by Messrs. J. Whitehouse and W. L. Wotherspoon.

June, 1911. —"The Testing of Transvaal Coals," by Mr. M. T. Murray.

In all 20 original papers were submitted to the Society, as well as an unusually numerous and varied supply of smaller notes on matters of local interest. This is slightly less than the average of recent years, yet in spite of this shrinkage in the number of contributions, our Proceedings continue to increase in bulk. The printed *Journal* indeed shows the altogether extraordinary increase of over 40% compared with last year's.

The Council is pleased to point out that the educative practice, not the least of the Society's varied activities, of holding visits and excursions has been resumed, and the following mines were visited:—City Deep, Ltd., Simmer and Jack Proprietary Mines, Ltd., Crown Mines, Ltd., and the Roodepoort United Main Reef G. M. Co., Ltd. The Council would like to express here its appreciation of the courtesy and the almost embarrassing hospitality of the authorities on these occasions. These ocular demonstrations so easy to "to be understanded of the people" are of extreme value to mining industry.

A matter in which your Council takes particular pride is that of having taken the lead in helping that valuable institution—the Seymour Memorial Library—out of the *impasse* into which it had fallen, chiefly for lack of funds. In addition to subscribing £25 per year with a promise of further financial help, we were able with the assistance of sister bodies to secure the appointment of an active and enthusiastic committee, on which the subscribing Technical Societies are now represented. The value of such a library as this is, of course dependent on its being well supplied with the latest scientific and technical literature and books, and for this purpose a considerable sum of money is necessary. The local societies, sensible of its advantages to their members and to the many unattached and independent workers and investigators in science on the Witwatersrand, voluntarily opened a subscription, to which the Chamber of Mines and the Council of Education have since given liberal donations. The Government, which it was hoped would subscribe on the pound for pound principle, despite an eloquent appeal by "our member," Dr. Macaulay, to whom our best thanks are due for his efforts, has not yet seen its way clear to assist in this direction, although making liberal grants to libraries which are not free, and whose chief function is the circulation of works of fiction. It is sincerely hoped that the next year's Estimates will include a liberal grant to the library, particularly so when the Head of the Department, which deals with "libraries," is the progressive and far seeing Minister of the Interior and Mines, the Hon. J. C. Smuts, M.L.A., who is one of our Hon. Vice-Presidents.

During the past year the Standardization of Materials Committee has, mainly through the energy of Prof. John Orr, commenced active work for the purpose of examining the quality of building materials, coal, lime, etc., in use in South Africa and endeavouring to lay down standard specifications for these articles. Your Society has been represented on the Executive by the President (Dr. J. Moir) and Prof. J. A. Wilkinson, and a chemical sub-committee, mainly constituted from your members, has been actively working on matters referred to it.

The Annual Dinner, though less well attended than usual owing to the coincidence of a railway accident at Jeppe and a somewhat inconvenient date, was a very successful affair in regard to interest and breadth of view, if less fraught than usual with the vein of *haute politique*, and more than usual with the rights and grievances of Johannesburg.

Chief among the more public activities of the Society for the year may be mentioned the presentation made to H. R. H. the Duke of Connaught on the occasion of his visit to South Africa to open the Parliament of the Union. This took the form of an illuminated address signed by the Presidents of the ten scientific and technical Societies of the Rand, and your President had the honour of being selected to hand the address to His Royal Highness at Johannesburg.

The rules for the handling of high explosives referred to in the last annual report were completed during the year. After consultation with the Association of Mine Managers, the Chamber of Mines supplied copies on linen both in English and Dutch to all Mining Companies for the purpose of being exhibited in prominent places on every mine.

The pamphlet on safety measures in mining, also referred to last year, has been further delayed owing to the absence, on parliamentary duties, of Dr. Macaulay, and to Dr. Irvine having gone to England. It is hoped, however, that it will not be long now before this very necessary publication is in the hands of every person engaged in mining work on the Witwatersrand.

Two papers, one by Mr. W. R. Dowling and the other by Mr. C. O. Schmitt, have been read conjointly at the meetings of our Society and the Institution of Mining and Metallurgy in London. It is, however, pointed out to members of both societies who may desire to avail themselves of this course, that ample time must be allowed in order to arrange for simultaneous reading, and it is necessary for papers to be submitted to your Council at least three months before they can be read in London.

The Council in December last adopted its sub-committee's report on the Standardization of Metallurgical and Mining Terms, which was printed in the *Journal* for December, 1910. The Council expects, therefore, that all contributors to the Society's Proceedings will conform to the recommendations set forth in the report in any papers and contributions submitted to the Society. The Institution of Mining and Metallurgy has referred this report to its Terms Standardization Committee which is now sitting, and the fuller and more comprehensive report which is expected, is anticipated with more than a passing interest.

The Corresponding Members of Council in Rhodesia and Oversea Countries are still assisting the Society with their hearty co-operation, and our best thanks are due to them for the help they are giving in furthering our interests throughout the mining world. During the year Mr. T. B. Stevens, a former resident on the Rand, has kindly promised to represent us in Western Australia.

The prizes offered by the Society to the Evening Class Students of the South African School of Mines (formerly known as the Transvaal University College) were competed for in November last, and although only half of the prizes offered were awarded, yet the results obtained were most encouraging. It is hoped that the results at the examinations to be held next November will be still better and that this year all the prizes offered will be gained.

Facilities for holding the Social Dinner before the Monthly Meeting at the Union Club having been withdrawn owing to lack of room, these pleasant functions had to be dropped, but when accommodation is available, they will be resumed.

The Council held 12 meetings during the year at which the average attendance was 13, showing that the Society's interests were carefully studied by your representatives during that period. It has been somewhat unfortunate, however, that so many members took their long leave this year, as their absence has been greatly felt. To the regret of his colleagues, Mr. E. H. Croghan was obliged to resign in November owing to his departure for Europe for a lengthened stay. Mr. H. Stuart Martin was elected to fill the vacancy, but business engagements compelled him to resign in May. Mr. C. B. Saner leaving for England in March, Mr. O. P. Powell was elected as alternate during his absence. Prof. Stanley and Mr. G. O. Smart left for England later in the year for health trips, and we hope to see them back again soon fully restored to health.

The Finance and Publications Committees have held regular monthly meetings which have been very well attended, and the Council is greatly indebted to the members who have acted on those and on the Editorial Committee, for their keen devotion to duty which does so much to make the work of the Society successful.

The Council wishes to express its appreciation of the services of our Secretary, Mr. Fred. Rowland, who has shown, not only great devotion to his secretarial duties, but has done much to extend the Society's usefulness to the Rand in many other directions.

The usual table of attendances at Council Meetings is given hereunder.

Dr. Jas. Moir	11	Mr. O. P. Powell[7]	2
Mr. W. R. Dowling[1]	8	,, A. Richardson	11
,, C. B. Saner[2]	7	,, G. O. Smart[8]	4
Prof. G. H. Stanley[2]	5	,, J. E. Thomas	9
Mr. J. Littlejohn	8	,, A. Whitby	6
,, F. F. Alexander	8	,, H. A. White	12
,, Robt. Allen	12	Prof. J. A. Wilkinson	8
,, E. H. Croghan[4]	3	Mr. A. F. Crosse	3
,, K. L. Graham[5]	6	,, W. Cullen	1
,, Tom Johnson	12	,, E. H. Johnson	1
,, E. J. Laschinger	7	,, R. G. Bevington	3
,, H. Stuart Martin[6]	0	,, A. McA. Johnston	5

[1] Granted 4 months' leave.

[2] ,, 4 ,,

[3] ,, 6 ,,

[4] Resigned November, 1910.

[5] Granted 2 months' leave.

[6] Elected 9th December, 1910; resigned 12th May, 1911.

[7] Elected alternate for Mr. C. B. Saner, 10th March, 1911.

[8] Granted 3 months' leave.

In conclusion, your Council would take this opportunity of again acknowledging the indebtedness of Members to the South African School of Mines and Technology for the accommodation of the Secretarial department and for the use of the Lecture Room for the monthly meeting; to Messrs. Van Hulsteyn, Feltham & Fry, the Hon. Solicitors, for legal advice and assistance; to the Hon. Auditors, Messrs. Alex. Aiken and Carter; and to your Hon. Treasurer, Mr. James Littlejohn, in whose capable hands has so long rested the financial control of the Society. Your Council would also express the acknowledgments and thanks of the Society to the local press for the space accorded and the publicity given to the Proceedings of the Society and notices of its activities.

The President: I think you will find it is a very satisfactory Report. I will now call upon the Secretary to read the Financial Statement.

The Secretary read the Financial Statement as printed on page 598.

The President: I have much pleasure in moving the adoption of the Report and Financial Statement.

Prof J. A. Wilkinson (*Member of Council*): I have very much pleasure in seconding the resolution adopting the Report and Balance Sheet which has just been read to you, and in doing so to congratulate the Society on another

THE CHEMICAL, METALLURGICAL AND MINING SOCIETY OF SOUTH AFRICA.

(INCORPORATED MAY, 1907.)

REVENUE ACCOUNT for year ending 1st June, 1911.

Dr. — Cr.

1909-10. £ s. d.	EXPENDITURE.	1910-11. £ s. d.	£ s. d.
	To Charges (including Postages, Bank		
(212 1 8	,, Charges, Insurance, etc.) ...	229 3 9	
223 2 1	,, Printing and Stationery ...	212 14 0	
30 6 8	,, Rent and Lighting	18 6 3	
447 5 0)	,, Salaries...	423 15 0	
912 15 5			883 19 0
(12 19 6	,, Depreciation on Furniture ...	24 13 8	
235 15 4)	,, Do. Transactions ...	177 8 2	
248 14 10			202 1 10
	,, Society's Journal—		
	Cost of Reporting, Printing		
(1,073 16 10	and Sundries	1,359 11 1	
384 19 5)	*less* Received for Subs. and Advts. after deducting commissions on		
688 17 5	Advts. and Collections ...	361 1 1	
+44 12 1			998 10 0
£1,894 19 9			£2,084 10 10

1909-10. £ s. d.	REVENUE.	1910-11. £ s. d.	£ s. d.
	By Subscriptions—		
(1,257 18 0	Members	1,252 13 0	
21 0 0	*Life Members*		
489 0 9	Associates	486 18 9	
21 0 0)	Students	29 2 9	
1,788 18 9			1,768 18 9
106 1 0	,, Entrance Fees		107 2 0
	,, Interest on Fixed Deposits ...		13 8 4
	,, Balance, deficit on year's working		195 6 0
£1,894 19 9			£2,084 10 10

BALANCE SHEET, as at 1st June, 1911.

Dr. — Cr.

1909-10 £ s. d.		1910-11. £ s. d.	£ s. d.
(136 6 3	To Sundry Creditors	215 11 5	
45 3 0	,, Subscriptions paid in advance ...	26 15 6	
6 0 0	,, Prize Fund	6 0 0	
14 0 0)	,, Subscriptions for Decennial Index	20 0 0	
201 9 3			268 6 11
	,, Revenue Account—		
(1,491 10 1	Balance at June 1, 1909 ...	1,536 2 2	
	Less deficit as per Revenue		
+44 12 1)	Account, June 1, 1911 ...	195 6 0	1,340 16 2
1,536 2 2			
£1,737 11 5			£1,609 3 1

1909-10. £ s. d.		1910-11. £ s. d.	£ s. d.
	By Cash at Bank—		
(187 2 11	Johannesburg	150 8 5	
239 11 10	London	30 1 4	
0 19 8)	,, *Cash in Hand*		180 9 9
427 14 5	,, Amount on Fixed Deposit and in Post Office Savings Bank, with accrued Interest		713 8 4
116 0 0	,, Office Furniture		120 0 0
707 5 0	,, Transactions, Vols. I. to XI.— Stock in hand		530 0 0
(488 14 9	,, Sundry Debtors	67 7 9	
2 2 9)	*less* Provision for Doubtful Debts ...	2 2 9	65 5 0
486 12 0			
£1,737 11 5			£1,609 3 1

We have examined the foregoing Statements of Account, compared them with the Books and Vouchers of the Society, and we find the statements to be true and correct, as shown thereby.

Sgd. ALEX. AIKEN and CARTER, } *Hon. Auditors.*

Johannesburg, 16th June, 1911.

(Signed) J. LITTLEJOHN, *Hon. Treasurer.*

(Signed) FRED. ROWLAND, *Secretary,*
Fellow Chartered Inst. Secretaries.

successful year. The finances as shown by the Statement you hold in your hands are in an extremely healthy condition, and I am of the opinion that it is the most satisfactory Balance Sheet ever presented to members of this Society. For the first time in its history your Society, as has already been mentioned, possesses a small sum which is placed with the banks on fixed deposit. In this we are following the custom set by the older societies and at the same time are assuring the future success of our own. If you will allow me, Mr. President, I desire here to express my sense of the duties Mr. Rowland has performed on behalf of the Society. It has been customary as all are aware, to vote a bonus to him as a token of our appreciation of his efforts. It is an open secret that in former years I have been somewhat opposed to this procedure, but I may say now that I am a convert and would also propose that a bonus of £50 be voted as formerly to Mr. Rowland.

Mr. W. R. Dowling (*Vice-President*): I second the motion that the bonus be given to our Secretary.

The motion was put to the meeting and agreed to, and the Report and Balance Sheet was adopted.

OFFICERS AND COUNCIL FOR 1911-12.

The President: I have to declare Mr. Saner unanimously elected President, and I am quite certain that you will agree with me that the Society will go on from strength to strength under Mr. Saner.

The Secretary read the Scrutineers' report, as follows:—

Johannesburg, 17th June, 1911.

THE PRESIDENT,

ANNUAL GENERAL MEETING OF MEMBERS.

Dear Sir,—We beg to advise you that we have examined 160 ballot papers, handed to us by the Secretary, and the following is the result of our scrutiny.

The President and Hon. Treasurer being unopposed we have to declare them returned unanimously.

PRESIDENT:

Mr. C. B. Saner.

VICE PRESIDENTS:

Mr. W. R. Dowling. Mr. E. J. Laschinger.
Mr. H. A. White.

HON. TREASURER:

Mr. James Littlejohn.

MEMBERS OF COUNCIL:

1 Mr. A. Richardson.
2 Prof. G. H. Stanley.
3 Mr. A. Whitby.
4 „ G. O. Smart.
5 „ K. L. Graham.
6 „ Tom Johnson.
7 Mr. Jas. E. Thomas.
8 Prof. J. A. Wilkinson
9 Mr. Robert Allen.
10 „ F. W. Watson.
11 „ M. H. Coombe.
12 „ J. Gray.

Yours faithfully,

Sgd. G. HILDICK SMITH,
„ A. THOMAS,
„ W. M. COULTER,
„ S. NEWTON,
Scrutineers.

Mr. Jas. Gray (*Member of Council*): I think before we go any further, we have forgotten one very important duty, and that is to thank the scrutineers for their work in regard to the election. I think we should thank them very much for their labours to-day, and for the very satisfactory result.

Mr. W. R. Dowling (*Vice-President*) seconded the vote of thanks, which was agreed to.

GENERAL BUSINESS.

The President: Under general business I have to refer to the very regrettable death of Mr. Horace Weldon, who was Commissioner of Mines for some years in this country, and who did a lot of good towards starting the agitation for good conditions underground in this country. We shall miss him very much.

NEW GRADING MACHINE.

The Secretary: I have also been asked by Messrs. Angus & Co. to show you this new grading machine, which has been designed to facilitate the grading of sand samples, which it is customary to take from time to time during reduction processes. The machine consists of two sieves the 60 and 90 mesh mounted on a receiver. The sieves are held in position by a clamping arm and winged nut. The motion is similar to that adopted in hand sieving, and begins and ends abruptly; thus the material in the sieves is thoroughly agitated and a clean residue left. The principal features of the machine are uniformity, accuracy and rapidity, while the sieves being covered, are practically dust proof. It is very carefully designed, well made and finished, with a view to eliminating any necessity for repairs and renewals. The driving pulleys are 6 in. in diameter for 1 in. belt and should be run at 100 revolutions per minute.

THE TESTING OF TRANSVAAL COALS.

By M. Thornton Murray, M.Sc. (*Member*).

At first sight, any addition to the already voluminous literature on the subject of coal testing would appear to be a work of supererogation, but the difficulties and disagreements which testing engineers, analysts and others in the Transvaal have experienced in testing coals mined in South Africa give ample excuse for a contribution on the subject. The author would point out, too, that the ground covered by this paper is quite different from that traversed by Mr. F. A. D. H. Moseley* in his excellent paper on "Coals of the Transvaal," which dealt more with the nature of the coals than with methods of testing them. The author wishes to apologise, however, for the more or less cursory nature of his remarks and for the fact that his experiments have not been so numerous or so complete as he could have wished. In extenuation it may be urged that the prominence of the subject at the present time, and the fact that the Engineering Standards Chemical Sub-Committee is at present discussing the question, made early publication advisable. It may be urged that the paper is not practical, in so far as the somewhat tedious calculations involved, and the accuracy of experiment aimed at, could never be carried out practically. While this may be perfectly true, the author wishes to point out that if the paper is to be of use to analysts and others who are anxious to standardise methods of commercial analysis, the greatest accuracy is necessary in order that the best and safest direction in which approximation may go, may be discovered. The main object of the paper, however, is to promote discussion on the subject, and if it succeeds so far the author will be satisfied.

At the outset it may be remarked that the average Transvaal, or even South African, coal with which the analyst has to deal, is, judged by the standards applied to coals in other parts of the world, an extraordinary material. Chief among its peculiarities is the abnormally high percentage of ash which it contains, the proportion of waste material in the coal varying from 10% to 20% of the whole, lower percentages being somewhat unusual, while higher proportions have often been found in coals in daily use. It is probable that this high percentage of ash materially affects the determinations of the "Heat of Combustion" of the coal, but, quite apart from that question (which will be discussed later), the larger quantity of waste material in the coal, and its comparatively uneven distribution, makes the operation of sampling difficult and liable to error. It may be that subsequent discussion will reveal sources of inaccuracy in the operation of sampling, and indicate the lines upon which standardisation of methods may run; it is scarcely within the sphere of a paper with this title to discuss that question in detail. It may be pointed out, however, that a coal high in ash percentage, and in which the waste matter occurs in patches, unevenly distributed throughout the mass, partakes of the nature of a rich ore, and in the sampling of it as much care should be exercised as in the sampling (say) of gold ore carrying many ounces to the ton. The only difference is that the waste material in the one takes the place of the valuable metal in the other. As to the methods at present employed for sampling, trucks, bins and bunkers of coal on the Witwatersrand the author wishes only to remark that any inaccuracy in this portion of the operation of sampling would fall, not on the shoulders of the analyst (whose small sample is generally "split" when his results are called into question) but upon the consumer or seller of the coal. Inaccuracies, on the other hand, in the crushing down and quartering of the laboratory sample may lead to a divergence of results among the analysts. It must be borne in mind, too, that the greatest accuracy should be aimed at in the preparation of a fine sample, since the final quantity tested often weighs one gramme, and is supposed to be representative of several tons. The question of the fineness of the sample needs little remark except to note that in this climate there is little likelihood of the sample absorbing damaging quantities of moisture from the air, and that therefore a fairly fine sample (say 80 or 60 sieve) is advisable. The author invariably uses an 80 sieve sample for testing for ash, moisture, sulphur, heat of combustion, etc. Uniformity on this point would, however, be advantageous.

Testing the Coal.

Ash.—Simply burning a weighed 80-sieve sample of the coal (say 2 grams.) in a muffle, without draught, or over a Mecker burner in a platinum dish, is the method generally adopted, and seems to give concordant results. Allowances should, of course, be made (in accurate work) for the calcination of calcium carbonate where this occurs in the ash, and for the burning of pyrite to iron oxide, $Fe_2 O_3$. The composition of the ashes of representative coals from collieries in the vicinity is at present being investigated, and the author hopes to be able to publish his results at an early date.

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* *Transvaal Institute of Mechanical Engineers*, Vols. V. and VI.

* Damour and Queneau's "Industrial Furnaces" pp. 220-223 (1906 ed.)

Moisture.—This seems to vary between 0·5 and 3·0%, but is in neither case of great importance. The relative merits of testing for moisture by repeated dryings in a steam oven until a constant weight is obtained, and by absorption with calcium chloride have been only partially investigated, and the author hopes to examine the question further.

"*Fixed Carbon.*"—The percentage of fixed carbon is of use in classifying the coal (as "bituminous" or "anthracitic"), or if the Goutal formula is to be used for estimating the heat of combustion of the coal. The method of determining the fixed carbon, which makes use of only one or two grammes of the fuel is not, in the author's opinion, the most accurate. A method which, while not introducing any greater source of inaccuracy, distributes the experimental error over a larger range, is to use 50 grams. of a 10 sieve sample of the coal, place in a Battersea "Juleff" pot, lute on the lid, leaving a small orifice for the escaping gases, and heat in a wind furnace until all flaming ceases. In this way the author has obtained more concordant results, while the larger quantity of fixed carbon remaining gives a better indication of the quality of the coke. In this connection it may be noted that the long accepted idea that there was no coking coal in the Transvaal was effectually negatived by Mr. F. W. Harbord's report on the "Iron Resources of the Transvaal."*

Sulphur.—This element is generally present in high proportions (up to 3%, although it has been found as low as 0·46%.) Although chiefly present as iron pyrite and organic matter, investigation of a number of Transvaal coals has gone to show that it sometimes occurs partially or wholly as calcium sulphate—a point to be remembered when using the bomb calorimeter or calculating the heat of combustion from the Dulong formula.†

Ultimate Analysis.—So far as the author is aware, comparatively few ultimate analyses of Transvaal coals have been made public, but, by the courtesy of Mr. A. Whitby, he is able to give one of a fairly representative Transvaal coal.

	Per cent.	
Total carbon	73·97	
Hydrogen ...	4·81	
Nitrogen ...	1·62	(by Kjeldahl's method)
Sulphur ...	0·46	
Oxygen ...	5·99	(by difference)
Moisture ...	2·40	
Ash ...	10·75	
	100·00	

The proximate analysis of the same coal gave :—

Moisture and ash, as above.

	Per cent.
Fixed carbon	59·05
Volatile matter	27·80

The coal is somewhat bituminous, and the ash and sulphur (the latter especially) are lower than the average, but, owing to the lack of time, a more representative ultimate analysis could not be obtained. The heat of combustion of this coal has been obtained experimentally, and it is purposed later on to compare this figure with the figures obtained from the analyses by calculation, using the Dulong, Mahler and Goutal formulæ.

The most important property of a coal, from a practical standpoint is undoubtedly the calorific power, or heat of combustion, which may be expressed in calories per gramme of coal burned (Continental system) or British Thermal Units (known as "B.T.U's.") or evaporative units—the latter two both making use of avoirdupois weight and the Fahrenheit thermometric scale. The last quantity is that most generally used in the Transvaal. It may be defined as :—"The number of pounds of water at boiling point, which can be evaporated into steam at the same temperature, by burning 1 lb. of the coal."

The three systems of units can be transformed one into another, thus :—

1,000 calories per gramme burned = 3968·32 B.T.U's per kilogramme burned
= 1,800 B.T.U's. per pound of fuel burned
or one small calorie per gramme of fuel burned, = 1·8 B.T.U's. per pound of fuel burned

$$\text{while } \frac{1\cdot8}{967} = \cdot00186 \text{ evaporative unit.}$$

Since the latent heat of steam on the Fahrenheit scale is 967, evaporative units can be obtained from B.T.U's. by dividing by 967, or from small calories direct, by multiplying by ·00186.

As a rough approximate method of determining the heat of combustion of a coal, the Lewis Thompson calorimeter* has the advantage of cheapness, but although certain experimenters are endeavouring to obtain a relation† between the heat of combustion, as obtained by the Lewis Thompson, and the heat of combustion as obtained by other forms of calorimeter, the results are so variable, and, at best, the calorimeter is so unscientific that the method can be left out of consideration in the present paper.

The most satisfactory calorimeter, and the one which is most generally used—both in South

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* *Government Gazette.*

† cf. Wm. Kent, Transactions of Amer. Inst. of Mining Engineers, vol. xxvii., p. 947.

‡ Both analyses from Mr. A. Whitby.

* Turner's "Practical Metallurgy," (Griffin) (1908)

† S. Lester [illegible], *Journal Society Chemical Industry*, Vol. VII., p. [illegible] gives 1% lower than Bomb, but not a constant difference.

Africa, and elsewhere, is some modification of the Berthelot calorimeter—either the Mahler-Berthelot, the Mahler-Kroeker, the Mahler-Donkin, the Mahler-Attwater, or the Mahler-Cook.

A description of this apparatus is outside the sphere of this paper, but may be obtained from any of the sources mentioned below.*

The essential parts of the apparatus, however, the methods of employing them, and their liability to error, will be discussed in some detail.

The Thermometer.—A "Beckmann" differential thermometer, reading to 0·01° C, or a set of zero thermometers similarly graduated, but giving different ranges of temperature, is most suitable. Both should be corrected against a standard thermometer.† By employing a magnifying glass, or a telescope, it is possible to read this instrument, accurately, to 0·002° C, or, with care and some practice, to 0·001° C. The accuracy of the thermometric readings is one of the most important points in connection with the instrument.

The Stirrer.—It has been contended that the motion of the stirrer in the water of the calorimeter, does not generate sufficient heat seriously to affect the thermometer. The author's experiments have shown that the motion has, at least, an effect on the rate of loss of heat by radiation —a result which has also been arrived at by another experimenter.‡ The effect, however, is not sufficiently great to consider, since the error is rendered a constant one, if the rate of stirring is kept constant by using a mechanical contrivance. This may be easily and cheaply brought about, and cannot be too strongly recommended, since, quite apart from the heat effects of changes in the rate of stirring, a momentary cessation of the motion of the stirrer, during the experiment may seriously affect the result, while a too rapid motion will often splash water over the lip of the calorimeter, and cause great loss of heat by evaporation on the outside of the calorimeter vessel.

The author uses a motor and crank attachment, giving 36 double stirs (up and down) per minute.

Insulation from Air Currents.—The insulation appears to be more perfect in the Mahler-Cook, than in any of the other varieties. The necessity of keeping a large quantity of water, as nearly as possible at the temperature of the room, in the annular water-jacket has been proved many times, while the author has found that better results were obtained when the water in this outer insulating jacket was occasionally stirred. The temperature of the water in this jacket is generally somewhat lower than the temperature in the calorimeter itself, even when the latter temperature is almost steady. This "lag" in the conduction and radiation from the one vessel to the other, is explained in Preston's "Theory of Heat."*

Method of Observing the Temperature.—The author has found that the best results were obtained by using, for this purpose, a small telescope, capable of being tilted or moved up and down, so as to follow the rise or fall of the mercury column. Readings were taken at one minute intervals, during the preliminary alteration in the temperature of the calorimeter, at 15 second intervals during the rise in temperature due to the combustion of the coal, and at one minute intervals from the time when the steady fall of temperature set in (after the maximum has been reached) until the end of the experiment.

Ignition.—Fine iron wire was used for this purpose, one inch of which weighed 0·0066 grm. $4\frac{1}{2}$ in. of this wire were used, and the whole of this, except $\frac{1}{2}$ in., twisted round the terminals, was burned, generating 34 calories. The current used for fusing this wire was of 8 amps at 20 volts.

Calibration.—Some discussion has arisen with regard to the best method of calibrating the calorimeter, or, in other words, finding its water equivalent. Calibration with some substance of known heat of combustion is apparently agreed upon, as the best method, but the best substance to use appears difficult to select. It is argued that naphthalene, which has hitherto been almost universally used, has a somewhat indefinite heat of combustion, and substances such as benzoic acid, salicylic acid, and cane sugar have been recommended as substitutes. So far as variation in the heats of combustion, as determined by different experimenters, goes, benzoic acid and cane sugar seem to have an advantage† (naphthalene shows a maximum difference of 90 calories) but even these show a maximum difference of 33 calories, while each has distinct drawbacks in other directions, benzoic acid being not always easily attainable, in the pure state, and cane sugar (according to the author's experience) failing to ignite, unless finely crushed and carefully dried, previous to burning. It is tentatively suggested, to isolate carbon from cane sugar, or prepare it by some other means; to heat it in a closed vessel to drive off any volatile constituents, to estimate the carbon (by

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[*] Paper by G. A. Burrell (American) *Western Chemical and Metallurgical Society*, Vol. IV., p. 6 (1908).

Fulton's "Principles of Metallurgy." (1910) p. 389, *et seq.*

Gill's "Gas and Fuel Analysis." (1906 ed.), p. 75, *et seq.*

Roberts-Austen's "Introduction to Metallurgy," and numerous other text books, pamphlets, papers, etc., on the subject.

[†] G. Neville Huntly, B.Sc. F.I.C. *Journal Society Chemical Industry*, Vol. xxxix., No. 15 (1910), p.p. 917 and 918.

[‡] H. A. Roesler, *Chemical News*, Jan. 6, 1911, p. 2.

[*] H. Preston "Theory of Heat" (1904 ed.), p. 251, *et. seq.*

[†] G. Neville Huntly, *loc. cit.*

combustion) in a portion of it, and to use this as a calibrating medium. The heat of combustion of pure carbon is taken, with a fair degree of certainty at 8,100 calories per gramme.

The instrument (a Mahler-Cook bomb calorimeter) with which the author is at present experimenting, has a water equivalent, as determined by naphthalene, of 780 calories, and by cane sugar of 778 calories (mean of two determinations $\left.\begin{matrix}778\cdot1\\777\cdot9\end{matrix}\right\}$ taking naphthalene as 9,692* and cane sugar as 3,954† calories.

Nitric Acid and Sulphuric Acid Corrections.—The nitrogen in the coal, under the high pressure and great excess of oxygen, is oxidised to nitric acid, the sulphur in the coal, if present as iron pyrite, will form sulphuric acid, or, rather, sulphur trioxide, which will combine with the water present to form sulphuric acid. These acids may be estimated by washing out the bomb with water, and titrating the washings with dilute ammonia, or sodium carbonate solution, the sulphuric acid being estimated by acidifying the solution from this titration, with hydrochloric acid, and precipitating as barium sulphate, by addition of barium chloride. Numerous errors are liable to occur here: to begin with, there is present in the bomb, before the admittance of the oxygen, one atmosphere of air, containing 79% of nitrogen, which, with a barometric pressure of 625 mm. of mercury, and a temperature of 15° C. gives 0·42 gm. of nitrogen, in a bomb of 540 c.c. capacity.

Doubts having been expressed as regards the purity of the oxygen, a sample was analysed, and showed only traces of carbon dioxide, but some 8% of nitrogen. In a bomb of 540 cc. capacity, and at a temperature of 15° C., and a pressure of 25 atmospheres, we should have, in the bomb, at the time of experiment:—

16·85 gm. of pure oxygen from cylinder
1·30 gm. of nitrogen (impurity in the oxygen)
and 0·42 gm. of nitrogen in the air in the bomb.

If the whole of this nitrogen were oxidised to nitric acid, some 1,800 calories of heat would be liberated, but, obviously, this cannot occur. However, the experiment was made of burning cane sugar in the bomb, under the usual conditions, but with great care to have the bomb free from any acids or nitrates. At the conclusion of the experiment, the products of combustion were bubbled through a dilute solution of caustic soda, and the bomb afterwards washed out with this solution. The solution was then tested for nitrates, and 0·025 gm. was obtained, which was equal to some 6 calories. In this case the amount formed was somewhat small, but it is enough to show that allowance should certainly be made for it, in calibrating, even though the calibrating substance contain no nitrogen, itself. The amount of nitric acid produced, on burning naphthalene or benzoic acid is now being investigated.

The sulphur, in the actual burning of the coal, burns to sulphur dioxide, generating 2,250 calories per gramme, while it is generally allowed for as sulphuric acid, which gives 4,388 calories per gramme. Again, when sulphur is present as iron pyrite, the combustion of the iron in the compound gives rise to 1,582 calories per gramme.*

Oxygen Pressure Used.—The oxygen pressure specified, in most text books is 25 atmospheres, but in this country, where the gas is very expensive, it has sometimes been the custom to use a less pressure. The effect of this practice on the ultimate heat of combustion obtained, has been well shown by Mr. E. A. Allcut†, at least in so far as it affects English coals. But the Transvaal coals behave somewhat differently, on account, probably, of their high percentage of ash.

So far as the author is aware the work of Vogt, Ackermann, Gredt and others‡ in investigating the "Latent Heat of Fusion of Slags," and their "Total Smelting Heat," has not been considered in connection with coal calorimetry. With a pressure of 25 atmospheres the author has discovered that the ashes of most coals are fused to a black. solid mass, particularly in the case of coals containing some 15% of ash, the mass partaking of the nature of a glass. In some Transvaal coals, however, the percentage of ash is so high that the temperature attained in the capsule is not sufficient to cause fusion of the ash, while lower pressures of oxygen almost always prevent fusion. The ash of a coal is generally either a potential slag or an already formed slag. When a slag is melted and solidifies to a glass the latent heat of fusion is not given out. This latent heat amounts to from 85—130 calories per gram. on substances likely to form the ash of coals. If we consider a coal having an ash consisting of iron silicate, plus a little alumina, we find that the latent heat is 85 calories per gram. Suppose the coal contains 15% of ash, we have in one gram. of coal tested 0·15 gram. of ash, having a latent heat of fusion of 12·75 calories.

While it is possible that this error may be avoided by careful experimentation, it is obvious that the oxygen pressure used must be almost identical, if the tests are to be checked by different experimenters.

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* Gill's "Gas and Fuel Analysis," p. [illegible]

† G. Neville Huntly *loc. cit.*

* *Trans. American Institute Mining Engineers*, [illegible] Haas and Lord, No. [illegible], pp. [illegible]

† "Experiments on a Bomb Calorimeter," *Engineering*, Dec. 2, 1910, pp. [illegible]

‡ [illegible] "Principles of Metallurgy."

Correction for Radiation.—The radiation correction appears to be one of the chief sources of error in testing coals by means of the bomb calorimeter, and, as Mr. Huntly[*] points out, great care should be exercised in calculating such correction.

Some analysts omit the radiation correction altogether, asserting that the acid and iron wire corrections counterbalance it. In certain cases this may chance to be true, but the method is to be deprecated from a scientific point of view, since the acid and iron wire corrections, when testing the same class of coal, are often nearly constant errors, while the radiation correction has been proved to be a very variable error. Others, again, after allowing the water in the calorimeter to cool through the same interval of time as was required to attain the maximum temperature, after firing the coal, add to the rise in temperature the *whole* cooling loss, while some add only half this quantity. It must, of course, be remembered that the bomb test can never be anything more than a comparative one so far as practical conditions of fuel burning go, but if the results obtained by different analysts are to agree, within fairly narrow limits, some strict method of standardisation must be enforced, and discrepancies such as the above (in the radiation correction) must not be allowed to occur. Further, if the results obtained are to be compared with calculations from formulæ, making use of a proximate or ultimate analysis of the coal, the greatest possible accuracy should be aimed at in all details of the experiment.

As pointed out by G. N. Huntly, Preston[†] and others, any methods of correcting for radiation which do not take into consideration the cooling effect, due to evaporation, and any other *constant* heating or cooling effect, are not strictly accurate. A formula which takes these into consideration, and, at the same time, gives probably the most accurate figure for the correction[‡] is that known as the Regnault-Pfaundler formula. In order to apply the formula the observed temperatures are divided into three epochs or periods known respectively as the *Initial Period* (during which the calorimeter vessel is adjusting its heat condition to that of the air, and losing heat by evaporation), the *Main or Combustion Period* (during which the heat due to the combustion of the coal is being communicated to the water in the calorimeter), and the *Final Period* (during which radiation is causing the thermometer to fall). Each of these periods should contain the same number of time intervals, and the last should not commence until the temperature has begun its steady fall.[*] The formula is as follows, and the method of application is shown below :—

$$C = NV + \frac{V_1 - V}{\theta_1 - \theta}\left(\frac{\theta_0 + \theta_n}{2} + \Sigma^{\,n-1}\theta - N\theta\right)$$

Where

C = Radiation correction in degrees Centigrade.

N = Number of Units of Time in the Combustion Period.

V = Rate of Fall, during Initial Period.

V^1 = Rate of Fall, during Final Period.

θ = Mean Temperature of Calorimeter during the Initial Period.

θ_1 = Mean Temperature of Calorimeter during Final Period.

$\theta_0 \ldots \theta_n$ = Temperature at the end of the first up to the nth time unit in the Combustion Period.

An example is given below with the full details of a test for heat of combustion of the coal the analysis of which has already been given.

A Beckmann differential thermometer was used, which was found to be, over the range used, almost exactly correct when tested against a standard fixed zero instrument. The amount of iron wire used was 4 in., giving rise to 34 calories, the nitric acid and sulphuric acid corrections giving rise to 24 calories, the amount of water used being 2250 cc., and the water equivalent was taken as 778 cc., making a total of 3,028 cc. The temperature was noted at one minute intervals during the initial period, at 15 second intervals during the combustion period (but these readings are plotted as one minute observations) and at one minute intervals in the final period.

Testing of Coal for Heat of Combustion, by the Bomb Method.—1 gm. of finely powdered (80 sieve) coal taken.

Total water equivalent of apparatus = 3028 cc.

Fine iron wire ignition, 4″ used, = 34 calories.

(See table on next page.)

i.e., 6,961 calories have been generated by the combustion of 1 gm. of the coal.

The heat of combustion of the coal is therefore
6,961 calories per gram. fuel burned,
or 12,530 B.T.U.'s per lb. of fuel burned,
or 12·955 evaporative units.

If the temperature readings be examined it will be found that the maximum temperature is reached at the fifth reading after ignition, while the sixth shows a slightly lower temperature. The author has found this to be the case in almost every one of a large number of tests on coals giving similar rises in temperature which he has conducted. The radiation correction, as

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[*] G. Neville Huntly, loc. cit.

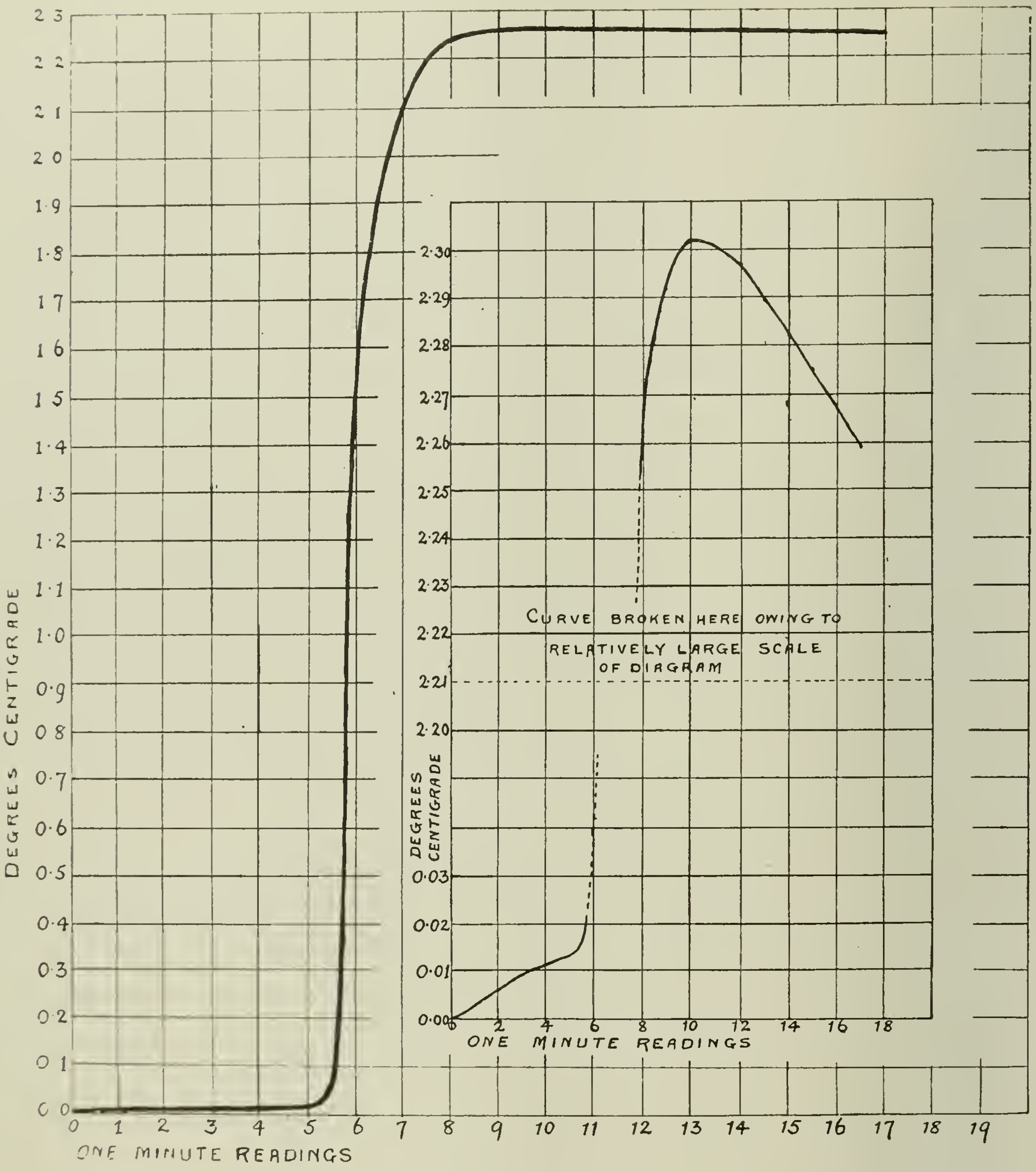
[†] Preston loc. cit. Huntly loc. cit.

[‡] E. Damour and A. L. Queneau "Industrial Furnaces," p. 204.

[*] Preston's "Theory of Heat," loc. sit.

Temperatures at one minute intervals.		Initial Period	$NV = 6 \times -·0022 = -·0132$
Initial Period.	2·242° C 2·245 2·248 2·251 2·253 $2·255 = \theta_0$	Fall = −·0130 $\therefore V = -·0022$ Mean temperature $= \theta$ = 2·249	$C = NV + \frac{V_1 - V}{\theta_1 - \theta}\left(\frac{\theta_0 + \theta_6}{2} + \Sigma^{n-1}\theta - N\theta\right)$ $\therefore C = -·0132 + \frac{·0085}{2·271}\left(11·683\right)$ $\therefore C = -·0132 + ·0437$ $\therefore C = ·0305$
Main Period.	$3·830 = \theta_1$ $4·360 = \theta_2$ $4·510 = \theta_3$ $4·536 = \theta_4$ $4·543 = \theta_5$ $4·542 = \theta_6$	Final Period Fall = ·038 $\therefore V_1 = ·0063$ and $V = -·0022$ $\therefore V_1 - V = +·0085$ Mean temperature $= \theta_1$ = 4·520 $\therefore \theta_1 - \theta = 2·271$	Rise in temperature $= \theta_6 - \theta_0 = 2·287$ + C = ·0305 2·3175 or 2·318 ∴ Corrected rise in temperature = 2·318° C.
Final Period.	4·539 4·531 4·524 4·517 4·509 4.501	Main Period $\theta_n = \theta_6 = 4·542$ $\theta_0 = 2·255$ $\frac{\theta_0 + \theta_6}{2} = 3·398$ $\Sigma^{n-1}\theta = 21·779$ $\frac{\theta_0 + \theta_6}{2} = 3·398$ 25·177 $N\theta = 6\theta = 13·494$	Iron wire correction = 34 calories Acids correction = 24 ,, Total correction = 58 ,, Total heat generated = 3,028 × 2·318 = 7,019 calories less 58 ,, 6,961 calories
$\therefore \frac{\theta_0 + \theta_6}{2} + \Sigma^{n-1}\theta - N\theta =$		11·683	

found by the Regnault-Pfaundler formula was ·031°C. Making the correction by adding the *whole* loss after the maximum has been reached gives a correction of ·0282°C, while the addition of *half* this quantity gives, of course, ·0141°C. Since the *total* water equivalent of the apparatus was 3,028 cc., a difference in temperature of 0·01°C gives 30·28 calories. The differences between the radiation corrections, as obtained by the formula, and those obtained by the rough and ready approximation methods are, therefore, equivalent respectively to 8·48 calories (= 0·0158 evaporative units) and 16·96 calories (= 0·0316 evaporative units), while neglecting the radiation correction altogether, would lead to an error of 93·57 calories, or 0·174 evaporative units. It would seem, therefore, that the first approximation gives a result not seriously erroneous. This agreement appears to be quite accidental, however, since another test gave 2·160°C as the rise as corrected by the Regnault-Pfaundler formula, and 2·193°C, corrected by the addition of the final fall through the same interval of time as was required to attain the maximum, the actual correction being in the first case ·041°C, and in the second ·073—a difference of ·032°, equal to 97 calories or 0·183 evaporative units, while the neglect of the correction would lead to an error of 0·232 evaporative units. It may be noted here that the total correction for iron wire and acids (58 calories) does not even approximately counterbalance this correction, the former being equivalent to 0·108 evaporative units, so that the practice of assuming the one correction to counterbalance the other is, from both scientific and practical points of view, quite indefensible. The nitric acid correction (using sodium carbonate solution to titrate the bomb washings) was 22·3 calories per gram. in the test previously tabulated. There was present in the bomb washings 0·0335 gram. of barium sulphate (equal to 3·35 calories), but this, on burning, in practice would give rise to sulphur dioxide, whereas it burns to sulphur trioxide in the bomb, and this combines with the water present to form sulphuric acid, so that the true correction would be :—$3·35 - \frac{3·35 \times 2250}{4388} = 1·62$ calories. This quantity is the heat generated in the bomb in excess of that necessary to burn the sulphur to sulphur dioxide, and must be subtracted (together with heat due to the burning of the



COMBUSTION CURVE OF TABULATED TEST. Large Diagram shows General Appearance of Curve of Thermometer Readings. Small Curve shows Relative Rates of Change of Temperature at Beginning and End of Combustion.

iron wire, and to the oxidation of nitrogen) from the total heat generated in the calorimeter. In this case the sulphur correction is small, but in coals containing more than 1% of sulphur it becomes important.

The heat of combustion of a coal may be calculated from formulæ. The heat of combustion of the coal analysed was calculated from the Dulong, Mahler and Goutal formulæ, and the results obtained compared with that obtained by actual experiment, with the following results :—

(*a*) *Dulong's Formula.*[*]

The best modification of this formula, for practical purposes, is

$$X = 8{,}100\,C + 29{,}000\left(H - \frac{O}{8}\right) + 2500S - 600E$$

Where,

X = Heat of combustion required in calories per gram.

H = Weight in kilogrammes of hydrogen present in 1 kg. of the coal.

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[*] Fulton's "Principles of Metallurgy."

C = Weight in kilogrammes of carbon present in 1 kg. of the coal.

S = Weight in kilogrammes of sulphur present in 1 kg. of the coal.

E = Weight in kilogrammes of moisture (as such) present in 1 kg. of the coal.

The formula which checks the bomb test most nearly is :—*

$$X = 8080C + 34{,}500\left(H - \frac{O}{8}\right) + 2250S$$

Using the complete analysis of the coal previously dealt with, this formula gives the heat of combustion of the coal as 7,439·6 calories per gram. of coal burned (= 13·84 evaporative units) or nearly 6·9% more than that obtained by experiment. It appears, then, that the Dulong formula is not successfully applicable to the particular sample of coal under consideration. Professors Haas and Lord‡ state that determinations by this formula were almost always within 2% of the heat of combustion as determined by the bomb on a large number of American coals which they tested.

(*b*) *Mahler's Formula*† is a variation of the Dulong formula, taking into consideration the amount of oxygen and nitrogen present in the coal.

$$Q^1 = \frac{8{,}100\ C + 34{,}500\ H - 3{,}000\ (O + N)}{100}$$

Since O + N (disregarding ash and moisture) = 100 − (C + H), substitution of this expression will modify the above equation to $Q^1 = 111\ C + 375\ H - 3{,}000$

where Q^1 is the required heat of combustion

C is the total carbon } disregarding ash and
H is the hydrogen } moisture.

This formula gives, on the coal under consideration, a heat of combustion of

7,014 calories per gramme
or 12,626 B.T.U.'s per lb.
or 13·05 evaporative units, or only 0·73% greater than the value obtained in the bomb calorimeter test.

(*c*) *The Goutal Formula*, which makes use of the proximate analysis of the fuel, is stated to "give results within 1% of the experimental determination, errors of 2% being quite exceptional,"‡ fails, when applied to the coal under discussion, and to another, the proximate analysis and heat of combustion of which the author has been able to obtain.

The proximate analysis of the coal, of which the complete analysis has been given was

Fixed carbon	59·05%.	Ash	10·75%.
Volatile matter	27·80%.	Moisture	2·40%.

From a table (see Damour and Queneau) a factor is obtained, from the quantity "V^1," which is equal to $\frac{V \times 100}{C + V}$, where C is the fixed carbon present, and V is the volatile matter. In this case V^1 is 32, and the factor is 97.

The Goutal formula is $Q = 82\ C + aV$, where Q is the heat of combustion required (in calories per gramme) and "*a*" is the factor obtained from C and V, as shown. This formula gives 7,539 calories as the heat of combustion, differing even more greatly from the experimental determination, than that resulting from Dulong's formula (by 8·33%, in fact). A second coal, treated by this formula, gave 7,154 calories per gramme, while the bomb test gave only 6,500 calories per gramme—a difference of 10%.

These results are somewhat discouraging, especially in view of the closeness of agreement in American coals, while the simplicity of the determination of the necessary data for the application of the Goutal formula, might have made it a useful check. The author is of the opinion (which is shared by others) that the more bituminous coals, such as those dealt with in this paper, will not give such concordant results, with formulae, as those containing higher percentages of fixed carbon. The closeness of agreement of the Mahler formula is (unless accidental) a hopeful augury for the future application of this formulae to bituminous coals. At best, however, formula are only approximately correct, while small errors in analysis may render them quite useless.

The ultimate analysis of a coal, too, is a tedious business, so that analysts would probably prefer to risk the accuracy of their calorimetric determinations, rather than expend time and care (perhaps in vain) upon ultimate analyses.

The chief points of the paper may be summarised as follows :—

1. It is essential to select the sample of coal to be tested with the greatest possible care, in the first place, so that it shall be representative of the bulk, and this care must not be relaxed during the later stages of sampling (in the laboratory) if the results obtained by different analysts are to agree within narrow limits.

2. A careful standardisation of the methods of testing with the bomb calorimeter as regards radiation correction, oxygen pressure used, and other less important details, is advisable

3. The bomb calorimeter, though an admirable instrument in itself, is liable to give discordant results in the hands of unskilled operators, and where tests are performed by different persons, more especially if allowances are not made for radiation, acid formation, etc

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* Lord and Haas, loc. cit.

† N. W. Lord and F. Haas, *Trans. Amer. Inst. [illegible] Engineers*, vol. xxvii. (1897), p. [illegible]

‡ Damour and Queneau, "Industrial Furnaces," pp. [illegible]

4. It is obvious, for commercial reasons, that some approximate method of computation will have to be employed for these corrections, but methods of approximation, before being allowed to pass as standard, should be closely scrutinised in the light of the results obtained by some such formula as the Regnault-Pfaundler. In no case, however, should these corrections be neglected, on the assumption that they balance one another.

The author is at present considering methods of approximation which will give fairly accurate results, or, at least, in which the error involved shall be as nearly constant as possible. This investigation involves the performance of a large number of tests, under differing atmospheric conditions, and using different classes of coal.

5. It is suggested that it would be to the interest of all concerned in the testing of coals, to standardize fully the method of procedure, and that further ultimate analyses of samples of representative Transvaal coals, together with the analysis of their ashes, might well be undertaken or financed by the Government, or some other interested corporation.

In conclusion, the author wishes to tender his thanks to Dr. Moir for his co-operation in the matter of nitric acid estimation, to Mr. Whitby for the ultimate and proximate analyses, and for the samples of coal, to Professor Wilkinson, and Messrs. Hendry, Moon and Estill, for their interest and assistance.

The President: This is a paper of most unusual interest, and I am sure you will thank Mr. Murray for it very much. I hope that by the time the paper is printed the author will be able to give us some data on the question of sampling, which in the opinion of many people is of great importance. Unless the sampling error can be brought down to 1% it is of no use to try and get the bomb error down to 1%. I hope all the members of the Engineering Standards Committee, mentioned in the annual report, will come forward and give us their views on this important matter. I hope myself to be able to make some contribution on it after it is printed, later in the year, when I am in a position of "greater freedom and less responsibility."

Mr. Jas. Gray, F.I.C. (*Member*): I have much pleasure in seconding the vote of thanks to the author for his extremely interesting paper, which comes at a time when the question of the correct manipulation of the Mahler bomb is occupying the minds of many of the local analysts and others interested in the matter, and in some cases may be said to be developing into a nightmare.

Not having had an opportunity of studying the author's paper, I do not feel in a position to discuss any of the points which he raised, but I hope to do so at a future meeting.

As we "experts" all hold very decided opinions on the question of coals, I anticipate that the opportunity to ventilate these views will be taken advantage of and extremely interesting and perhaps forcible discussion will result.

Prof. J. A. Wilkinson (*Member of Council*): There are two points in this paper which I think are worthy of immediate notice. The first is the question of a standard heat unit, which has been raised by Mr. Murray, with a suggestion for a definite spelling to indicate different units. As is well known different sizes of units have been used, which has made thermo-chemical literature somewhat confusing. The three in common use are known respectively as the large calorie, the small calorie, and the middle calorie. The first named represents the amount of heat required to raise 1 kgm. of water from 0° to 1° C., and would be better named the kilogram calorie. The second would represent the gram calorie, and the middle calorie, used chiefly by Prof. Ostwald, represents the amount of heat required to raise 1 gm. water from 0° to 100° C. The relation between these is usually represented thus:

1 Kal = 10 K = 1,000 cals.

In addition to these, as already mentioned in the paper, is the Commercial British Thermal Unit.

The other point, which occurred to me whilst the paper was being read is that sufficient account has not been hitherto taken of the purity of the commercial oxygen used in making the combustions. It is a well known fact that the two chief methods of preparing commercial oxygen are such, that the oxygen is liable to contamination with nitrogen in greater or less amount. It would, therefore, be useful if some further analyses could be given of the oxygen sold in cylinders in this country.

Mr. E. J. Laschinger (*Vice-President*): It is a long time since I have had to do with chemical analyses and laboratory tests on the physical properties of coal, but to the engineer the practical point of these determinations is to know how much variation in results may be expected in determinations made by different analysts, whether 5% or more.

Coal is now-a-days generally bought on a specification as to the properties of the coal supplied, and there are guarantees and penalties to ensure compliance with the standard specified. It is therefore well to know what degree of accuracy can be depended upon when a reputable analysist makes his report, and the accuracy of his figures should not be open to question by any large margin of uncertainty.

From the scientific point of view, the determination of the calorific value is most interesting as it involves one of the most difficult measurements known, that is the measurement of heat. Distance, time, forces, mechanical work and power can be very accurately measured, but heat is so elusive and so difficult to determine and has so many outlets that it is even to-day the most uncertain element in scientific accuracy. A great deal of discussion centres upon the accurate measurement of heat because of these difficulties, and I am gald to see that the subject has aroused a great deal of interest to-night, and that our local analysts have taken this matter up.

It is also fairly well known that our South African coals have peculiar properties which have caused a great deal of trouble and have been the subject of much discussion not only here but abroad.

THE AMALGAMATION OF GOLD IN BANKET ORE.

(*Read at March Meeting, 1911.*)

By W. R. Dowling, M.I.M.M. (Vice-President).

DISCUSSION.

Mr. Jas. E. Thomas (*Member of Council*): Having now had over four months experience at The Simmer Deep-Jupiter Joint Plant of the system of amalgamation brought before you by Mr. W. R. Dowling, I venture to think that a comparison of results under the old and new systems, together with a few remarks thereon might prove of interest.

The following will serve to show the position of affairs under the old system from 1st September to 31st December 1910:—

Average number of stamps running	235
,, aperature of mill screening	0·0461 in.
,, ratio of water to ore in mill	5·75 to 1
,, duty per stamp per 24 runnning hours ..	9·459 tons
,, tons crushed per day ...	2,125 tons
,, number of tube mills running (22 ft. × 5 ft. 6 in.)	7
,, number of nominal crushing units in use* ...	510·8
,, amalgamating plate area in use	5,276 sq. ft

The plates placed in front of each mortar box in the usual manner, accounted for 3,351 sq. ft. of the plate area. The tube mill circuit contained the remaining plates (shaking) having an area of 1,925 sq. ft. and a grade of 10 %. The pulp flowing over the latter had a consistency of 2 of liquid to 1 of solids.

The grading analyses of the different products were as follows:—

	+ 60	− 60 + 90	− 90	+ 200
Screen sample	41·30	14·16	44·54	—
Cyanide pulp	5·68	16·74	77·58	—
Sand original	11.45	33·93	54·62	—
Sand residue	11·00	33·50	55·50	—
Slime original	—	—	—	8·50
Slime residue	—	—	—	7·80

The extraction by amalgamation was 56·8 % of the gold in the ore sent to the mill, by cyaniding 84·67% of the gold in the cyanide pulp and the total extraction 93·38 % of the gold in the ore.

As pointed out by the author, this system has the disadvantage of limiting (1) the coarseness to which the ore could be crushed and (2) the ratio of water to ore: on account of the effect on the mill tables. It also reduced the capacity of the stamp mill as stamps had to be hung to allow of plates being dressed, during which operation crushing was liable to be neglected.

During January and February of this year the the amalgamated tables were removed from the mill some being sealed and others used for the extension of the tube mill plant, which was already under way. From 1st March to 31st May 1911 the following was the position viz:—

Average number of stamps running	220
,, aperture of mill screening	0·0685
,, ratio of water to ore in mill	5·17 to 1
,, duty per stamp per 24 running hours ...	10·834 tons
,, tons crushed per day ...	2,243
,, average number of tube mills running (22′ × 5′6″)	10
,, average number of N. C. U's. in use	581·6
,, average plate area in use	1,700 sq. ft.

The plate area consists of three stationary plates per tube mill with a grade of 18 %. All the amalgamating tables in the stamp mill were removed and replaced with short observation tables covered with 1/16 in. steel plate. The ratio of liquid to solid in the pulp flowing over the amalgamating tables in the tube mill circuit is about 1·1 to 1. During this period the grading analyses were as follows:—

	+ 60	− 60 + 90	− 90	+ 200
Screen sample	50·14	11·93	37·93	—
Cyanide pulp	4·49	15·84	79·67	—
Sand original	9·43	33·31	57·26	—
Sand residue	9·00	32·35	58·65	—
Slime original	—	—	—	3·74
Slime residue	—	—	—	3·36

(*A nominal Crushing unit is taken as a stamp with a running weight of 1,250 lb. having an actual drop of [illegible] in. 100 times per minute. See "The Development of Heavy Gravitation Stamps," by W. A. Caldecott, this *Journal* Vol. x, p. [illegible], April [illegible])

The extraction by amalgamation was 55·52% of the gold in the ore, by cyaniding 87·03% of the gold in the cyanide pulp, and the total extraction 94·23% of the gold in the ore.

Daily pannings of samples of the cyanide pulp and sand originals showed the presence of free gold in small quantities but no amalgam. Careful panning of the residue samples failed to detect any trace of free gold. As might be expected the value of the sand original samples was slightly higher than with a corresponding value of the cyanide pulp under the old system but this in no way affected the residue which show a considerably lower value due, no doubt, to the greater degree of fineness to which the pulp in the latter period was crushed. The original and residue values of the slime remained practically as before.

The consumption of mercury under the new system is from one-quarter to one-third of what it was under the old method of working. As mentioned by the author the ratio of liquid to solid in the pulp passing over the stationary amalgamating tables is of the utmost importance so that it will be found advisable to use clean water when available for diluting the pulp instead of slime contaminated water from the overflow of the tube mill classifiers, especially as the supply of clean water is much more likely to be regular.

With regard to the author's statement about the amount of gold locked up in mill plates in the ordinary way, it seems to me that a considerable quantity of free gold will always be in the tube mill circuit in the new systen, which would have previously formed part of the locked up gold, so that the benefits derived from scaling plates may not be quite as great as depicted by the author. In conclusion I wish to thank Mr. Macgregor, General Manager, The Simmer Deep Ltd., for permission to publish the above figures and remarks.

Mr. H. J. Filmer, Jr. (*Associate*) : The author is to be congratulated on his paper, not merely for the extensive review of the modern methods of classification for the increased efficiency of amalgamation, but also for the comparison of pre-war means of extraction with those of the present time.

Though his paper may not be open to any severe criticism, there are however one or two points which may be considered in the discussion. On page 416, mention is made of the metallic iron, which is produced from the wear of shoes and dies and from other sources. A fairly large proportion of metallic iron finds accommodation in the crevices of the launders, but for that which is still in circuit, no reliable means of elimination appear to be employed. With the modification of some of the present arrangements in the tube mill circuits, better amalgamation should be obtained.

The arrangements as given in diagram "E" appear to call for a little alteration. To prevent chips of pebbles and iron which pass through the screen of the pebble eliminators from scouring the plates, I suggest gravitating the effluent of the tube mills direct to the elevator. The unamalgamated pulp is then elevated to the first cone or dewaterer, the overflow going to the two safety cones and the underflow of these safety cones over the plates. The coarse material likely to cause scouring of the plates and increasing the amount of amalgam in the circuit would thus be intercepted and returned to the tube mills. In addition the screen of the pebble catcher might be of finer mesh say 36 holes to the square inch with a small stream of water playing on the catcher, thus further reducing the coarse material to be dealt with by the tube mill.

It seems impossible to entirely prevent iron from entering the tube mills, but much could be done if iron in the shape of drill ends, hammerheads, etc., were kept out of the ore by the underground men. It is really surprising what an amount of iron does find its way into the mortar boxes with the ore. Another source of iron is from the wear of cams and tappets. This could be caught under the came shafts. All iron introduced into the pulp finally reaches the amalgam room.

It seems that unnecessary expense is incurred by causing the cyanide works to deal with the large volumes of water used in milling. It should be possible to tap slimy water from the launders before entering the slime collectors and gravitate this back to the mortar boxes to be used as feed water. This slimy water would not cause inconvenience where coarse crushing obtains.

The statement that amalgamators wish to present a uniform bright service over the plate is undoubtedly true ; if rather than allow the shiftman to give a forward thrust across the plates with the whisks, an upward circular movement be given, this will be found more beneficial.

It might also be stated that the copper plates of the present day are made of too thin sheet copper, this being apparent when steaming the plates, as they undoubtedly show an inclination to buckle, notably at the sides, thereby not ensuring an even flow of the pulp down the plate.

The following contributions to the discussion on this paper were made by Members of the Institution of Mining and Metallurgy at a meeting in London on April 26, 1911, and are printed by arrangement.

Mr. G. T. Holloway: In introducing the paper, pointed out that Mr. Dowling's official position in connection with the Consolidated Gold Fields of South Africa rendered his paper directly or indirectly representative of the most up to-date practice in reduction work in that country. The methods referred to in the paper were responsible for the recovery of £20,000,000 worth of gold per annum, about two-thirds of the total output of the Rand, which, it should be remembered, accounted for somewhat over one-third of the world's output.

The introduction of modern machinery had resulted in the possibility of treating material previously unworkable, and such modifications had been made in the principal departments of the reduction plant that practically every unit in the reduction mills on the Rand had been remodelled during the last 20 years and substantially altered during the last three or four years. The sketches on pp. 4, 5, 6, and 14 showed, in the form of diagrammatic flow sheets, the present working on three or four of the largest mines, and certain suggested changes in these and others. Mr. Dowling had also given an interesting historical description, although a very brief one, of progress in milling, etc., and particularly of changes in methods of amalgamation. These, as indicated by the title, naturally took first place in his paper, so that he had almost ignored the question of cyaniding, although that also had been the subject of enormous modication, and the President would no doubt allow cyaniding to be included in the discussion in so far as it was connected with the question of amalgamation.

The cyanide process, introduced in 1890, had revolutionised gold ore treatment; while the slime treatment, introduced by Charles Butters and J. R. Williams in 1894, had led to substantial alterations in the arrangement of both amalgamating tables and cyaniding plant after a considerably simpler preliminary treatment than previously.

Perhaps the matter which was most likely to lead to discussion was the arrangement of the amalgamating tables, and it was interesting to see throughout the paper that the abandonment of many of the shaking plates in the tube mill circuit and their replacement by stationary plates, together with other alterations in the amalgamating arrangements, had resulted, not merely in a saving in plant and a better control of the extraction, but in a distinct improvement in the total recovery of gold, together with a decrease in cost. In the Simmer and Jack plant, as stated on p. 421, a better extraction was being effecting with only 18 stationary plates in the tube mill circuit, in place of the former 64 battery plates and 30 shaking tube mill plates. In that plant, the point had been reached where it was attempted only to catch the gold which was too coarse for cyaniding. Such reduction in the reduced area of plates led to a saving in mercury consumption, a reduction in the water-ratio and a substantial saving in the amount of gold locked up. The latter might be taken at 1 oz. per sq. ft. of plate area and the rearrangement of the amalgamating plant might, as stated on p. 6, result in the release of as much as £20,000 worth of gold.

The improved sand and slime treatment which was introduced in South Africa in 1904 had been evolved from the work carried out in Western Australia during the South African War, and followed up in South Africa when active work was recommenced there in 1904. It had led to the almost total abandonment of inside amalgamation in the battery and of the plates on the amalgamating tables.

Mr. Fearnside Irvine said that the paper was of considerable interest as showing the development of amalgamation on the Rand. The author in the first place described historically what happened on the Rand, in what he termed the "early days": but he quoted from J. S. Curtis, who wrote in 1894 that with a 900 mesh screen, which would be termed a 30-mesh now-a-days, recovery by amalgamation was over 60%. These results were supposed to be obtained without cyaniding: but as cyaniding was established on the Rand in 1890, it was to be presumed that Mr. Curtis was writing of a period before that date. Be that as it might, it was interesting to note that this percentage of over 60% recovery by amalgamation was not exceeded by the most modern practice in fine grinding on the Rand to day.

The whole trend of the author's paper was to show the changes made by the introduction of tube mills and fine grinding into the work of amalgamation. This he did by citing the practice at different mills.

He advocated the abandonment of the mill plate entirely, and also would eliminate any plates in the circuit between the classifiers and the cyanide plant, stating that it was unnecessary to have these provided the tube mill cones were made of proper size and were satisfactorily arranged. After discussing various methods of doing this, he put forward on the last page, in Fig. 5, a proposed arrangement for crushing and amalgamating in the tube mill.

In this diagram it would be noted that hydraulic classifiers, or spitzlutten, as they were often termed, were done away with, and reliance was placed merely upon the tube mill cone, which was fitted with a diaphragm. This cone and diaphragm the speaker took to be Caldecott's cone and diaphragm; but there did not appear to be any attempt made to introduce a rising current of water.

The overflow from this tube mill cone, or for that matter, from two or more tube mill cones, went into another cone, the underflow from which went on to the tube mill plates along with the pulp from the tube mill, the overflow from this second cone passing direct to the cyanide works without any further attempt at catching fine gold or amalgam.

The tailings from the tube mill plates ran into a pulp elevator and were returned to the first cone, so that this cone not only dealt with the current product from the mill, but with the whole of that product returned over and over again until it was fine enough to go through the overflow to the second cone, whence, as before stated, the second overflow went to the cyanide works.

In another portion of the paper, the author stated that the doing away with the mill plates and the fine pulp plates enabled a very much smaller quantity of water to be used, as the ratio of water to ore could be reduced to 4·5 to 1 without ill effects, provided the launders had sufficient grade to carry pulp of this consistency. He did not say what this grade would be, but stated that it was necessary that particular care should be taken to have the launders smooth and without break in their surface, and recommended that concrete should be used for this. Here it might be suggested that experience in the use of concrete in launders had shown that it was necessary to make this of reinforced concrete; that was to say, the launders should not be lined with plain concrete, but with concrete in which ordinary screening, say, 4 or 3 mesh screening, was embedded. This prevented the concrete from cracking, made it much stronger and ensured it against cracking even when left dry, thus making a better job.

The size of plates which the author finally recommended was certainly startlingly small, and it would be considered by many reduction men that, notwithstanding all the advantages which were claimed for the reduction of plate area in the way of saving of capital in installation, in gold held by the plates, and the reduced cost of operation, there was considerable risk run in cutting the area down to such a low figure.

He stated that the average consistency of tube mill pulp was about 55% water and 45% comminuted ore, and he furthermore said that pulp of this consistency could be carried over amalgamating plates which had a grade of 2¼ in. per ft., or 18%, with perfectly satisfactory results as far as amalgamation was concerned. From these statements it would appear that in diagram No. 5 about 4·5 tons of water would be used in the stamp mill per ton of ore crushed, and that the whole of this pulp, with this amount of water mixed, would run into the tube mill cone; that there it would be de-watered until it consisted of 1·1 water to 1 of ore, that this pulp would be ground in the tube mill: passed up to the tube mill plates: re-elevated by the pulp elevator, and passed back into the tube mill cone, which would then necessarily be filled with pulp consisting of a mixture of 4·5 to 1, + 1·1 to 1 returned from the tube mill.

This seemed to be somewhat thick for proper separation of the slimes from the sands, and as no spitzlutten were used it would seem to the speaker very possible that a considerable amount of fine gold and amalgam would be carried away by this dense water to the cyanide works.

It would be interesting to know what saving in total reduction costs this curtailment of the total amount of water in circulation was going to make, especially as the author stated that the total consumption of water would not be reduced; that was to say, he admitted that the make-up water for the reduction plant would still remain about the same as before. Probably most reduction men would prefer to use the small additional power required to circulate the extra quantity of water rather than take the risks incurred by this method.

It would also be useful to hear from the author why he considered the spitzlutten, which had done such very good work in the past and which were accepted as a portion of every reduction plant to-day, should be done away with and the tube mill cones alone relied on to satisfactorily separate sands and slimes.

As far as the elimination of the mill plates was concerned this was now accepted as the best modern practice.

It was also accepted as the best modern practice that all amalgamation should be done in a building separate from all other operation of the reduction process, so as to enable the best attention to be given to the dressing and working of the plates, and to eliminate at the same time as far as possible any chances of theft.

With those portions of the paper all up-to-date reduction men would agree; but the question of the sweeping reduction of area in the plates and the restriction of the amount of water used, required further elucidation.

It would be useful, therefore, if the author would provide replies to the following questions:

How many tons per 24 hours does he consider can be crushed by a 1250 lb. stamp through a 9-mesh screen, with the addition of 4½ tons of water to each ton of ore?

What grade of apron plate will be required to collect the discharge and lead it to the launders?

What grade of launders will be required to carry this pulp to the tube-mill cone; and, having delivered it there, will any further water be added to any of the tube-mill cones, so as to make the pulp more liquid before passing to the cyanide works?

What form of pulp elevator will be used, and what power will this elevator take to drive it, in terms of tons of tube mill pulp elevated and returned to the tube-mill cone, to carry out a practice in accordance with the Diagram No. 5?

Mr. S. J. Truscott said they must all admit this was a most important paper. There were principles contained in it which were far-reaching. Mr. Dowling spoke about the percentage obtained by amalgamation, and one would naturally take it that this percentage was with reference to the original value of the ore, but this was never actually stated in the paper. He recalled that in 1894 Mr. Curtis recorded that 60% was obtained in the early days by amalgamation, this figure being undoubtedly referred to a figure representing the original value of the ore. In this paper, however, where percentages pertaining to recent work were given, it was not stated to what values these percentages referred. He, the speaker, did not think that on the Rand such percentages necessarily referred to the original value of the ore, though this was naturally the true basis; they sometimes were given in terms of the total bullion obtained.

The essential question, however, raised by the paper was that which concerned the relative parts to be played by amalgamation and cyanidation. He thought they would agree with him that the determination of this question depended very materially upon two factors. It depended upon the factor of the ore's nature and to some extent upon the bias of those who were engaged in its treatment. With regard to the nature of the ore, it might be said that gold ores naturally fell into two classes: those ores in which the gold was free and open to amalgamation, and those in which it was so imprisoned, either by its intimate connection with the matrix or by its chemical combination, that an adequate extraction was not to be obtained by amalgamation. The ore in most of the gold mines of the world belonged to that class where the gold was free, and he thought it would not be disputed that the banket ores belonged to that class. In support of this statement he would read an extract from the paper read by Mr. Curtis, in 1894, before the Chemical and Metallurgical Society of South Africa.*

"The gold in the Rand banket is free and not coated with substances which prevent its union with mercury; it has been my experience that if the Rand sulphuret ore is pulverised sufficiently fine the gold can be obtained by amalgamation."

And again, in the same paper, when advocating a greater application of amalgamation, he said:

"The mainstay of the whole method would be, however, the cheapness and facility with which the gold, when liberated from Rand ores, can be amalgamated."

Granted, then, that the Rand banket is an ore where the gold is primarily free, it becomes of interest to see what other mines of that class are doing.

At the Alaska Treadwell they obtained from 70 to 75% of the assay value of the ore by amalgamation, the remainder of their revenue being obtained from the concentrates, which amounted to about 2% of the weight of ore.

At the Giant Mine in Rhodesia they obtained an equally large amount by amalgamation, about 75% of the ore's value, and they obtained the remainder of their revenue by treating their concentrates; this, he believed, being also obtained by amalgamation after acid treatment.

At the Homestake Mine they obtained 70 to 75% by amalgamation, and the remainder of their extraction by cyanidation.

Then at the Mysore Gold Mines they obtained 80% and more by amalgamation, and so little did their prosperity depend upon the cyanide process that at this date no mine there was treating its slimes by that process.

Those were some of the mines where the gold was free. On the other side there were those mines represented by the Waihi, where the gold was imprisoned and where it was necessary to have some such treatment as that of the cyanide process. The dependence upon that process was shown by the following extracts from a paper read in January of this year before the Australasian Institute of Mining Engineers:

"Very perfect amalgamation is not aimed at, because any fine free gold escaping the plates is easily accounted for by the cyanide plant."

And again:

"Concentration after tube-mill grinding is not usually considered good practice; but with the Waihi it is found that, providing the coarser portion is separated by the vanners, it does not matter if the very fine sulphides escape to the cyanide plant."

Was such an attitude, however, justifiable for ores like the Waihi, also the proper one to apply towards the ore upon the Rand? He should say not; but from recent practice one would consider that it was the attitude which was also being adopted there. It was mentioned in the paper under discussion that when lime was first used with the ore upon the Rand there was a pronounced improvement in the reduction by amalgamation, amounting to from 4 to 8%, for which those concerned got all due credit.

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* See this Society's Proceedings, Vol. I., Dec. 15, 1894, pp. 76–80.

In this paper, however, there was no mention of the continued application of lime for the benefit of amalgamation. It was useless to say that by the addition of lime after amalgamation, and for purposes of settlement only, the mill water was rendered sufficiently alkaline to benefit amalgamation. It was that action and property of lime which produced settlement which also was of advantage to amalgamation, and both amalgamation and settlement must follow at once after the introduction of the lime, or those benefits were not obtained. Then, again, when tube mills were first started upon the Rand it was found that if a plate were put at the discharge end, a second crop of amalgam was obtained and everybody was quite pleased and due credit was distributed. Now it was proposed in many cases to do away with any such second crop of amalgam because the cyanide plant would be able to look after that. It seemed to him, therefore, that it would be better if on the Rand there was a little more opposition to these present tendencies, and perhaps a little new blood would also be good.

Mr. E. T. McCarthy : I would cite, as bearing on this subject, the experience obtained in 1904-1906 on a different class of ore, viz., that of the Gwendoline Mine, Korea.

The mill for this mine was designed by Mr. Alfred James, based on the results of experimental work carried out in the laboratory of the mine only. As it was known that gold values had given out in the mine in depth, it was a race to erect the mill without losing a season and there was therefore no time for any experimental treatment of the ore at home without the risk of losing a year, which would have meant keeping up an expensive establishment without corresponding benefit.

The ore was pyrrhotite carrying a silicious gangue in the proportion of 20-30% of the former to 50-70% of the latter.

The lines on which the mill was erected were those of a 20 stamp mill, a tube mill and cyanide plant with filter presses. The whole plant was most expeditiously got out, erected by Mr. Marriner and run under the guidance of Mr. Holloway, and until it shut down practically ran without a hitch of any kind and successfully.

The point of interest in connection with this mill was that of amalgamation. The results of experimental laboratory work gave only poor amalgamation results, and with a 25 mesh screen this was afterwards borne out by the results on the mill plates.

We decided to put in fixed plates for experimental purposes below the tube mill, which so far as we then knew had not been tried elsewhere. The results obtained on these plates were so good that could I have foreseen them I would not have put in the cyanide plant, as with the end of the ore in sight the saving made here did not compensate for its first cost. Unfortunately the records are stored away, so I am unable to give the exact figures.

I note the tendency is now to substitute fixed for shaking plates. It would have been interesting if the author had gone into a little more detail as to the reasons for this change. I note the Luipaards Vlei Company have just decided, on the advice of their consulting engineers, to substitute fixed for the shaking plates now in use.

With regard to amalgamation with a thick pulp, my own experience in the United States some years ago on a talcose slate ore was that better amalgamation results were obtained with a thick pulp and steeply inclined plates than by the usual practice. I think the same is true in pan amalgamation ; the thicker the pulp, compatible with obtaining a distribution of the mercury through it, the better the results than with a thinner pulp (sufficiently thick to keep the mercury in suspension).

AIR LIFT AGITATION OF SLIME PULP.

(*Read at March Meeting, 1911.*)

By Robert Allen, M.A., B.Sc., M.I.M.M.
(Member of Council).

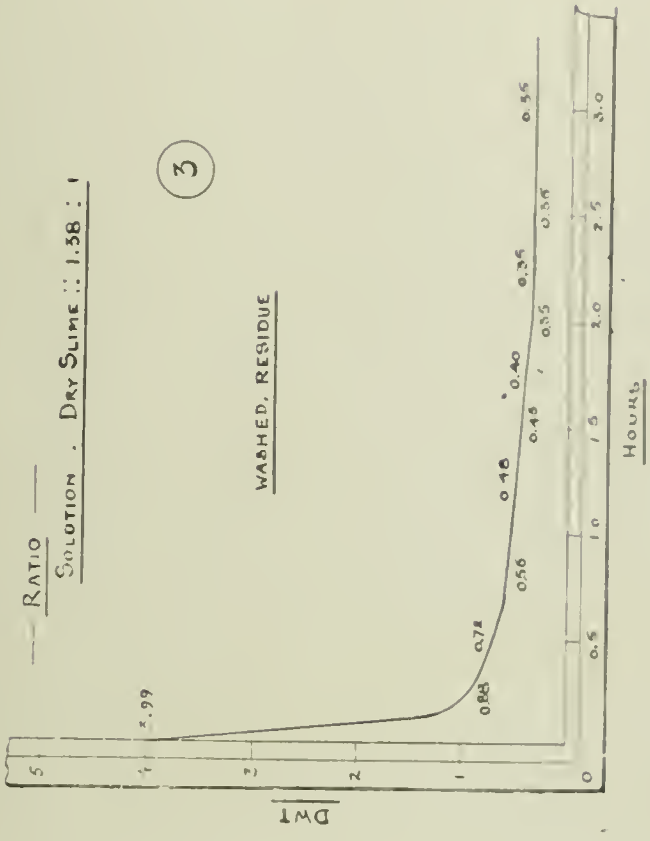
DISCUSSION.

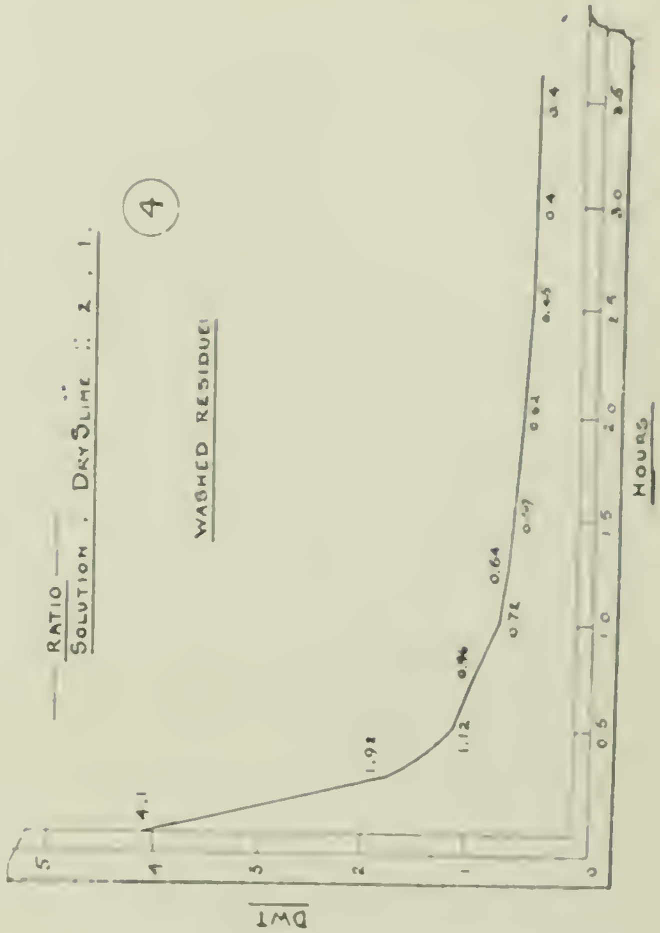
Mr. Chris. Toombs (*Member*) : Mr. Allen's paper has opened up a subject of great interest to metallurgists on the Rand, more especially now that the air lift agitation method of dissolving gold has been installed on one of our latest Rand plants, viz., the Brakpan Mines, and it is rather unfortunate that this plant has not been running sufficiently long to enable us to have a contribution as to its efficiency compared with the "decantation" process, from one of those responsible for its working.

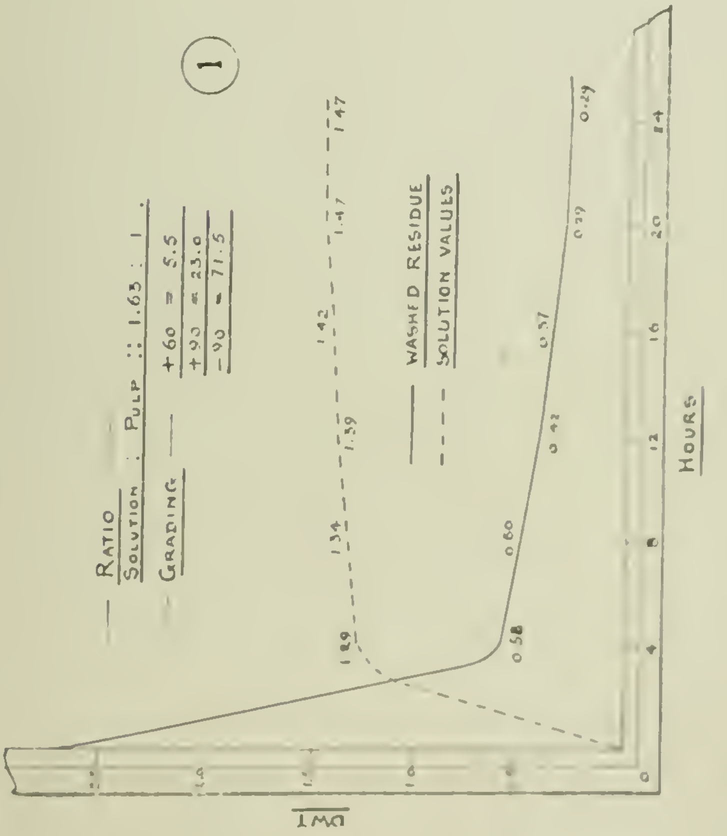
For the treatment of accumulated slime agitation by this method is particularly effective.

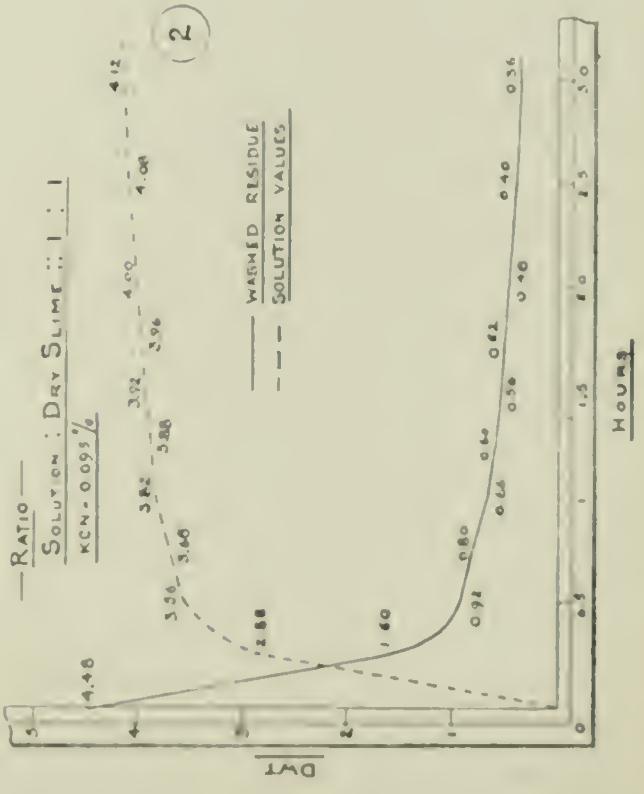
Recently reading through Dr. Caldecott's now classical paper on "Accumulated Slime Treatment," read before the Society in July, 1897, the following paragraph struck me as being prophetic :—

"It is possible that before long aeration now regularly used for the first time on a working


③
Ratio —
Solution . Dry Slime :: 1.38 : 1
Washed, Residue
3.99
0.88
0.72
0.56
0.48
0.45
0.40
0.35
0.35
0.36
0.36
DWT
0
1
2
3
4
5
0.5
1.0
1.5
2.0
2.5
3.0
Hours


④
Ratio —
Solution . Dry Slime :: 2 . 1.
Washed Residue
4.1
1.92
1.12
0.96
0.72
0.64
0.47
0.62
0.45
0.4
0.4
DWT
0
1
2
3
4
5
0.5
1.0
1.5
2.0
2.5
3.0
3.5
Hours


①
Ratio —
Solution : Pulp :: 1.63 : 1.
Grading —
+60 = 5.5
+90 = 23.0
−90 = 71.5
Washed Residue
Solution Values
1.29
1.34
1.39
1.42
1.47
1.47
0.58
0.60
0.42
0.37
0.29
0.29
DWT
0
4
8
12
16
20
24
Hours


②
Ratio —
Solution : Dry Slime :: 1 : 1
KCN - 0.095%
Washed Residue
Solution Values
4.48
1.60
0.92
0.80
0.66
0.64
0.56
0.62
0.40
0.40
0.36
2.88
3.56
3.60
3.82
3.88
3.92
3.96
4.02
4.08
4.12
DWT
0
1
2
3
4
5
.5
1
1.5
2.0
2.5
3.0
Hours

scale will serve also as a means of agitation in place of mechanically driven paddles at present employed."

In the same paper Dr. Caldecott also refers to the advantage of deep vats for aeration purposes, but he scarcely foresaw the large chimney-stack-like structures known as "Brown" or "Pachuca" vats now being used.

The pyritic portion of accumulated slime has through course of time been partially oxidized to ferrous sulphide (FeS), ferrous sulphate ($FeSO_4$) and sulphuric acid (H_2SO_4). These three compounds are those with which the metallurgist is called on to deal, so the fully oxidized compounds present need not be detailed.

Ferrous sulphate ($FeSO_4$) is really the *bête noir.* On the addition of sufficient lime to neutralize the H_2SO_4 and $FeSO_4$ the latter is precipitated as ferrous hydrate $Fe(OH)_2$. This compound has a powerful affinity for oxygen, and immediately denudes the slime pulp of this element, only a very small proportion being converted into ferric hydrate $Fe_2(OH)_6$ in doing this. Before perfect conditions for cyaniding exist it is necessary that all $Fe(OH)_2$ be converted into the innocuous $Fe_2(OH)_6$, and the pulp mass resaturated with dissolved oxygen, and for this purpose agitation by air lifting gives every particle of the pulp intimate contact with air, and that in a very short space of time. The insoluble ferrous sulphide (FeS) also slowly oxidises into $FeSO_4$, but in the presence of excess of lime and oxygen the change to $Fe_2(OH)_6$ is instantaneous. Complete conversion of $Fe(OH)_2$ to $Fe_2(OH)_6$ is very rapid if intimate contact with air is assured. Those interested in this subject and unacquainted with laboratory work can see this for themselves by dissolving a small crystal of $FeSO_4$ in water in a small flask and adding a little ammonia or lime water. A dark green precipitate of $Fe(OH)_2$ results. If the contents of the flask are swirled round without actually shaking up, considerable time will elapse before the green precipitate turns to the reddish brown $Fe_2(OH)_6$. On the other hand, if the flask is closed with the palm of the hand or a cork, and the contents agitated vigorously, renewing the air supply once or twice by removing the cork, the green precipitate changes quickly to the fully oxidised brown. This will also give a rough idea of the different effects of mechanical and air lift methods of agitation.

The affinity $Fe(OH)_2$ possesses for oxygen has been made the basis of a simple and accurate method for determining dissolved oxygen in waters by Prof. Letts, Queen's University, Belfast, which is simple in its manipulation, much simpler than Thresh's well-known method, and the results obtained by it compare very closely with oxygen determinations made by analyses of the dissolved gases. This method is worthy of note by any of our analytical members requiring information on that important point, the amount of dissolved oxygen contained in potable waters, more especially if suspected of contamination with sewage or animal excreta, or in which ærobic bacterial fermentation may be taking place.

To illustrate the efficiency of air lift agitation both as a time saver and a dissolver I have plotted the following curves from data obtained by the experimental work carried out on this method at the East Rand Proprietary Mines.

Fig. 1 shows rate of solution and extraction obtained with ordinary pulp after amalgamation.

Figs. 2, 3 and 4 show results obtained from treatment from accumulated slime by this method, with different ratios of dilution. These diagrams illustrate the rapidity with which gold contained in slime dissolves, the very small amount dissolved after the first hour's treatment probably coming from sand present with the slime to the extent of from 5% to 10%.

Treatment of Plumbago Pots.—A few details of results obtained by this method in conjunction with amalgamation, on ground plumbago pots may be of interest.

A charge containing 21·7 oz. gold and 21·8 oz. silver per ton was treated with the following result :—

	Au. Per cent.	Ag. Per cent.
Extraction by amalgamation	32·95	13·07
Extraction by KCN ...	62·9	54·82
Total extraction	95·85	67·89

Giving a residual value of 0·9 oz. gold per ton.

This is a summary of several experiments made on a laboratory scale.

On a larger scale the treatment of some 3,000 lb. of the same material gave a gold extraction of

	Per cent.
By barrel amalgamation ..	81·68
Agitation with KCN	13·41
Total extraction	95·09

The thanks of the Society and members generally are due to Mr. Allen for introducing this interesting subject, of which I am convinced we will hear a great deal more in the future.

PROFIT PER FATHOM.

(*Read at February Meeting, 1911.*)

By R. E. Sawyer, A.R.S.M., A.I.M.M
(Associate).

REPLY TO DISCUSSION.

Mr. R. E. Sawyer (*Associate*) : I have to thank Mr. Richardson for his interesting criticism of my notes on "Profit per Fathom."

I must plead guilty to a terminalogical inexactitude in saying that my figures were merely compiled as examples. They are actual figures from a real mine—the Cason. My excuse is that at the time the notes were written, I had not received permission to give any names.

He says that the distribution of compressor charges proportionately to fathomage broken is unfair ; to distribute on a tonnage basis would be worse. Take the case of two stopes, one 3 ft. 6 in., the other 4 ft. 6 in. stope width, both with good foot and hanging parting. One hole drilled in each stope with equal burdens will break nearly equal fathomages, but the tonnages will be very different.

I may say that the original investigation on which these notes are based was set on foot to determine whether certain large bodies of reef of an average value of 3·5 dwt. were being worked at a profit. The results are conclusive, and it is a fact, that in certain cases rock containing only 2·5 dwt. per ton, was being mined at a profit. In this connection it is interesting to note that in the annual report of the East Rand Proprietary Mines 3,500,000 tons classed as unpayable are valued at the very low figure of 2·2 dwt. This, as Mr. Richardson says, should be of great importance to the Rand as a gold field. My object was not to advocate any particular system, fathomage or other, but merely to determine a fair approximate method of distributing costs, and thus arrive at a definite conclusion as to the payability of certain ground. I maintain that this could not be done on a tonnage basis.

Another point on which the method throws light is the true comparison between large and small machines. I will leave out the hammer boy as we cannot get him. I have figures from several mines which show that the actual cost of stores and labour for small machines is (unlike the example in Table I.) about 10s. per fathom less than for large machines, and I understand that on the Robinson Deep large machines are being replaced by small ones. As a rule, small machines are placed in stopes with small stope widths and consequently cost per ton is high, the cost per fathom, however, being low. I maintain that if the small machines were placed in the wider stopes the cost per ton would also be lower than the large machines. This experiment is actually being tried under the author's supervision and the result to date entirely confirms his opinion. The fathomage per machine shift is only slightly lower for the the small machine while the cost is less. It must be remembered that one white man runs at least four small machines as against two large ones, thus the fathomage per white shift is far larger for the small machine, hence the reduction in cost. The efficiency of the small machine depends largely on getting in six holes per machine, three holes to a bench ; long holes are not necessary or desirable in narrow stopes. The author, however, does not recommend small machines for very large stope widths. He has seen over 100 tons broken at one blast by three holes, each 12 ft. long in a stope 14 ft width. Small machines could hardly compete with this.

Mr. Richardson thinks that the adoption of this system would lead to selective mining : I think the opposite would be the case. The profit per fathom depends largely on the contents per fathom and this can be increased by including poor reefs of the foot or hanging wall in the stope width. The author's policy of mining 2·5 dwt. rock can hardly be called "picking the eyes out of the mine."

I agree to a large extent with Mr. Tom Johnson's remarks, and thank him for correcting my arithmetic. He asks why small machines are used in narrower stopes than hammer boys? Because with a fairly flat reef they can mine narrower than hammer boys.

In conclusion, I regret that there has been so little discussion, as these notes were only written in a tentative manner to elicit information on the points dealt with.

SOME ASPECTS OF MINE VENTILATION AND TEMPERATURE ON THE WITWATERSRAND.

(Read at May Meeting, 1911.)

By J. Whitehouse, B.Sc., M.I.M.E. (*Associate*)
and W. L. Wotherspoon

DISCUSSION.

Mr. Rowland Gascoyne (*Member*) : The questions of underground temperature and mine ventilation were undoubtedly attracting increased attention as the mines grew in depth, but that of temperature seems to have received some attention almost ever since the mines attained any importance. The most remarkable feature about these observations regarding underground temperature seemed to be that every new observation indicated that the temperature gradient gradually decreased with the depth. In 1893 Mr. Hamilton Smith, when reporting upon the deep level question, seems to have taken some observations and gave the increase as 1° F. for every 82 ft. The mines at that time were of course very shallow, but the Rand Victoria borehole had then just struck the Main Reef series at a depth of 2,500 ft., and this borehole seems to have been used to arrive at the above result

Borehole results are not, of course, as reliable as those obtained from actual observations in mining operations, but they nevertheless constitute a guide. Mr. Andrew F. Crosse made some observations in 1895 down to a depth of over 1,000 ft., and calculated that the temperature increased 1° F. for every 100 ft. There seems to have been no further observations recorded until 1906, when Mr. Hugh F. Marriott, as a result of observations made to a depth of 3,916 ft., gave the result as 1° F. for every 208 ft. in depth, a remarkable result and one that attracted some attention. Still more remarkable were the results given by Mr. Lionel Phillips two years ago at a Chamber of Mines meeting of observations made at Brakpan and elsewhere, when it was stated that the increase was 1° F. for every 255 ft. in depth, so that it would seem that undoubtedly the temperature gradient flattens as the depth increases. It would be interesting to know more about the observations mentioned by the authors, as they appear to confirm, more or less, those mentioned by Mr. Lionel Phillips, in a remarkable manner, and if the date of the observations can be given it would of course add to their value.

I do not think that underground temperature conditions are anywhere else in the world so favourable to deep mining as on the Rand, but in the Lake Superior copper regions they run very close to those prevailing on the Rand. Years ago I was engaged making observations for the committee appointed by the British Association for the Advancement of Science to investigate this question of underground temperature, and as far as my observations went they seemed to indicate that the temperature gradient was inclined to be higher in rocks running comparatively flat than where they had a considerable dip. I hope that the somewhat steeper heat gradient indicated by the writers as existing at Brakpan may not be a prelude to similar conditions when at great depths the reefs may be expected to have flattened considerably on these fields. Another indication of some importance I noticed was that in a coal seam, the heat (probably due to oxidation) was several degrees higher at the face than at a depth of 4 ft., and I came to the conclusion that to obtain reliable results a borehole in a coal seam should be at least 6 ft. deep.

I wonder whether many have read that somewhat romantic geological work of Prof. Schwarz on "Causal Geology," because this question of underground temperature is mentioned, and the decreasing gradient of increase on the Rand is taken by him to show that the earth grows gradually cold in depth, so that mining may possibly be carried on to such enormous depths on the Rand without attaching any importance to this underground temperature question—but I doubt whether any engineer who has had much experience of deep mining elsewhere will be inclined to agree with the learned Professor in this respect. Still it must be admitted that with the Rand possessing the lowest temperature increase of any mining field in the world and a class of unskilled labour that ought to stand high temperatures underground, the Rand gold fields have been particularly favoured by nature, and no doubt such mining depths will be attained here as will prove impossible in other parts of the world.

It is much to be regretted that on the Witwatersrand, where boreholes have reached such great depths as at Rietfontein, Vlakfontein, and Gemsboklaagte, temperature observations were not made to compare with the borehole temperatures in other parts of the world. For instance, in Upper Silesia, at Gzuchow, a borehole was put down 7,346 ft., and gave an increase in the temperature of 1° F. for every 54 ft., whilst another borehole at Paruschowitz, also in Upper Silesia, at a depth of 6,429 ft., the increase was 1° F. for every 56 ft. Then in France, at Pont a' Mousson, where the depth was 5,104 ft., the increase was 1° F. for every 55 ft. In all these cases, however, it must not be overlooked that the borings were made in coal measures, or in the neighbourhood of coal seams, where temperatures underground are always high and liable to be influenced by the close proximity of coal seams. In borings put down to a depth of over 4,000 ft. in Western Virginia, U.S.A., the temperature increased at the rate of 1° F. for every 73 ft. Rock temperatures seem to vary in different rocks for from observations I made at a deep borehole put down under my charge near Lincoln through the lias, rhaetic, keuper, bunter and permian beds I estimated the increase at 1° F. for every 74 ft. as compared with 1° F. for every 60 ft. generally met with in the coal measures. It is perhaps when we come to consider actual temperatures experienced in deep mines that the subject becomes more interesting. Now up to the present the Victoria Reef Quartz Mine, Bendigo, is said to be the deepest gold mine in the world, having attained a depth of 4,600 ft. from the surface, and the temperature of the rock at that depth is 112° F. as compared with 83° F. to be expected on the Rand at that depth, thus showing a difference in favour of the Rand of 29° F., which means in other words, that before a similiar temperature is met with on the Rand the mines will have attained a depth of nearly 12,000 ft. The only other part of the world where the temperature gradient at all approaches that of the Rand is in the Lake Superior copper regions, where mining is being carried on successfully at a depth of a mile more or less from the surface. If we come to coal mining experiences we find that a rock temperature of 120° F. is attained at the Ste Henriette shaft, in Belgium, at a depth of 3,900 ft. as compared with a rock temperature at the Ashton Mors Colliery of 102° F. at a depth of 3,200 ft.

The task of cooling the working places of these deep warm mines so as to enable white labour to work therein is not an easy one. At the deepest gold mine in the world at the Victoria Quartz Mine, Bendigo, an overhead brattice is put in which delivers about 1,500 cub. ft. of air per minute at the face, thereby reducing the temperature of the working places to 87° F. I found, when manager of the Ashton Moss Colliery that the only feasible way to keep the working places cool was by having a separate travelling road free from side branches or other methods of escape all the way from the downcast shaft to the working face. Even then the fresh air reaching the lowest workings soon assumed a temperature of 75° F., and every 30 yards or so it travelled along the face it seemed to pick up a degree of temperature until it reached 90° F., which was about the highest temperature encountered anywhere along the working faces of the mine. It was surprising how easily the air escaped at these depths, even when no opening was apparent, and this would seem to threaten to constitute one of the greatest difficulties in efficiently ventilating the future deep mines on the Rand, more especially if the workings are left open and sand filling on an extensive scale is not resorted to. Water sprays were tried to cool the air but it became so soon vitiated and oppressively hot that the experiments were soon abandoned.

The records given by the authors as to the temperature of the atmosphere in the Village Deep Mine both before and after starting the fan are indeed very valuable, because after all it is the atmospheric temperature of the mine, rather than the rock temperature, that affects the workers for after continued exposure to cooler currents of air, rock temperatures became gradually cooled down. At the greatest depth at the Village Deep the highest recorded temperature was 80·2° F. before the starting of the fan, and at the shallowest depth the temperature recorded was 74·1° F. There are so many different conditions, local and otherwise, to be considered as affecting the atmosphere of a mine that it becomes difficult to form reliable conclusions, but the temperature of the atmosphere seems to have increased before starting the fan somewhat similar but not exactly the same as the rock temperature. Some striking reductions in temperature seemed to have occurred at the level stations as the effect of starting the fan, but along the levels and particularly where the observations were made considerable distances inbye, the difference is much less noticeable, thus showing how, as previously remarked, the air current, even if strong, soon becomes warmed up to its surroundings. Had there been recorded on the tabulated statements the amount of air circulating at the different points as measured by the anemometer before and after the fan started, much valuable information would have been gained thereby, nevertheless, the tables show how beneficial regular air currents are to the underground health conditions of a mine. The tabulated statement of the results of the underground observations, with the thermometer taken in conjunction with the plans, also show how difficult it is even under the favourable conditions existing at the Village Deep in the Main Reef Leader to ventilate the lower unconnected workings in such a manner as to materially reduce the temperature. This difficulty is one that exists in all deep mines, and I have previously referred to the special steps required to overcome it in very deep mines in order to reduce the temperature to such a level as will allow of mining operations being carried on with any degree of comfort.

The writers refer to the large quantity of moisture conveyed by the return air from the mine and with a current of 260,000 cub. ft. per minute at 8·2 gr. of moisture per cubic foot of air it is estimated to have been as high as 40,400 gallons of water per day. The writers go on to say that the removal of such a large quantity of water from the mine workings daily has naturally a tendency to make the workings very dry, but they have evidently overlooked the fact that the intake air also conveys moisture into the mine which if allowed for materially reduces the estimated loss of moisture. For instance at the Banner Mine with a current of 200,000 cub. ft. intake air at 40° F., 14,000 gallons of moisture were carried into the mine and with the return air heated to 65° F., it was estimated to carry 34,000 gallons of moisture out of the mine, thus depriving the mine of 20,000 gallons of water per day. That a large current of air passing through a mine does carry off a large amount of moisture does not require scientific investigations to prove, it is easily detected in rapid return currents by mere personal contact, and in sluggish currents gives rise to very disagreeable mists, whilst it is highly deleterious to the health of the workers who have the misfortune to work in return air currents. Before mine ventilation reached its present stage of perfection in Europe miners frequently fell ready victims to asthma and other diseases of the chest and lungs through having to work in moist return currents, but since the splitting of the air currents so as to ventilate the whole of the workings with fresh air all these diseases have practically disappeared.

The writers refer to the question of size and shape of shafts for deep level mines and point out that for 250,000 cub. ft. of air per minute for a depth of 3,000 ft. a five compartment shaft would be too small and they recommend a circular shaft. They do not however state what size of a circular shaft they would recommend for this purpose. The biggest rectangular shaft yet sunk on the Rand is only equal in area to an 18 ft. dia. circular shaft but for hauling purposes in utility area it is generally considered to be equal to a much larger circular shaft than one of 18 ft. in diameter. What size then, would the writers recommend a deep level circular shaft be constructed so as to admit of seven useful compartments and at the same time provide for the requisite ventilation at a normal W.G.?

On page 545 the writers point out that in upcast shafts the air has a deleterious effect upon the shaft timbers. In discussing Mr. Penlerick's paper I pointed out that return air had not only a deleterious effect upon timber but likewise upon steel winding ropes; in fact I have known instances where it has been deemed necessary to use upcast shafts for winding men and timber, the ropes even when of identical construction have only lasted as many months in the upcast shaft, as similar ropes have lasted years in the downcast shaft say only fifty yards distant. One therefore naturally expects that if steel and timber cannot stand return air, it is equally deleterious to human beings as previously pointed out, and the passing of the air current immediately it has done its work into the return airway, without coming into contact with other working places, is one of the primary conditions necessary in a ventilation scheme. As a matter of fact on the same page the writers point out that "in laying out a ventilation scheme it is desirable that the air should be split into small circuits, in order to give each district a cool supply, since air which has been carried over great distances is usually hot and moist. After traversing these districts it should be brought into the main return as late as possible, so that the velocity in the latter is kept down and hence the pressure required is kept low."

With these expressions I cordially agree, but if Fig. 7 on page 548 be referred to it will be found that the exceedingly simple arrangement recommended by the writers violates every condition they lay down as desirable in a ventilation scheme. According to Fig. 7 the air is split, it is true, after entering No. 2 shaft, but the currents come together again at the drift driven through the dyke, thus to some extent neutralising the splitting results, are then coursed through the dip working places, thence to the fan, and through the whole of the workings served by No. 1 shaft. I need scarcely point out that this is not ventilating a mine by independent splits and returning the foul currents to the main returns, as recommended by the writers, and whilst the example given may have some advantages, its disadvantages are very obvious. It has the advantage, for instance, of cheap installation as a temporary means of aiding natural ventilation in a short lived mine, and making it possible to get the air currents well into the deep workings. But it has the disadvantage that it ventilates by return air the whole of the workings served by No. 1 shaft, whilst according to the plan such an installation cannot be installed until practically the whole of the property has been developed and nearly exhausted. This example may perhaps work, without serious objection being raised on the Rand, as an auxiliary where considerable elasticity in ventilation schemes is permissible, but, for instance, in a fiery coal mine, such an installation for permanent use would, for obvious reasons, be impracticable. The installation of a fan so far in the workings overcomes, it is true, a good deal of the difficulty experienced in getting the air into the more distant working places, but, of course, until a mine is considerably developed such installations are impossible, and the time seems to be fast approaching on the Rand when, at the preliminary opening of a very deep mine, it will be as necessary at the outset to instal a ventilation scheme just the same as it will be necessary to have a drainage scheme.

With regard to the most suitable position in which to place a fan, that must, of course, depend upon circumstances. My experience of deep mining, both as regards ventilation and other questions connected therewith, has taught me that there is a good deal of truth in the saying that "needs must where the devil drives," and to get the best results orthodox procedure has sometimes to be departed from, so that it is impossible to slavishly follow precepts and cast-iron rules, particularly in deep mining methods.

The tabulated statement, showing the temperature condition of the Village Deep, illustrates the difficulty of materially lowering the temperature in all cases by means of ventilation, in a few cases it is even higher, but the figures given by the writers show how much the atmosphere of the mine has been improved on the intake side by the installation of the fan. There can be no doubt about the convenience of using abandoned outcrop mines for the escape of foul return air from deep level workings when under the same control, but the necessary care must be observed to prevent any moist currents passing through actual working places after leaving the fan, otherwise the result of ventilating underground workings will be, that whilst on the intake side of the fan all may be much improved, on the return side the working places may be perhaps in a worse condition than before.

NOTES ON THE MATTE ASSAY.

(*Read at December Meeting, 1910.*)

By L. J. Wilmoth, A.I.M.M., M.Am.I.M.E.

REPLY TO DISCUSSION.

Mr. L. J. Wilmoth (*Member*): I have to thank Mr. Graham Martyn for his complimentary remarks, and at the same time express regret that circumstances have prevented me carrying on further experimental work in connection with this method of assay. There seems, no doubt, that further work would develop this method of assay very considerably, both as to detail and to scope. It is possible that the use of a matte might be carried into the assay of most base ores when determining their gold contents.

The President: I have now to say good-bye to you after a very pleasant year of office. I thought at first the work would be extremely hard and trying, but it has been very much lightened by the harmonious way in which the Council has assisted me at its meetings and by the support which the general body of members have given me at our ordinary meetings. I am quite sure the Society will go forward next year and increase both its membership and its usefulness.

The meeting then closed.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Analytical Notes.—"*Separation of Iron, Aluminium and Chromium.*—Tcharviani and Wunder proceed by fusing the mixed oxides with sodium carbonate in a platinum crucible for seven hours; the cooled contents of the crucible are leached out with water and the iron oxide is filtered off. In the filtrate the alumina is precipitated by excess of ammonium nitrate; the filtrate from this is acidified with nitric acid, and, after the reduction of the chromate by alcohol, chromium is precipitated as hydroxide by excess of ammonia, the precipitate dissolved in hydrochloric acid and reprecipitated with ammonia. By this method the alumina is obtained in a much purer state than either by precipitation from any acid solution with ammonia or from an alkaline solution after decomposition of the aluminate by ammonium chloride.

Estimation of Small Amounts of Lime in the Presence of much Magnesia.—In the method employed by C. Liesse the solution is diluted to the extent of 1 in 1,500 when magnesium oxalate is soluble. One gramme of the substance is dissolved in 25 parts of hydrochloric acid, and after dilution with 100 parts of water the solution is made slightly alkaline with ammonia, heated, filtered, cooled and diluted to 1,500 cc. Four grammes of ammonium oxalate are added, the solution is acidified with acetic acid, well stirred and allowed to stand two hours before filtering. Good results are obtained with 99.5% of magnesia and 0.5% of lime.

Estimation of Chromium in Chrome Tungsten Steel.—Henrik Wdowiszewski recommends a modification of Von Knorre's method. To 1 gm. to 2 gm. of borings in a half litre Erlenmeyer flask 10 cc. to 15 cc. of a 15% solution of disodic hydric phosphate and then 8 cc. to 16 cc. of sulphuric acid, sp. gr. 1.65, and 5 cc. of water are added. On gently heating the metal dissolves in about 15 minutes; the heat is then increased and 2 cc. of nitric acid, sp. gr. 1.4, are added to convert tungsten into soluble phosphotungstic acid. After dilution of the clear liquid with 300 cc. to 500 cc. of hot water the addition of 3 gm. to 5 gm. of ammonium persulphate oxidizes chromium to chromic acid. Excess of persulphate is decomposed by boiling, a little more sulphuric acid is added and the chromic acid is titrated with ferrous-ammonium sulphate, the excess being determined with permanganate.

A Sensitive and Stable Litmus Solution.—A. Puschel states that the sensitive violet colored constituent of litmus can be separated from the other coloring matter, gum and inert substances contained in commercial litmus by heating with dilute sulphuric acid 1% which precipitates it, probably as a sulphonic acid. The precipitate is collected, washed, redissolved in hot water and neutralized with sodic hydroxide. The indicator thus obtained has a violet colour and is extremely sensitive.

Estimation of Sulphur in Brass and Bronze.—Gustav Thurnauer obtains excellent results by the following method. Thoroughly mix 1 gm. of filings with 6 gm. of a mixture of two parts of potassic chlorate with one part sodic carbonate, transfer to a wrought-iron crucible lined with sodic carbonate, cover with more of the fusion mixture and heat gradually until fusion is complete. When cooled, digest the mass with hot water, filter off the oxides, acidifying the filtrate with hydrochloric acid, and determine the sulphur as barium sulphate. The author admits that the error may be as much as 0.01%, and adds that if great accuracy be required a silver crucible should be used. If 'an error of 0.01%' mean a difference of 0.01% of sulphur in the metal, the inaccuracy due to the use of an iron crucible would amount to a tolerably large proportion of the sulphur present in brass or bronze of average quality."—*Metallurgical and Chemical Engineering*, ix, 4, 217, April, 1911, p. 217. (J. A. W.)

The Portable Assay Outfit for Field Work.—"Engineers who have had occasion to prospect in isolated regions where assay offices are few and far between will be interested in a contribution to the January, 1911, *Bulletin* of the Amer. Inst. of Min. Engineers, by S. K. Bradford. He gives full details of equipment and procedure and shows the ingenuity which can be exercised when ordinary laboratory facilities are not at hand. The equipment is packed in an ordinary 26 in. valise and suffices for about 100 assays of commercial accuracy.

Equipment.—Small hammer; steel mortar with stand sawed off flat to serve as an anvil; small steel pestle; blow pipe and blow pipe cupel holder; small and large spatulas; button pliers; light 1 in. cupel mould; two 3·5 in. × 8 in. muffles; 18 ' F ' crucibles; six crucible covers; crucible tongs; 25 cc. burette; six 0·5 oz. parting flasks; six 0·5 oz. annealing cups; two or three beakers; hand balance, folding in a tin box; set of gram weights; one 6 in. 60 mesh screen; six small glass stirring rods; teaspoon and tablespoon; a micrometer for measuring diameter of buttons. The latter may be a short fixed microscope stand in which has been placed a scale of 0·1 m.m.—two or three extra object glasses being carried for safety. Chemical reagents in bottles in metal containers: 0·5 lb. KCN; 1 lb. C.P. HNO_3; 1 lb. C.P. HCl and 2 lb. aqua ammonia. A few ounces of sheet copper; 1 oz. C.P. sheet silver; 0·5 lb. sheet lead; 2 lb. tenpenny wire nails; 5 lb. bone ash; 5 lb. sodium bicarbonate; 5 lb. litharge; 3 lb. red lead; prepared charcoal for blow pipe use; candles and old newspapers.

It will be possible to obtain sodium bicarbonate at country stores and bone ash may, on occasion, be made in the field. Sheet lead may be made from bullets and silver can be obtained from coins of known fineness.

The assays can be fused in a blacksmith's forge, which is to be found at every mining camp, and the cupelling can be done at the same place by setting up the small muffle mentioned. The weight of the buttons obtained can be determined by measuring them and referring to a previously prepared scale based on specific gravity multiplied by solid contents. This method of determining weights can be found in the *Transactions* of the A.I.M.E., xxxi, 800 (1901)."—*Metallurgical and Chemical Engineering*, March, 1911, ix., 3, p 155. (J. A. W.)

INTERNATIONAL ATOMIC WEIGHTS, 1911.—"Revised atomic weights are reported as follows by the International Committee, composed of F. W. Clarke, T. E. Thorpe, William Ostwald and G. Urbain:—

Element.	Symbol.	Atomic Weight
Aluminium	Al	27·1
Antimony	Sb	120·2
Argon	A	39·88
Arsenic	As	74·96
Barium	Ba	137·37
Bismuth	Bi	208·0
Boron	B	12·0
Bromine	Br	79·92
Cadmium	Cd	112·40
Caesium	Cs	132·81
Calcium	Ca	40·09
Carbon	C	11·0
Cerium	Ce	140·25
Chlorine	Cl	35·46
Chromium	Cr	52 0
Cobalt	Co	58·97
Columbium	Cb	93·5
Copper	Cu	63 57
Dysprosium	Dy	162 5
Erbium	Er	167·4
Europium	Eu	152·0
Fluorine	F	19·0
Gadolinium	Gd	157·3
Gallium	Ga	69·9
Germanium	Ge	72·5
Glucinum	Gl	9·1
Gold	Au	197·2
Helium	He	3·99
Hydrogen	H	1·008
Indium	In	114·8
Iodine	I	126·92
Iridium	Ir	193·1
Iron	Fe	55·85
Krypton	Kr	82·92
Lanthanum	La	139 0
Lead	Pb	207·10
Lithium	Li	6·94
Lutecium	Lu	174·0
Magnesium	Mg	24·32
Manganese	Mn	54·93
Mercury	Hg	200·0
Molybdenum	Mo	96·0
Neodymium	Nd	144·3
Neon	Ne	20·2
Nickel	Ni	58·68
Nitrogen	N	14·01
Osmium	Os	190 9
Oxygen	O	16·0
Palladium	Pd	106·7
Phosphorus	P	31·04
Platinum	Pt	195·2
Potassium	K	39·10
Praseodymium	Pr	140·6
Radium	Ra	226·4
Rhodium	Rh	102·9
Rubidium	Rb	85·45
Ruthenium	Ru	101·7
Samarium	Sa	150·4
Scandium	Sc	44·1
Selenium	Se	79·2
Silicon	Si	28·3
Silver	Ag	107·88
Sodium	Na	23·0
Strontium	Sr	87·63
Sulphur	S	32·07
Tantalum	Ta	181·0
Tellurium	Te	127·5
Terbium	Tb	159·2
Thallium	Ti	204·0
Thorium	Th	232·4
Thulium	Tm	168·5
Tin	Sn	119·0
Titanium	Ti	48·1
Tungsten	W	184·0
Uranium	U	238·5
Vanadium	V	51·06
Xenon	Xe	130·2
Ytterbium (Neoytterbium)	Yb	172·0
Yttrium	Yt	89·0
Zinc	Zn	65·37
Zirconium	Zr	90 6

Mining Science, Nov. 24, 1910, p. 499. (K. L. G.)

RAPID ELECTROLYTIC METHOD OF ANALYSIS.—"Electrolytic methods of analysis have for some time been of importance in giving a satisfactory and accurate means for determination of a number of elements. A great disadvantage as to practical application, however, has been the consumption of time in the deposition of the metal. The older methods, in many cases, required from 3 or 4 to 15 hours. Nothwithstanding this great drawback, they have been successfully utilised in many places, for the determination of copper, lead, nickel, and other metals.

The rapidity with which a quantity of metal is deposited electrolytically, in weighable form, from a given electrolyte of definite volume, depends upon two factors: (1) Amount of electricity passed through the cell. (2) The number of ions present at the surface of the electrode.

The amount of any element deposited by the electric current depends upon its electrochemical equivalent and is proportional to the amount of electricity passed through the cell. In order that it be deposited in an adherent and weighable form the current must not be too high.

In order to be deposited, at any given instant, ions of the element must be touching the pole on which the deposition is to be made, and in order that this be true at all times, as long as there are any ions present in the solution, the electrolyte must be agitated, so that fresh solution is always being brought in contact with the electrode on which the deposition is being made. This fact has been utilised many years by men making the deposition of different metals on a large scale, for other than analytical purposes, before the fact became impressed upon chemists working on electrochemical methods of analysis. The principle has been applied to this branch of analytical chemistry in different ways by many men, with varying degrees of success, and in the greater number of such trials the deposits have been as good, and have been made much more rapidly than by the older methods. They have, at times, likewise, allowed the use of electrolytes, which do not give satisfactory deposits by the slower methods.

In 1902 to 1903, F. F. Exner, while working at the University of Pennsylvania, agitated the solution by the rapid rotation of the cathode and was enabled to deposit, in a weighable form, in from 5 to 10 minutes, from 0·25 gm. to 0·50 gm. of many of the metals. About this time, F. A. Gooch, at Yale, was working along the same line, but instead of rotating the cathode, he, in his work, stirred the solution by the rotation of the other pole or anode. His experiments, were, likewise, very successful and show that agitation is, without doubt, the point to be considered in the rapid deposition of metals in a weighable form. A quotation from the work of Henry Sand, which states in a concise manner why the use of these high currents is possible, will not be out of place. He says: 'It is most probable that high current densities are possible and depend solely upon the rapidity of renewal of the liquid at the electrodes. It is extremely likely that in metal precipitation the potential at the cathode is independent of current density. The great variations observed, when applying different current densities, are almost wholly the consequence of local concentration changes. The great role, which such changes, under circumstances, can play, I showed, four years ago, in the electrolysis of copper sulphate solution containing sulphuric acid. Just as long as copper ions, in appreciable concentration, were present at the surface of the touched electrode, these alone were precipitated. When, however, they had practically disappeared from the touched surface, all copper, migrating in that direction, was, by diffusion, set free simultaneously with the hydrogen. In all instances, as a consequence of the local exhaustion of copper sulphate, in spite of convection, heating, hydrogen evolution, etc., over 60 % of the current was consumed in liberating hydrogen. On agitating the solution energetically, copper alone was precipitated. Had the purpose of these trials been to determine copper, that metal would, in the first instance, have separated in a pulverulent form; in the second, in a coherent precipitate.'

The different forms of apparatus, which have been devised to apply this principle to electrochemical analysis, have been multiplied almost indefinitely. Two will serve as illustrations. Fig. I. represents one of the most simple forms, in which the cathode is rotated. The cathode consists of a platinum crucible, placed over a cork on the end of the motor shaft, by means of which it is rotated from 100 to 1,200 times per minute. The anode, in this form of apparatus, is a common foil electrode, placed as near as possible to the side of the beaker, in which the deposition is made. Fig. II. is another form of apparatus, in which the anode is rotated by any means at hand and the deposition made on the platinum dish, which holds the electrolyte and at the same time serves as a cathode. One of the more recent devices has been the application of the solenoid. By the utilisation of this, it is possible to make the solution itself revolve, by means of the magnetic stress set up within it. This form of apparatus, while allowing of a more rapid precipitation of some

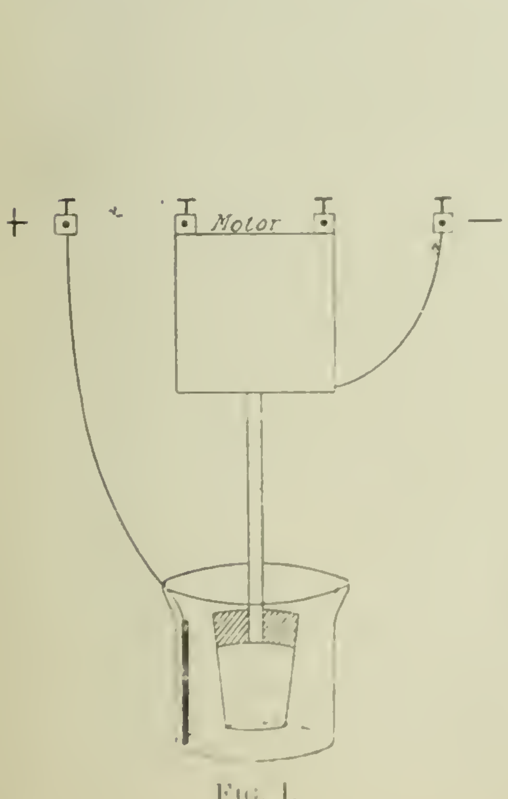


Fig. I.

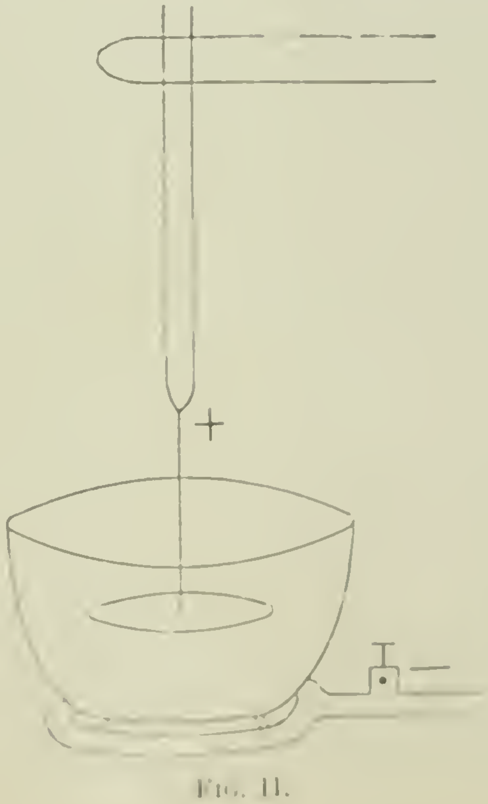


Fig. II.

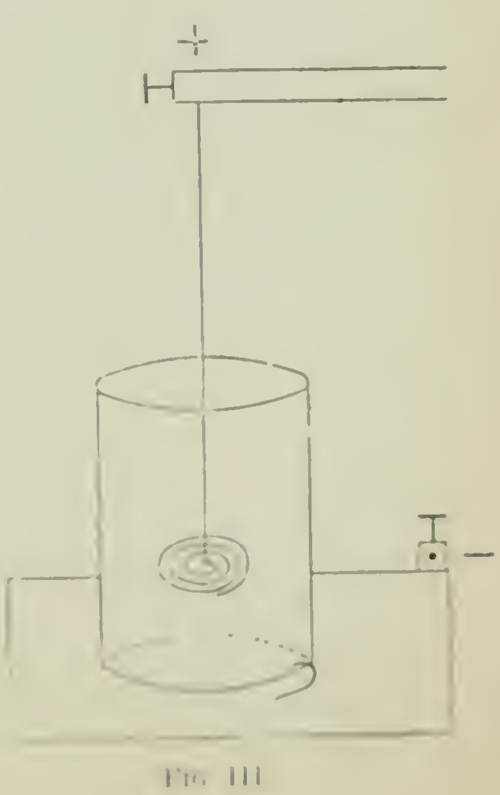


Fig. III.

of the elements than is obtainable by the older methods is not equal, in this point, to the rotating cathode or anode.

Probably the apparatus which will prove itself of the most value, especially to the technical chemist, is the gauze electrode recently used by me for the determination of lead and copper. This, while it has no more, and in many cases, not even as much, surface as the ordinary foil electrode, allows of the free circulation of the electrolyte, when agitated by the rapid evolution of the gas caused by the use of high currents. In form this has the advantage of simplicity and where applicable, it allows of a deposition sufficiently rapid for all technical determinations, that is, it is possible to deposit all of the copper or lead found in a 0·5 gm. sample of copper or lead ore in less than 20 minutes.

Another electrolytic device, which deserves much more attention than has been as yet given it, is the mercury cathode. This is nothing more than a beaker of 50 cc. capacity (Fig. III.) with a platinum wire fused through the bottom, or, better, at the edge, where the bottom joins the side. The platinum wire makes a contact with the mercury, which is placed in the bottom of the beaker, the electrode weighing, when containing 40 gm. to 50 gm. of mercury, not more than 80 gm. The solution to be electrolysed is placed in the beaker over the mercury, an anode introduced, and the mercury made the cathode by connecting the other pole of the battery to the platinum wire, which was fused through the glass. By means of this simple apparatus, it is possible, with currents varying from 3 to 4 amperes, where 20 cc. of the solution is used, to deposit as high as 0·5 gm. of a number of the metallic elements in from 15 to 20 minutes and, by washing with distilled water, alcohol, and ether, to make a determination as accurately as by most other methods, and in some cases, such as that of bismuth, with much more ease and accuracy. Again, some of the most difficult chemical separations can be successfully made by this method. The separation of iron and titanium, in a sample of titaniferous iron ore, will serve as an example. The iron can be quantitatively deposited in the mercury, from a sulphuric acid solution containing iron and titanium, if from 0·1 gm. to 0·2 gm. of iron be present and the volume of the solution is not too great, in as short a period as 20 minutes. The mercury cathode weighed, before and after, gives the amount of iron present. The electrolyte can then be reduced by zinc and the titanium titrated in the usual manner.

This simple form of apparatus shown in Fig. III., gives results as quickly as can be desired in most cases. However, they can be obtained even more quickly, if, instead of the stationary anode, a rotating anode is used."—L. C. BENNER.—*Mining and Scientific Press*. Oct. 29, 1910, p. 576. (K. L. G.)

METALLURGY.

TELLURIUM-CONTAINING ORES. — "*Minerals.*—The following tellurium minerals have been found and identified in Canada :—

Sylvanite, (Au Ag) Te.
Hessite, Ag_2 Te.
Petzite, $(Ag, Au)_2$ Te.
Calaverite, Au Te_2.
Nagyagite, a sulpho-telluride of lead and gold, containing about 7% of antimony.
Altaite, Pb Te.
Tetradymite, approximately $Bi_2 (Te, S)_2$.
Tellurite, (Telluric ochre), Te O_2.

In addition, a mineral thought by Dr. Hoffman to be native tellurium, was found at Long Lake, Greenwood mining division, British Columbia.

Oxidation.—All these tellurides oxidise readily and give rise mainly to telluric oxide, tellurites or tellurates with bismuth, iron, and probably mercury, and native gold, silver, also rarely, bismuth ; the telluric oxide, also known as tellurite and telluric ochre, in its pure state is a soft (H.2) white to yellowish mineral, but generally occurs mixed with other oxides, frequently those of iron, when a yellowish to reddish powder or coating results.

Mineral Associations.—With the exception of tetradymite, these minerals occur in Canada only in veins which may have been produced either by fissure filling or by metasomatic replacement or may owe their existence in a large degree to both agencies. The veins are found in a great variety of formations, including various schistose rocks, granite, granodiorite, diorite, andesite, volcanic breccias, quartzite, slate, shale, and arkose ; but with but one known possible exception they are all either in, or in the vicinity of, igneous rocks. In all cases, so far discovered, the veins consist predominantly of quartz, but calcite, and, rarely, ankerite and sericite occur as gangue minerals. Several varieties of tellurides are in many cases intimately associated in the same deposit and wherever one or more of these occur, native gold, silver and in places bismuth and copper are characteristic accompanying minerals, in addition to which, pyrite, chalcopyrite, and galena are present in many of the veins, and pyrrhotite, chalcocite, siderite, and zinc blende are found in some of them.

Sylvanite, hessite, petzite, calaverite, nagyagite, and altaite are, in all the deposits so far discovered, minerals of primary deposition, so that their presence indicates a zone of unoxidized or at least incompletely oxidised material. The native gold, so commonly found associated with the tellurides, has in many cases been derived from these minerals by oxidation, when it is generally spongy and frequently has a brownish colour ; where this origin for the gold can be proved, the zone in which it and the telluride occur together lies above that of the primary sulphides and consequently (as the zone below ground-water level is reached, *i.e.*, below the oxidized zone and the zone of secondary enrichment), the gold may be expected to cease to be free and to occur instead, combined with tellurium.

Tetradymite is known to have been discovered at but two points in Canada, viz., at the Nickel Plate mine Osoyoos mining division, and on the Roderick Dhu claim, Greenwood mining division, both in British Columbia. At the Roderick Dhu claim tetradymite occurs in a quartz vein like the other tellurides ; but at the Nickel Plate mine it is found in contact-metamorphic rocks, and is probably also a primary constituent of the ore, although Mr. Camsell, who has described the occurrence, considers that possibly this mineral may be the result of secondary surface alteration.

Uses and Value.—Tellurium as yet, has practically no technical use and consequently there is very little if any demand for the metal itself except for experimental purposes. A number of uses to which this metal may be put have recently been proposed, however, and concerning it, Edison says : 'I have worked it to a considerable extent and I pronounce it a marvellous substance ; I could use it in large quantities if it were cheaply produced.'

Potassium tellurate is employed to a limited extent, however, in therapeutics, being used for night sweat of phthisis.

Treatment and Assay.—To enter into a detailed discussion concerning the assaying and treatment of tellurium-containing ores, would be beyond the scope of this short paper; but a few lines on this topic may not be entirely amiss, since a study of the literature dealing with these ores cannot fail to impress one with the fact that tellurium has a much more general distribution and is more commonly associated with gold than has been commonly supposed.

Treatment.—Ores in which tellurium occurs associated to any considerable extent with the precious metals have been a source of perplexity to many millmen and metallurgists, particularly in American practice; in many of these cases, possibly in the majority of them, the tellurium has not been detected or if it has been discovered, its presence has been ignored with disastrous consequences. In milling processes, it is found that ores in which compounds of tellurium are prominent, are not susceptible to amalgamation or to the ordinary methods of hydraulic concentration. Concerning smelting operations Kustel states that 'not all tellurium combinations with gold, lose gold to a notable extent while roasting; but some do, and that up to a considerable amount, 20%, perhaps even more.' He also adds: 'The loss is no mechanical one, occasioned by draft of the furnace, but principally by volatilization.' The result of the various investigations from those of Plattner to the present, seems to be that in all ordinary oxidizing roasting of gold ores containing no tellurium that there is no loss of gold other than a mechanical one; but where ores are involved in which there is any appreciable amount of tellurium, there appears to be always a loss of gold by volatilization, in any operation involving the roasting of the ores or any of their products from which neither the gold nor the tellurium has been separated. Mr. F. C. Smith has made a careful investigation of such ores and in an excellent article concerning them writes:

'Careful experiment with such ores, especially where conducted at a reduction works handling large quantities of them, should not only finally avoid embarrassment from the presence of the tellurium, but even turn it to a profit.' This indicates that the only objectionable influence tellurium in ores possesses is likely to be overcome; when this is established the discovery of this metal in an ore deposit will be cause for unalloyed gratitude on the part of the owner, as tellurium is almost invariably an indication of the occurrence of gold.

Assay.—On the subject of the assaying of ores containing tellurium, Mr. Chas. A. Fulton has published a very carefully prepared article, and according to him scorification of any kind is a very bad, and the crucible assay is the best, method. The fire should be moderately hot, a large excess of litharge should be used, and a button of 20 to 28 grams should be obtained, which can be cupelled directly.

Identification.—The presence of tellurium is frequently overlooked partly because, in many cases, it occurs in very small amounts, and also because its determination is a somewhat difficult and laborious process in comparison with that of gold and silver. A small fraction of an ounce of gold to the ton of ore may be easily and accurately determined by dry assay; but a similar amount of tellurium, or other metal to be found by wet methods, calls for a refined analysis. A refractory character in ores, or even the recognition in the gold carried by them of the brown spongy appearance characteristic of gold derived from the alteration of tellurides, is sufficient to arouse suspicion as to the presence of tellurium. A further significant fact concerning ores containing this metal is that even when very rich, they often fail to show colours in panning."—D. D. CAIRNES, Ph.D., *Geological Survey, Canada*—*Canadian Mining Journal*, April 1, 1911, p. 215. (A. R.)

CORNISH BATTERY COSTS.—"The interesting analysis of battery costs issued by East Pool and Agar United Mines for the year 1910 deserves attention. According to the figures presented, the costs were respectively:—Pneumatic stamps, power per ton 10·67d., stamping per ton 6·68d., total 1s. 5·35d. Cornish stamps, power per ton 1s. 5·66d., stamping per ton 7·90d., total 2s. 1·56d. On this it appears that no less than 6·99d. per ton was saved on the engine costs and 1·22d. on the stamping costs. The air-cushioned stamps at East Pool are driven by a modern cross-compound Corliss engine, which was also supplied by Messrs. Holman Bros., and evidently the combined results attained are very satisfactory."—*Mining World and Engineering Record*, April 15, 1911, p. 438. (A. R.)

ANTIMONIAL GOLD ORE TREATMENT.—"A new proposal for the treatment of gold-bearing antimony ores is made by John Jones and H. S. Bohm, of Mount Morgan, Queensland, Australia. They claim that aside from being effective for the purpose, the process also is economical by reason of its possibility of regenerating the chemicals used.

A hydrochloric acid generator and tower are part of the plant. The first treatment is with HCl, and as the ore is a sulphide the action of the HCl on the ore is to form H_2S. In another operation the chloride solution of antimony is electrolyzed, generating chlorine which is used to combine with the H_2S formed in the first instance to regenerate HCl, according to the equation $H_2S + Cl_2 = 2HCl + S$. The regenerated hydrochloric acid gas is returned to the the tower for enriching the solutions.

The cycle of operations proposed is as follows. The pulverized ore is agitated in a dissolving vat with hydrochloric acid solution, heated by exhaust steam or other means. The generated H_2S is conducted to a regenerating chamber where it combines with Cl conducted from the electrolyzing cell, to be referred to later. When the antimony has been dissolved, the solution is filtered and transferred to the electrolyzer, fitted preferably with carbon anodes and antimony cathodes. The Cl from the cell is conducted to the regenerator mentioned above where it combines with the H_2S. The sludge from the first treatment tank contains the gold and other valuable metals, and may be treated in any suitable way, preferably by chlorination."—*Metallurgical and Chemical Engineering*, IX, 4, 218, April, 1911, p. 218. (J. A. W.)

FINE GRINDING.—"It may be said without fear of argument that the developments in the metallurgy of clean gold and silver ores in the past decade have been synchronous with the extended adoption of the principles of fine grinding, and to-day this factor in the treatment of such ores is receiving closer attention than ever before. The many problems involved may be divided into two parts, the first dealing with the mechanical devices for the performance of the work, and the second with the particular application of the method to any given ore.

In the minds of many engineers a prejudice exists against what is known as the 'all-sliming' scheme because of a belief that enormous cost both in outlay for the necessary machinery and in operating it must ensue, the latter to such a degree that any improvement in extraction to be obtained by fine grinding would be more than off-set by the contingent disadvantages. To a limited extent and in special cases this is undoubtedly true but as a generalisation it is certainly wrong. 'All-sliming' in its general acceptation is intended to mean the absolute reduction of the whole of the ore to something over 150-200* fine, but it is also being applied where portions of the pulp are classified and subjected to varying degrees of fine grinding. It appears to me that the term is still applicable to such cases provided that the coarsest product handled is not less than 100 mesh. The use of the term, however, is of no particular moment so long as it conveys something reasonably understandable and the final dictum on that point may well be left to some of the technical societies.

Having conceded the undoubted virtues of fine grinding the questions of greatest importance are its limiting factors and its efficiency in the treatment of a given ore. The limiting factors are primarily involved in a comparison between the added cost of each extra unit of fine grinding and the added extraction, but this is by no means easy to determine accurately. The recent contributions by H. Stadler on 'Grading Analysis and their Application' go a long way toward elucidating the more abstruse of the problems involved in the computation of the actual work done in regrinding operations, and if it be allowed that the actual work done can be measured against an accurate record of the power consumed in doing that work, it becomes possible to arrive at the all-important figure of cost.

As to the relative recoveries on various products measured according to the varying degrees of fineness, the difficulties are less pronounced and provided that due care is observed in carrying out the investigation there is no doubt that the point is susceptible of ready determination. It is after making the assumption that these questions can be settled on a practical basis—and this is by no means mere assumption, although finality has not yet been approached—that the really economic considerations arise, and it is with these that I now propose to deal.

Take, for example, a silicious ore containing 5% of pyritic material. Let us assume that this material be crushed to 60 mesh and that from the resultant pulp 90% of the pyritic content can be concentrated out, carrying 70% of the gold. Taking the assay of the ore at 8 dwt. per ton, we then have after concentration:—

4·5 tons of concentrate carrying 560 dwt. = 124 dwt. per ton.

95·5 tons of pulp carrying 240 dwt. = 2·51 dwt. per ton.

The pulp would then be classified into 65·5 tons of sand containing, say, 3 dwt. per ton and 30 tons of slime at 1·45 dwt. Again assume that each of these products by agitation and filter-pressing and without any further regrinding would yield 80% recovery. The question would then naturally arise as to the application of finer grinding and in what particular form. At the outset it is quite clear that the pyritic constituent of the ore would demand special attention. An 80% extraction on the 124 dwt. would leave a residue of 24·8 dwt., and, although from the whole pulp without concentration on the basis of 80% of 8 dwt. the final residue would only amount to 1·6 dwt., it would at once seem possible materially to reduce even that residue by a proper handling of the concentrated portion.

Fine grinding would at once be suggested as the remedy and this might be achieved by:—

(*a*) Regrinding the whole of the pulp from the crushing plant that is coarser than 200 mesh.

(*b*) Regrinding a hydraulicly classified portion of the pulp.

(*c*) Regrinding a carefully concentrated portion of the pulp.

Assuming that if the products were reground to pass 200 mesh, which postulates the grinding of a great portion of it to something much finer, we should increase the extraction from 80% to 94%. The position would then be as follows:—

Concentrate—

80% of 124 dwt.	99·2 dwt.
94% ,, ...	116·5 ,,
An increase of	17·3 dwt.

Sand—

80% of 3 dwt.	2·4 dwt.
94% ,, ...	2·82 ,,
An increase of	·42 dwt.

Slime.—As the slime is already of the required fineness we may assume that the extraction from this product remains at 80% of 1·45 = 1·16 dwt., although at 94% the extraction would only amount to 1·36 dwt., being an increase of 0·2 dwt. per ton.

It is obvious that it is to the concentrate that we must look for the chief improvement in adopting fine grinding and the question is whether the right policy in this particular case would not be to aim at a very clean concentration with the subsequent fine grinding of that product only. This is an important question and the solution of it must have far reaching effects on the ultimate construction and operation of the plant. The points that arise are:—

1. What is the cost per ton of regrinding from 60 mesh to 200 mesh?

2. Assuming that the sand is not reground would it not be necessary to provide a special plant for final treatment. In other words, could the whole pulp be treated as one product—including the reground concentrate—if the sand were not reground?

3. Would it be necessary to regrind the sand to as fine a mesh as the concentrate either for the purpose of getting extraction or of making the product suitable for filter pressing?

4. Which is likely to be the more economical in capital outlay and operating cost, three separate plants, one for concentrate, one for sand, and one for slime, or one plant to take the product in one operation?

5. Assuming that 50% of the valuable contents could in the first place be extracted by amalgamation, although the ultimate recovery would not thereby be increased, would it be advisable to adopt amalgamation?

6. As the concentrate contains the bulk of the precious metals would it not be advisable to make special provision for regrinding this product, in order to ensure an absolute reduction to impalpability. Could this be done, and how?

Taking these points in sequence:—1. The cost of regrinding from 60 mesh to 200 mesh will vary according to the nature of the material to be reground and also according to the cost of power and the par-

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That is, able to pass through a screen having from 150 to 200 apertures per linear inch.

ticular machinery used for the purpose. In my experience the average figure may be put at 3s. per ton.

2. It would be necessary to have a separate plant for the handling of the sand if it were not reground, whether the subsequent treatment of the slime involve decantation or filter pressing. In filter pressing the presence of coarse sand causes channelling and vitiates the results.

3. It would not be necessary to regrind the sand as fine as the concentrate either for purposes of extraction or with the object of producing a product suitable for filter pressing. The added cost necessary to reduce the large tonnage of sand to pass 200 mesh would not be justified from the increased extraction point of view, while from the filter pressing standpoint 150 mesh would be quite fine enough.

4. There can be no question that a plant designed to handle one product, and one product only, continuously, will be much more simple in design, less costly in capital outlay, and cheaper to work than the plant consisting of three separate parts.

5. There is nothing to be gained in recovering 50% by amalgamation: first, because the operation, though not an expensive one, costs something; second, it is always easier to steal gold caught in amalgamation than gold held in zinc boxes; thirdly, the ultimate recovery is likely to be rather more than less without amalgamation as the losses arising from the use of that process are thereby avoided: fourth, the extra consumption of cyanide and zinc is almost inappreciable. For these reasons there is no object in using amalgamation.

6. Given that a clean concentrate can be made such as that indicated in the case under review, there can be little doubt that it would be wise to make as clean a concentrate as possible. For this purpose it would be necessary to use some mechanical device in order to ensure the saving of the bulk of the concentrate. It is possible to arrange for the regrinding in such a manner that the reduction to impalpability can be absolutely assured.

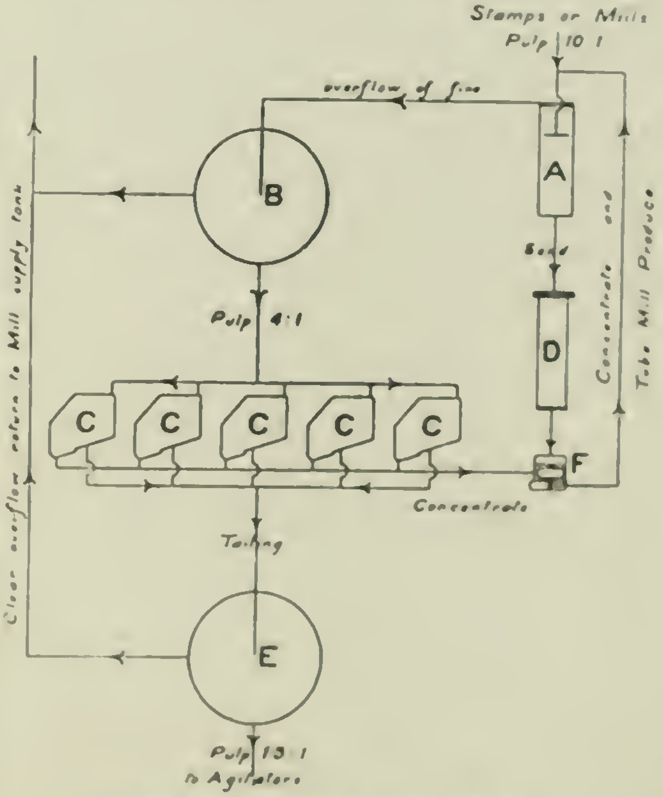
One of the great objections to rejecting amalgamation is that the coarse gold will offer such a resistance to a dilute cyanide solution that in all probability it would not be dissolved in the time provided. Those who raise this objection, however, overlook one important point: that the concentrate with which the coarse gold is associated is taken direct to tube mills from which no escape of coarse gold is possible. In fact, once the coarse gold reaches the tube mill its condition is rapidly changed to thin flaky gold and in this condition it offers little resistance to solution. I desire to emphasise this point as it has been so often raised and there is no doubt at all that in actual practice those who raise the objection will find that the difficulty will disappear. Of course, I assume that in the handling of the pulp as one product, cyanide solution would be circulated throughout the entire plant, in which case the coarse gold, after being thinned in the tube mill, will pass from it in solution. In every case where the pulp can be handled as one product there can be no question as to the advisability of circulating solution throughout the entire plant. The advantages that accrue in the matter of saving of water, pumping cyanide solution, damming, etc., are too well known to require a detailed description here, but the principle is so important that it may be taken as the crux of all-sliming work.

In regard to the method of securing with certainty the thorough regrinding of the concentrate I cannot do better than give the following brief description of the suggestions put forward by A. Grothe in a paper read by him before the Mexican Institute of Mining and Metallurgy.

'The ground pulp from stamps or mills, containing about 10 of solution to 1 of ore, goes to a Dorr classifier (*A*), the overflow from which is condensed in the pulp thickener (*B*) to 4 : 1, or any other proportion suitable to the kind of slime table used. This overflow consists of particles all passing 150 mesh, only a small percentage being retained on 200 mesh. The coarser part of the pulp, constituting the underflow of the classifier, goes to the tube mill, or other regrinder, with the proper degree of moisture. It contains the sand from which the precious metals have not yet been completely liberated, the pyrite, and the coarser particles of silver sulphides and gold. The overflow of the classifier, after having been properly thickened in (*B*), is concentrated on slime tables (*C*). The concentrate from these tables, instead of being sacked and sold as usual, is returned to the pump (*F*), joined to the tube mill discharge, and enters the classifier again.

'It will be seen that the concentrate is thus kept in circulation until so fine that it will not settle on the tables but float off by surface tension and is then carried with the tailing to the second pulp thickener (*E*), which produces the proper density of pulp for treatment by cyanide in agitation vats. The fineness of this concentrate, which, after repeated grindings, at last pass off with the tailing, surpasses anything that can be separated by the finest screens or by the overflow from cones or classifiers. In this state it will behave exactly as the metals in the ordinary pulp: the extraction will be completed in 24 hours or less; there will be no reprecipitation and the consumption of cyanide will be in proportion to the metallic contents of the ore.

'The objection might be raised that an accumulation of concentrate will take place on the tables, and that additional work will be thrown on the tube mills. Both are no doubt correct to a certain extent,



Flow Sheet suggested by Albert Grothe

but accumulation will not proceed indefinitely, because, as more particles are presented to the grinding apparatus, a greater effect will be produced, and after a short time a state of equilibrium will be established and the quantity of fine concentrate flowing off the tables will equal that of the coarse concentrate contained in the ore. As to the extra work thrown on the tube mills, this cannot be correctly estimated, but supposing the ore to contain 2% of concentratable matter, which thus requires regrinding, and that it were necessary to pass it five times through the mill before it is fine enough, then the loss of the capacity of the tube mill would be 10%. The additional cost for regrinding would therefore be insignificant when compared to that of disposing of the concentrate to the smelter or subjecting it to a special treatment.

'A further advantage of the system is that the action of the solution on the concentrate passing several times through the tube mill is considerable, and shortens the time necessary for agitation. It can easily be tried and its economic value for every case determined, in any existing mill, without making important changes or the addition of expensive plant.'

The main point made by Mr. Grothe is that it is necessary to make certain that the concentrate shall be reduced to the finest possible condition and the adoption of his suggestion gives an assurance that this end will be achieved. J. L. Mennell, in discussing Mr Grothe's suggestion, pointed out that another advantage of the system lies in the fact that the concentrate is not reground alone. He states that unless mixed with sand the concentrate is apt to pack and cause difficulty in regrinding, whereas when mixed with the sand the tendency to pack is obviated and simultaneously assistance in the operation is actually gained by such presence of the sand. Furthermore he points out that it also avoids the use and expense of a separate mill. The only disadvantage, according to Mr. Mennell, lies in the 'diminution of the capacity of the tube mill by the repeated handling of the same material,' but this in his opinion is more than counterbalanced in the saving of the additional material requiring separate attention.

Another feature of this system is that the concentrate by repeated passage through the tube mill gets a longer time of actual treatment than the ordinary pulp. This certainly is an advantage.

Every ore must naturally be treated on its own merits and according to the constitution of the ore so will practice vary: for instance, we may have any of the following conditions:

1. An ore that could be concentrated to give 80% of its valuable contents to the concentrate such concentrate not exceeding 10% of the total ore; in this case unless the ore were high grade it would certainly be un-necessary to regrind anything but the concentrate and it would be a matter of experiment to decide whether a hydraulic classification would serve the purpose or whether mechanical concentrators should be employed.

2. An ore that contains practically no concentrate and in which the gold is distributed fairly evenly throughout; in this case it is exceedingly probable that after the first separation of the slime the rest of the pulp would have to be reground.

3. An ore containing a large proportion of pyrite with which only a portion of the gold or silver is associated; in this case the same practice would have to be followed as above.

4. An ore containing a small porportion of rich pyrite but still leaving considerable gold and silver in the pulp; in this case probably the most satisfactory procedure would be to resort to a hydraulic classification by means of spitzlutten, with special regrinding of the concentrate, and with auxiliary regrinding of the remainder of the ore.

All of these cases will be found to occur in actual practice and each one must be judged on its own merits. It may be taken as axiomatic, however, that in every case where, either by hydraulic classification or mechanical concentration a comparatively small proportion of the ore of relatively high value can be separated, it will prove expedient to make that separation and provide for a special regrinding of that product embodying the 'return' features set out in Mr. Grothe's suggestion. Of course, it is not a new idea to suggest the return of the coarser particles, and in fact the idea has been employed in grinding work for the past 20 years; it is clearly stated in Mr. Grothe's notes, however, and I deem it well worthy of emphasis.

The day of the percolation method of treatment for sand and the decantation system for the treatment of slime has gone, except in special cases, and it is only due to the persistence of some of the metallurgical fraternity who have been wedded to these ideas that they are to-day included so freely in the treatment of simple gold ores. There can be no logical argument opposed to the principles of the one-product treatment now becoming almost universal and the only points remaining for settlement are the methods and devices to be adopted in fine grinding and the degree to which this principle is to be pushed in order to gain the best results. Modern practice has so emphatically taken up the circulation of cyanide solution throughout the entire plant followed by fine grinding and filter pressing, and the results have so abundantly demonstrated the fact that this practice is a great step forward as compared to Rand methods even of to-day, that there can be no longer any discussion as to the relative merits of the respective principles. The fight in future will lie rather in the finer points of the newer process than on the question of its superiority to old practice.

I venture to predict that the tendency of the future will be to crush in stages, classify, grind in stages, agitate, and filter press on the one-product principle and I believe that the upshot will be cheaper plants, lower working costs, and improved extractions. Probably the unit vat-capacity of plants will be considerably increased, especially those vats responsible for the feeding of the filter presses. There is no reason why two big vats should not hold a three days' supply for a filter press plant of 400 tons capacity and it will be at this end of the equipment that the storage reserve will be created rather than in the battery bins. Air agitation has been so successfully used on pulp consisting of slime and fine sand that it is bound to continue to play an important part in agitation work, and although the shape of the Pachuca tank may be considerably modified the principle upon which it is operated is bound to be recognised, and in some form or other it will be embodied in all new plants."—H. S. Denny.—*The Mining Magazine*, Vol. iv., March, 1911, p. 219. (J. E. T.)

Notes on Cripple Creek Mill Practice.—"The interest which, for the past two years, has centred in the cyanidation of Cripple Creek raw ore will not abate much this year. In fact it is more likely to increase for the reason that we do not yet know the chemical details of one of the processes used, and further, that a wholly new principle is to be applied this year in a new mill now building. The successes of 1909 and 1910 will be followed, we hope, by still another in 1911, and the field for local treatment of Cripple Creek ores be thereby greatly enlarged. It seems certain even now that the list of metallurgical failures at Cripple Creek will not be added to, if intelligence is displayed by those who direct future ventures.

At the present writing the secret of success is told in a few words—ordinary cyanidation assisted by preliminary concentration and the application of cheap chemicals with the cyanide. The object of the concentration is to remove a small quantity of refractory mineral, the retention of which would only increase the time of extraction of the whole mass, or possibly defeat the whole process. The comparatively small quantity of the ore thus removed can be sold to smelters or given such special treatment as its requires. But the prospect now is that even this step will be rendered unnecessary by the Clancy process which will be in operation in a few months.

Stratton's Independence Mill.—The method of treatment used in this mill was worked out by Mr. Philip Argall after many years' experience with the cyanide process and Cripple Creek ores. Based on experiments made at a much earlier date, the plant finally was put in operation in April, 1909, after some unavoidable delays and in spite of the unfavorable criticism of doubting Thomases. The problem was to treat at a profit a large dump of low-grade ore, amounting to almost 1,000,000 tons, which had accumulated for many years. The average value of the dump was determined by extensive sampling to be $3·80 per ton. This figure is much lower than the cost of treatment at some of the earlier mills and is even less than the lowest treatment rate now offered by custom mills on low grade Cripple Creek ores, viz., $4 per ton. That the cost of treating this dump in the Independence mill is about $1·50 per ton is evidence of the general improvement in milling methods and of the ability of the technical staff in charge of the work. The mill was the first successful cyanide plant treating sulpho-telluride ores in the Cripple Creek district, although several successes had been scored previously in cyaniding the oxidised ores. It has been steadily enlarged since beginning operations from an original capacity of 4,500 tons per month to practically more than 9,000 tons per month at the present time.

Crushing and Grinding.—The dump ore is mined by electric shovel, loaded into 4-ton cars and hauled up an incline to the coarse crushing plant. Two cars are in use, one being loaded while the other is dumping. In this way a car is loaded and dumped about every three minutes, which permits crushing a day's run of 300 tons for the mill in about four hours. The cars are dumped directly over a 7½ D Gates gyratory crusher which reduces the ore to 4 in. size. The crushed ore is discharged over a grizzly with 2 in. openings, the oversize falling on to a picking belt where pieces of steel and wood are removed. The belt discharges into a 5 K Gates gyratory, which crushes the ore to 1¼ in. size. The product of the second gyratory is combined with the undersize of the grizzly and elevated by a belt conveyor to the storage bin in the concentrator building.

The design of the storage bin and feeder insures a positive steady feed of ore. The bin is circular with conical bottom, the cone terminating in an opening 6 ft. in diameter, through which the ore feeds into an open cone about 12 ft. in diameter at the top and placed with its upper openings in the same horizontal plane with the 6 ft. opening of the main bin. The lower cone terminates in an opening about 12 in. in diameter, through which the ore is discharged by a revolving table feeder. By this construction the ore flows freely and the bin may be completely emptied without extra attention.

The method of adding lime for protective alkalinity differs from the usual practice of adding it in a solution form. As the ore leaves the storage bin it is moistened with a small quantity of mill solution, which settles dust and slakes the solid lime, which is added continuously at the rate of about 1,000 lb. for the day's run of 300 tons of ore. This gives a protective alkalinity in the solution of 0·02 %. The mixed ore and lime are then elevated to bins above the rolls, and further reduced in two successive sets of rolls to [illegible] in. size. This material, which constitutes the Chilean mill feed, is then elevated and distributed to three bins, one for each mill. The Chileans are of the fast running type, 6 ft. in dia., making 33 r.p.m. They grind in mill solution which is kept at a strength of about [illegible] lb. KCN per ton and discharge through 12-mesh screens. The capacity is about 100 tons of ore each per 24 hours, 60 % of the pulp passing 150-mesh screen.

Classification.—Sand and slime are classified prior to concentration in two spiral classifiers, each of 300 tons daily capacity. The sand is delivered 5 ft. above the feeding point, carrying from 4 % to 7 % slime, depending on the varying nature of the ore. The slime emerges 6 in. below the feeding point, carrying about 3 % of sand coarser than 150 mesh.

Sand Concentration.—Mill solution is added to the sand as it leaves the classifier, and the pulp is distributed to 20 Card tables. Two grades of concentrate are obtained, the first carrying about 5 oz. gold per ton, which is shipped to the smelter; the second with about 0·75 oz. gold, which is sold to chlorination works. The Card table is preferred for this work as it gives a cleaner division between the two classes of concentrate and a consequent financial advantage in their disposition. The sand tailing is laundered to a sump and delivered by centrifugal pump to the cyanide plant for further treatment.

The coarser and heavier sand is separated by spitzlutten and reground in a tube mill. The reground pulp is classified by cones into sand and slime, the former being treated on two Card tables and the latter on four Deister slimers. All of these tables produce first and second grade concentrates.

Slime Concentration.—The slime from the classifiers is thickened in 16 8 ft. cones. The underflow is distributed to 10 Deister slimes tables and four vanners. Like the Card tables, the Deisters make two grades of concentrate, the first carrying about 7 oz. gold per ton and the second from 0·50 oz. to 0·70 oz. The total concentrate removed by sand and slime treatment amounts to about 3 % by weight of the ore treated and contains 40 % of its value. The tailing from the Deisters and vanners is laundered to a sump which also receives the overflow from the thickening cones and the tailing from the tables treating reground sand. A centrifugal pump returns the slime to the cyanide plant for further treatment.

Cyanide Treatment.—The regular cyanide treatment is supplemented by the use of bromo-cyanogen. The sand and slime pumped from the concentrator are delivered to the head of the cyanide mill where they are [illegible]. The sand is discharged to a reciprocating conveyor which distributes to six [illegible] filter bins, each of 250 tons capacity. The sand enters the bins with an assay value of 0·06 oz to 0·08 oz. gold per ton and is discharged, after treatment, with 0·03 oz. to 0·04 oz. The treatment is as follows: After a bin is filled and drained it is leached with solution strengthened by the addition of 25 lb. KCN to each bin. When this solution has filtered the sand is given successive washes with barren solution and water and sluiced out.

The slime from the classifiers flows to four large settling tanks, the overflow from which is usually returned to mill solution storage. It is the intention to keep the mill solution at a strength of ½ lb. KCN and at a value not exceeding $0·50 per ton. When this is exceeded the tanks are not allowed to overflow, but are decanted to the precipitating department. The thick settled slime is drawn from the bottom of any one or all of the tanks and pumped to the agitator for 10 hours agitation, with air at 30 lb. pressure. A charge consists of from 75 to 100 tons of dry slime and from one and one-half to two times that amount of solution. Thirty lb. KCN are added to each charge. The agitation tanks are modified from the standard construction in air agitators by shortening the central tube one-half, thereby securing the full benefit of the air in agitating the pulp in the upper half of the tank. After 10 hours' agitation the charge is pumped to a second agitator for special chemical treatment.

Filtration.—The vacuum filter is of peculiar construction, generally according to the Cassell type, and consists of 110 leaves 7·5 ft. x 6 ft. The capacity is sufficient to handle the contents of an agitation tank in seven or eight charges in 24 hours, three hours being the time required for a complete cycle of loading, washing and discharging. Vacuum is maintained at 10-12 in., which is practically the best obtainable in slime filtration at the altitude of 10,000 ft., where the barometer normally stands at about 20 in.

Precipitation.—The filtered gold solution is clarified in two Johnson presses and flows to the zinc boxes for precipitation. The daily tonnage of solution precipitated is about 350 tons, or little more than the tonnage of ore treated. Occasionally, when the mill solution rises too high in value, several hundred tons may be precipitated in addition to the regular quantity.

The gold precipitate receives the usual acid treatment and is discharged to a pressure tank connected with a small filter press. The press cake is dried, roasted and ground in a small tube mill using pieces of shafting for the grinding medium. The object of the last step is to put the precipitate into as uniform condition as possible so that an accurate sample can be taken. Prior to the introduction of this practice a great deal of time was consumed in making settlement with the smelter on account of the variation of the sample taken for assay.

The Independence mill was erected at a cost of over a quarter of a million dollars. The preliminary experiments indicated that a fair profit could be made by the treatment of 10,000 tons per month, and it is interesting to know that the actual working costs have checked the estimates remarkably close. No official figures have yet been given by the company, so that details of costs and profits are not actually known to the public, but it is stated that operations are being conducted at a satisfactory profit.

The mill is equipped with duplicate pieces of machinery at those points where a break would interfere seriously with continuous operation. Cement floors, launders, and sumps effectually prevent the loss of valuable cyanide solution and contribute to the cleanliness of the plant. Crushing and concentrating in cyanide solution, which was one of the features concerning which sceptics expressed their doubts, has not given rise to any trouble, either by serious loss of solution or by ill effects on the workmen. Altogether the plant has been a success on which the company and staff are to be congratulated.

Following is a condensed statement of the flow of ore through the mill:

Ore from dump in 4-ton cars, to 1.

1—7½ D Gates gyratory, discharging to 2.

2—Grizzly; oversize to picking belt, thence to 3; undersize to 4.

3—5K G Gates gyratory, discharging to 4.

4—Incline belt conveyer to 5.

5—Mill storage tank; lime and a small quantity of mill solution are added to the discharge and the mixture elevated to bins, thence to 6.

6—Two sets of rolls; product elevated to bins, thence to 7.

7—Three 6 ft. Chilean mills, 12 mesh screen; discharge to 8.

8—Screw classifiers; sand to 9, slime to 14.

9—Distributor to 20 Card tables; high grade concentrate to smelter and low grade to chlorination mill; tailing to 10.

10—Sand tailing sump; centrifugal pump to cyanide department, 17. Coarsest sand reground in tube mill; product to 11.

11—Two classifying cones; underflow to 12, overflow to 13.

12—Two Card tables; concentrate shipped, tailing to 16.

13—Four Deister slime tables; concentrate shipped, tailing to 16.

14—Sixteen thickening cones; underflow to 15, overflow to 16.

15—Ten Deister tables and four vanners; concentrate shipped and tailing to 16.

16—Slime tailing sump; centrifugal pump to cyanide department, 17.

17—Cyanide mill; screw classifiers; sand to 18, slime to 19.

18—Sand leaching tanks; filtrate to 23, sand discharged.

19—Four settling tanks; overflow to mill solution or to 23; thickened pulp to 20.

20—Agitating tank for 10 hours; thence to second agitator for special chemical treatment for six hours, thence to 21.

21—Storage tank for filters, to 22.

22—Vacuum filters; gold solution to 23, slime to discharge.

23—Two Johnson clarifying presses, to 24.

24—Seven zinc boxes; barren solution to mill storage; precipitate to acid tank and filter press. Press cake dried, roasted, pulverised, sampled and shipped to smelter.

The Colburn Mill.—The latest development in the local treatment of ore at Cripple Creek is the decision of Mr. E. A. Colburn, principal owner of the Ajax mine, to erect a mill embodying the much discussed Clancy cyanide process. This will be the first commercial application of the Clancy principles and the outcome will be awaited with great interest. Metallurgists generally are favourably impressed with the principles of the process and earnestly hope that the first installation will successfully meet the prime requisite of making money.

The Colburn mill, as it is to be known, is being erected near the Ajax shaft, above the town of

Victor. Both mine and dump ore will be treated. The plant will comprise two principal buildings, one for crushing and sampling and the other for cyanide treatment. The excavations have been made, concrete foundations put in and it is expected that the plant will be in operation this summer, probably in June.

An aerial cableway will be installed for transporting ore from the dump to the sampler, while mine ore will be delivered from the shaft by electric tramway. The sampler will have a capacity of 100 tons in eight hours and will have to be run two shifts to supply the cyanide plant, the capacity of which will be 200 tons in 24 hours.

The sampler will be arranged for the separate treatment of leasers' and company ore. Coarse crushing will be done in a Symons gyratory, the product from which will be elevated and diverted to steel bins. One of the latter will be a scale hopper bin for leasers' ore and the other an ordinary bin for company use. From these bins the ore will be elevated to the roll and screen system. The sample will be taken by two Snyder samplers when the ore is reduced to ¼ mesh. The final roll product will be reduced to 12 mesh and elevated by belt conveyor to the cyanide plant. An exhaust dust collecting system will gather the dust from various parts of the sampler and conduct it to a settling chamber with hopper bottom, from which it can be drawn on to the conveyor belt and sent with the ore to the mill.

At the head of the cyanide mill will be three grinding and classifying units, each consisting of a storage bin for crushed ore, a classifier and a tube mill. The bins will be circular in form, built of steel, with bottoms of rock and cement sloping at an angle of 45°. The ore will flow from the bins into the classifiers, which will be of the spiral vane form. The slime will overflow to the agitators and the sand will be ground in tube mills. The tubes will be of a shorter type than customary, 5 ft. × 16 ft., and will have El Oro lining and screw feed. They will grind in closed circuit, their discharge being returned to the classifiers.

The air agitation will be of special construction, suited to the application of electrolysis as well as of agitation. The form will be circular, with cone bottom, and the ratio of height to diameter of the circular portion will be 1·1. The central air tube will be about half the height of the tank, so that the pulp in the upper half of the tank will be thoroughly agitated by the air rising from the central tube.

The agitators will be fitted with electrodes for the application of the electric current during agitation, both for extraction of the gold from the ore and for regeneration of the cyanide solution. The cathode will be in the form of iron sheets suspended in the pulp around the inner periphery of the tank, and the anode of fused ferric oxide suspended near the centre. There will be five of these tanks, which will be used for storage as well as for agitation.

The Moore vacuum filtration system will be used. On account of the location of the mill with respect to the town of Victor and adjacent mining property it will not be possible to discharge the slime tailing by sluicing. An automatic system of stacking the tailing on the dump is now being worked out whereby the tailing will be removed from the Moore tank by a screw conveyor and delivered into a hopper. From the hopper it will be carried by bucket tramway to the dump. It is planned to make the loading and dumping automatic and to have the bucket dump at a series of different points on successive trips.

The Merrill system of zinc dust precipitation will be installed and the gold refined at the mill.

The plans contemplate a mill as nearly automatic in operation as possible, with a minimum labour requirement. It is expected that two men for each of two shifts in the sampler and two for each of three shifts in the mill, in addition to the superintendent and chemist, will be sufficient. The mill design and construction is under the direction of S. A. Worcester and the metallurgical details are being worked out by A. W. Warwick."—*Metallurgical and Chemical Engineering*, IX., 4, p. 192, April, 1911. (J. A. W.)

TREATMENT OF AURIFEROUS SAND.—"A process for treating black sand, concentrate and the like has been patented by Carl F. Siegert, of Jeppes, Johannesburg. The invention consists in roasting the refractory material, or heating it to a dull red heat, and quenching it in a solution of sodium and ammonium chlorides to free the gold from oxides. The solution is then evaporated and sulphuric acid added to the comparatively dry mass. Water is subsequently added and the mass mixed. After the dilute acid has remained in contact with the ore for a suitable time it is drawn off and the mass is washed with water. It is claimed that this treatment will render the precious metal content free for recovery by amalgamation in the usual manner.

For each ton of material the inventor uses about 42 gal. of water and 10 lb. NaCl, to which is added 2 gal. of strong salammoniac solution made by dissolving 2·5 lb. to 3 lb. salammoniac in 100 gal. of water. This solution is used for the first treatment. For the acid wash he uses 1 lb. of sulphuric acid, more or less, as needed, and adds water after the acid has been poured on the mass."—*Metallurgical and Chemical Engineering*, April, 1911, p. 156. (J. A. W.)

MERCURY PRODUCTION.—"The price of mercury, which for some time has been maintained at £8 per bottle, has shown a decided upward tendency during the past month. It is quoted in London at £9 10s. to £10. The scarcity of the metal is indicated by the January statistics. In January the imports into England were only 54,197 lbs., whilst the exports reached the extraordinary figure of 750,125 lbs. The small quantities imported came from France and Italy. It is well to remember that Spain overshadows the mercury market, since it supplies a third of the output of the world. According to *L'Echo des Mines*, the figures are as follows:—

	Tonnes
Spain ...	1,000
Italy ..	700
California	683
Austro Hungary ..	609
Mexico	201
Russia ..	7
	3,200

London Mining Journal, April 5, 1911, p. 382. (A. R.)

THE WAIHI-PAEROA GOLD EXTRACTION CO., LTD.—"The Ohinemuri River, flowing into the Hauraki Gulf, New Zealand, receives the tailings from the large mills of Waihi, Karangahake and Waitekauri. Since it was proclaimed a sludge channel in 1895, millions of tons of silt have been deposited in it, the quantity at present amounting to about 550,000 tons per year. Of this, the larger particles have been laid down in that portion of the stream which traverses the flat land round about

Paeroa, and have so laden the banks and the bed, that where 20 feet of water once existed, the gleaming white of the quartz sand now rises above the level of the stream. A large proportion of these tailings date back to the early history of the field, when less perfect processes and equipments were in use, and hence the actual assay values amount to as much as 13s. per ton.

The Waihi-Paeroa Gold Extraction Company was formed in 1908 by Mr. P. N. Kingswell, now managing director, to treat this material. Their first plant, costing £16,000, was erected at that part of the river where the Waihi Co.'s dam had caused an extensive deposit. The treatment consisted of regrinding in four 16 ft. by 4 ft. tube-mills, and agitating the product in nine conical pneumatic tanks, followed by vacuum filtration. To obtain the sands, an air-lift dredge, capable of lifting 40 tons per hour, was employed, and a barge was thus loaded and towed by an oil launch to the elevators at the mill. This plant successfully treated 80 tons per day, and during 18 months' work extracted gold to the value of £14,000 from 32,000 tons of silt, an extraction, that is, of 8s. 7d. per ton. Then the great Ohinemuri flood came down the river, scouring out the bed, and completely removing the tailings from the vicinity of the works, and giving rise to the Silting Commission, which evoked so much general interest in the country. To continue work, the company had to remove their plant much further down the river, and it is now being re-erected, and added to, on the lower reaches of the river. The new mill will have a capacity of 500 tons per day, and in its reconstruction, every advantage is being taken of previous experience. For example, loss of gold was traced, after much trouble, to the presence of charcoal discharged from steam plants higher up the river. In the new mill, therefore, ten 6 ft. Union vanners will completely get rid of this, as well as separate out valueless river sand, before the silt is passed on to the tube mills, now numbering 29. Charcoal was also detected in the kiln-burnt lime employed, and this, too, is being remedied. Altogether, the residue is expected to decrease in value from over 4s. per ton, as it used to be, to below 2s. 3d., and the working costs and renewals from about 5s. per ton to 3s. 6d.

The new dredge will lift 80 tons of sand per hour, and will be used to fill four 65-ton barges, towed by a couple of oil launches. By means of the air-lift, about equal volumes of sand and water are raised, instead of a much greater proportion of water, as in the case of a centrifugal pump. The dredging appliances are actuated by a 30 b.h.p. Crossley kerosene engine."—R. M. AITKEN, Aust. I.M.E.—*The Mining and Engineering Review*, April 5, 1911, p. 277. (W. R. D.)

FINE CRUSHING WITH CONICAL MILL.—"The ore fed to the ball mill will not all pass a 2½ in. ring, yet this mill, installed for crushing to from 8 to 16 mesh, produced a product 33% of which passed 200 mesh, the oversize going to an 8 ft. diameter conical pebble mill and 66% of its product passed 200 mesh even though the pebble mill contained a little more than one-half of its full charge of pebbles.

An interesting experiment with a ball mill was made in the presence of J. M. Callow in crushing a Butte lead-zinc ore which had all passed a Blake crusher. Steel balls were used, the mill making 32 revolutions per minute with an initial feed at the rate of 30 tons per 24 hours and a ratio of water to ore of 2·67 to 1. For comparison, a sample of the crusher run of this ore was recrushed in seven successive stages with rolls and hand screens, screening out the fines and returning only the coarse to the rolls after each screening, and the following results were obtained by the two processes:

	Once through Hardinge Mill Per Cent.	Rolls 7 Returns Per Cent.
Held on 60 mesh - -	10·18	30·20
Held on 80 mesh - -	9·92	21 03
Held on 120 mesh - -	23·15	18·62
Through 120 mesh - -	40·32	21·34
Through 120 mesh as slime	16·83	8·81
	100·40	100·00

It will be noted that in the crushing with rolls an equivalent to seven screens was used, while in the conical mill the sizing was automatic and without any screen whatever. The latter only produced an excess of 8% as slime, which could have been materially lessened by the use of additional water or any one of three other adjustments to which the mill is adapted.

As to power and range of capacity of this smallest size of mill, the data given below are furnished by one of the largest iron companies in the world, which is testing the conical mill, grinding for concentration a very low-grade, hard, silicious, amorphous hematite: Mill used, 4½ ft. diameter conical ball mill; charge of balls, 3,000 lb.; revolutions per minute, 30; power, less than 15 h.p.; capacity per hour, 7,850 lb.: size of feed, crusher product passing 1 in.; size of product, all but 2% passed 8 mesh.

The above shows considerably more than 5 tons per horse-power day, and is fully three times greater capacity than could be realised for similar crushing with stamps, rolls, or Chilian mills, and for simplicity of operation and freedom from necessity of oversight, there is absolutely no comparison.

The Camulet Copper Co. is constructing three 8 ft. diameter Hardinge mills that will be belt driven. Driving such large machines as these with a belt is an innovation that might be considered from the mechanical standpoint."—*Mines and Minerals*, May, 1911, p. 604. (A. R.)

TUBE MILL PRACTICE.—"At Waihi the standard tube mills are about 19 ft. long, and they are all of the trunnion type; whilst at Broken Hill the roller type has been generally adopted. In the latter case, where the desire is to avoid sliming, the tube mills are 5 ft. in diameter by 10 ft. in length." (A. R.)

ESTIMATION OF ZINC IN COPPER ORES.—"The process consists of the separation of manganese by nitric acid and chlorate of potash; iron by ammonia; copper by potassium thiocyanide, and the zinc then being thrown out as a phosphate to be dissolved in a measured quantity of acid and titrated under standard conditions.

The ore is treated in a casserole with strong nitric acid which has been saturated with chlorate of potash, and the solution is evaporated to dryness without baking. Solid ammonia chloride is now introduced (about 10 gm.), and then about 25 cc. of hot water, and the assay kept warm until the mass is disintegrated. Excess of ammonia is then added and the solution is filtered. Boiling should be avoided, as there is a tendency to form insoluble basic compounds of zinc. The precipitate is well washed with hot water containing ammonium chloride and ammonia. If the precipitate is bulky it is advisable to assist the washing by bringing the precipitate back into the casserole with the wash bottle; a cone of platinum is of great value in the filter. The main lot of zinc and copper is now in the filtrate, but it is necessary to re-treat the precipitate. If the process has been carefully carried out it is only necessary to treat the precipitate once. The ammoniacal filtrate about 100 cc. is rendered just acid, with hydrochloric acid, boiled with a little sulphurous acid, and the copper is thrown out with a 10% solution of potassium thiocyanide and sodium sulphate is added drop by drop to the solution at about 60° C. If only a little copper is present the addition of a little cold water assists the formation of the precipitate. To the filtrate a few drops of methyl orange is added, and then 5 E. hydrochloric acid. The solution is now heated to boiling, and the zinc precipitated by the addition of ammonia phosphate, the precipitate being kept warm for a few minutes until the precipitate becomes granular. Care must be taken to avoid bumping.

The precipitate is then readily filtered off and is then treated with the filter paper in a beaker with 150 cc. of hot water and 5 cc. of strong hydrochloric acid and titrated at 60° C. with standard ferrocyanide as usual.

In dealing with ores that contain under 1% copper a very good alternative method is to take the ammoniacal filtrate from the iron, and after adding the required excess of acid, add a small crystal of sodium sulphide and titrate with the copper sulphide precipitate present. If the sodium is used in the form of crystal, the copper sulphide precipitate present settles out readily, whereas if a solution were used a brown coloration would result which would interfere with the results of the titration." F. E. CONNAH. — *Australian Mining Standard*, Aug. 24, 1910. (K. L. G.)

DEVELOPMENTS IN CYANIDE PRACTICE.—"Chemically the cyanide process has changed little since the Forrests and MacArthur completed their elaborate and comprehensive experiments. But mechanically the developments have been so great that it may be said that, were it otherwise, the cyanide process would to-day be only applicable to a few especially favourably adaptable ores.

While previously it was the cyanide millman's object in life to make the minimum amount of slime, the best practice to-day is to slime the entire product. Slime is generally considered the product which will pass a 200-mesh screen. No one machine has yet been devised which will economically reduce the ore, as it comes from the mine, to slime in one operation.

Eliminating the discussion of the preliminary crushers, the point where the slime first appears as a quantity to be dealt with is taken up in this article. This will be after crushing to a 16 or 20 mesh-screen, because when crushed this fine, almost every ore will make considerable slime. Various modifications of the hydraulic cone classifier have been used at this stage of the treatment to eliminate that material which has already attained the desired fineness. The latest machine for doing this, and one which is being very widely adopted, is the Dorr classifier. This machine can be set so that the discharge will fall into the feed-hopper of a tube mill, thus the amount of mill head required for hydraulic classification is reduced to a minimum. The machine requires no addition of clear water, as do the cone classifiers, thereby effecting a large saving in the water consumption. It makes a clean sand and a slime at one operation, is cheap, consumes little power, and requires no attendance. At the 600 ton mill of the Goldfield Consolidated there are 6 double-rake classifiers which discharge into the tube mill feed-hopper. Each machine handles 65 to 70 tons of battery sand per day, 16 mesh and under, and delivers to the tube mills a sand free from slime and containing the proper quantity of moisture (30%) for efficient tube mill work.

Tube milling presents one of the most interesting, most anomalous, most complicated, and perhaps least understood of any of the mechanics of the cyanide process. It is by no means mysterious; but the matter of testing and comparing various types of tube mills involves such complications and often such dissimilarity of conditions that the tube mill field is one of the broadest and most interesting for the technical investigator. The efficiency of the tube mill is the measure of the slime produced per horsepower consumed; but this imposes the most unfair relative conditions for judging the comparative merits of two tube mills in different localities. For example, tube mills at the Black Mountain mill in Sonora, Mexico, treat an ore which is soft and crushes almost like volcanic ash, while the Goldfield mills treat a hard quartzose andesite. A comparison of daily tonnage through these different mills is obviously both unfair and misleading, as one ore is soft and the other is hard, the one requiring a short mill and the other a long mill. Until some better method of classifying conditions and comparing results is devised, the comparison of types of tube mills, except when operated on the same ores at the same time, will always be open to discussion.

The size and proportions of tube mills are for the same reasons disputative subjects. With an ore which slimes easily, the necessary fine grinding may be accomplished in the first 10 or 12 feet of the mill; with a harder, tougher ore, a 22-foot mill may be required in order to give more pebbles sufficient time to work on the sand. The proper diameter of the mill is likewise a question. One of the most prominent manufacturers of tube mills says: "The small-diameter mills are very inefficient." As the results of a series of tests on mills of three different diameters, running side by side on the same ore and under the same conditions, an engineer of standing reports "the efficiency of a tube mill is inversely as its diameter." The speed of a tube mill is fixed within narrow limits. If run too slowly the pebbles simply slide on the lining, causing excessive wear and giving poor grinding effect. If the speed is too high the pebbles are held against the shell by centrifugal force and no grinding is accomplished.

An extensive series of tests conducted at El Oro, on three tube mills of the same make, but of different diameters and different lengths, led to the following conclusions:

'The efficiency increases proportionately to the amount of pebbles contained in the mill.'

'The efficiency increases with the coarseness of the sand fed to the mill.'

'The efficiency decreases proportionately to the rate of feed.'

In a paper on the 'Economics of Tube Milling' are given results of working tests on tube mills 5 ft diameter by 23 ft long made at the Colorado Springs plant of the United States Reduction and Refining Co. These tests demonstrated that the greatest grinding efficiency was obtained with these mills when the pebble charge was 19,000 lb. With a pebble charge of 25,000 lb. the amount of fine grinding was increased, but at the expense of a disproportionate increase in the horsepower required. With 19,000 lb. of pebbles the charge filled the tube to within a few inches above the centre line. The percentage of solution to ore in the tube mill pulp was varied, and it was found that both the amount of fines made and the amount of power required was affected. It was demonstrated that when the amount of solution in the pulp reached about 35% the best and most efficient work was done. This is in accord with other good tube mill practice and it is on account of this that the Dorr mechanical rake classifier is such a satisfactory machine to dewater and classify the pulp before it is fed to the tube mill.

The Hardinge conical mill is a new type of grinding mill, a competitor of the tube mill, which is attracting attention. This mill consists of a short cylindrical section with conical sections of different lengths at each end. The cylindrical section is of considerably larger diameter than the tube mill which it is designed to replace. The most important claims for this conical mill are that 'it has three times the capacity for the same power, requires one-fifth the pebbles and has less wear on the pebbles and lining.' Unfortunately this mill has not yet been tried extensively enough to provide accurate data, especially as to fine grinding, to thoroughly establish these claims. The most unique claim made for this mill is that it classifies its pulp and its pebbles, so that the larger pebbles work on the coarse sands and the smaller pebbles work on the fine sands. This is claimed to be a most efficient natural selection. (But the idea of natural phenomana in this machine working so beneficently in the right direction makes one stop and ponder over the almost human cussedness of some of the other machinery about the mill.) That the larger pebbles and the coarser sands tend to remain in that portion of the mill with the larger cross-section can be seen from a glass working model of the mill. It is undoubtedly true that a plain cylindrical tube mill would not give this sorting action when working dry, but it is also true that in a tube mill there is a certain amount of similar classification due to the action of the current of pulp through the tube. This is evidenced by the small worn pebbles which are found at the discharge end. There seems room for a reasonable difference of opinion as to the relative merits of having the finer sands worked on by smaller pebbles. But for grinding where 100-mesh is sufficiently fine the Hardinge mill has shown some astonishingly good results. It has not been tried out sufficiently for sliming to give the desired amount of data, and 100-mesh is not fine enough for the subsequent slime treatment.

In the Colorado tests referred to, the mills were started with a charge of only 3,000 lb. of pebbles and the required horsepower was 18·8. The power increased in almost exact proportion to the amount of the pebble charge, until the latter had reached 23,000 lb. which filled the mill much fuller than even poor judgment would dictate. It would seem natural that the power consumed should increase thus with the added pebble load revolved, and this would seem to apply to the conical as well as to the tube mill. So that in this respect they would seem to be on the same footing. With these Colorado tests showing an increasing efficiency with the addition of pebbles, from the initial charge of 3,000 lb. to a charge of 19,000 lb. it would appear that the smaller charge of the conical mill must have an efficiency per pound very much in excess of the tube mill in order to give a greater relative efficiency, and very much greater than might naturally be expected.

The product from the tube mill is generally not of a wholly uniform grade, especially if the machine is being pushed a little for tonnage. The result is that a portion of the product needs to be reground. Therefore a classification is necessary. If the pulp from the grinding mill is as thick as it should be for the best grinding effect, it emerges too thick for classification without the addition of further water. Therefore this product is classified, usually in hydraulic cone classifiers, the slime going direct to the slime-treating department and the sand being returned to the tube mill for regrinding. If the head is not too great the sand is most easily and economically elevated to the tube mill feed by means of the Frenier spiral pump, made of light sheet steel with sheet-steel discs forming the sides of the pump wheel. This wheel revolving in a hopper to which the coarse pulp is supplied, scoops up the pulp and forces it around through the spiral channel toward the centre. When it reaches the centre it emerges through the hollow trunnion, then through a flexible joint and thence to the delivery pipe and to the point of discharge While this pump is very satisfactory for handling fine pulp, it has its limitations. The head against which it will work varies from 14 ft. to 20 ft. depending upon the material handled and upon the altitude of the locality. Heavy coarse pulp is not a material to which this pump is adapted.

During the classification of the pulp subsequent to the tube milling the addition of classifying water has made the pulp too thin for cyaniding. The overflow from the classifiers also contains too much water, these two overflows are therefore brought together and are dewatered. The old style conical-bottom settling tank with its filling, settling, decanting, discharging, and flushing, and then a repetition of the cycle, is fast being relegated to abandoned practice. A continuous pulp thickener is displacing the old intermittent method. Present practice decrees that the pulp when submitted to the action of the cyanide solution shall have a consistency of about $1 : 1\frac{1}{2}$ or $1 : 2$. The Dorr continuous slime thickener has been developed for this particular service. The capacity of this type of thickener depends upon the nature and density of the slime feed, and the amount of moisture desired in the thickened discharge. A tank 35 feet diameter by 12 ft. deep, fed with as much thin slime as the tank would take and give a clear overflow, discharged 285 tons of thickened slime per day, containing 39%, or 111 tons, of dry solids. At the Liberty Bell mine, in Colorado, a tank 33 ft. diameter by 10 ft. deep, fed with a pulp containing 120 tons of solids to 660 tons of solution, discharged at the bottom a thickened slime containing 120 tons of solids to 288 tons of solution, or a $1 : 4{\cdot}2$ pulp. The vertical shaft with the plows made 4·8 revolutions per hour and required ·14 h.p. Three tanks equipped with the thickening mechanism had the capacity of five of the same tanks used in the old intermittent method and made a considerable saving in the amount of labour required. At the Goldfield mill, thickeners were installed to handle the pulp from the agitators which contained $2\frac{1}{2}$ tons of solution to 1 ton of dry slime and they discharged a thickened slime containing only 35% moisture. This is an example of the successful working of the machine over a wide range of conditions, delivering a pulp containing from 35% to 61% moisture. Except in the case of an extremely argillaceous slime, a long settling period is not necessary, and the old intermittent method ties up capital in equipment and in product without adequate return.

The Callow tank has justly come into considerable repute as a dewatering device for concentrating mills. This tank consists of an inverted conical shell of steel with a gooseneck discharge spout at the apex. The pulp is fed into the centre of the now upturned base of the cone through a small cylindrical funnel which is only long enough to discharge the pulp below the top surface of the solution in the cone. The slime settles to the apex and flows continuously through the gooseneck while the clear water containing practically no slime overflows the rim into a launder connected with the water storage tanks. At the Goldfield combination mill during a test of concentrating machines the following data were obtained from the Callow tank: Flow to Callow tank, pounds per minute, 137·7; dry solids to Callow tank, pounds per minute, 16·36; water to Callow tank, pounds per minute, 115·5; overflow in clear water in per cent., 48·15; per cent. of solids in discharge, 22·62.

The pulp having the desired fineness of 200 and the desired consistency of about 1½ to 1 or 2 to 1, it is ready to be cyanided. One of the two devices which has made economical slime treatment possible is the Pachuca tank. It consists of a plain cylindrical steel tank 13 ft. diameter by 55 ft. high. Inside the cylinder, extending from within about 30 in. of the top to within about 18 in. of the bottom, is a tube 18 in. in diameter, open at both ends. At the bottom of this tube is a small pipe through which is furnished compressed air. The tank is filled with pulp to the top of the tube or within 30 in. of the top of the tank. Compressed air at a pressure of 22 to 25 lbs. per square inch is admitted at the lower end of the 18 in. tube, and by expanding carries the pulp up through the tube where it is discharged into the tank proper. The operation is continuous so long as the compressed air is supplied, in fact it is the old principle of the air lift and ejector applied in a rather novel way. The result of this operation is a perfect agitation during which every particle of this slime is afforded the most intimate contact with the cyanide solution. The advantages or disadvantages of aeration as affecting the cyanide itself are a somewhat mooted question, but it has been found that by increasing the height of these tanks and thus increasing the aeration the cyanide consumption is lower than in tanks only moderately high, and very much lower than in low tanks. A tank 13 ft. diameter by 55 ft. high will hold a charge of about 100 tons of dry slime in a 1½ to 1 pulp. The free air required varies from 4 to 6 cub. ft. per minute at an expenditure of about 2 h.p. The time of the agitation depends upon the kind of ore and the economic point of extraction, because beyond a certain point it does not pay to increase the extraction even though it is mechanically and chemically possible.

The second important improvement in the mechanics of the cyanide process is the perfection of the vacuum filter. Without it most of the low-grade ores would still be impossible of profitable treatment. The first type to come into successful practice was the leaf filter, as exemplified in the Butters. This filter consists of a number of leaves made of thin wooden frames covered on both sides with cocoa matting and canvas, the space between the canvas sides being connected with a vacuum system by piping through the top of the frame. These leaves are suspended in a tank into which the cyanided pulp is discharged. The problem now is to extract from this pulp all the gold bearing cyanide solution. When the tank is filled with pulp so that the filter leaves are immersed the vacuum pumps are started and the gold solution is sucked through the filtering canvas and matting and is discharged into a gold solution storage tank. The solid particles of the pulp adhere to the sides of the leaves forming a cake of slime. This process continues until the cake has reached a thickness beyond which it is not economical to continue. This thickness is from ½ to 1½ in. The pulp remaining in the tank is then drawn off and the tank filled with a weak solution of cyanide. During these operations a sufficient vacuum is maintained to prevent the cake from falling off the leaves. The weak solution is now drawn through the cake and is discharged into weak solution storage tanks and the filter tank is filled with wash water. After the cake is thoroughly washed and the wash water remaining in the tank drawn off, the slime cake which is now more or less barren is discharged to the tailing dump. This tailing carries ordinarily about 30% wash water, whereas had the cake been discharged at the point where the drawing off of the gold solution was completed, this 30% moisture would have been the gold solution. Thus the advantage of washing the cake is clear.

The Moore filter process is similar to the Butters, except that instead of the filter leaves remaining in the tank with the filling and drawing off of the pulp and the various solutions, the Moore leaves are arranged in nests which are lifted from one tank compartment to another containing, respectively, pulp, wash solution, and wash water.

There are several modifications of the vacuum filter, but the one which at present is making greatest strides in application, due both to its efficiency and remarkable low first cost, is the Oliver continuous vacuum filter. The intermittent feature of the other processes is of course objectionable.

The Oliver filter, whose operation is continuous, has given astonishingly good results, is simple to operate, and requires probably smaller initial investment than any other. It consists of a large drum 11 ft. 6 in. diameter by 8 ft. wide, with open heads, mounted on a light cast iron spider, and is about two thirds submerged in a wooden tank into which the pulp flows continuously. The drum makes a complete revolution in 5 minutes. It is covered with canvas and other filtering material supported by light framework within the drum. The circumference is divided into sections which are connected by pipes which lead through the hollow trunnions of the drum, by which it is supported and revolved, to the various pumps. Through that portion of the drum's surface which is submerged the gold solution is drawn by the vacuum pump while the slime cakes on the canvas cover of the drum just as on the leaves of other vacuum filters. As the drum revolves and a section emerges from the pulp the excess solution in the cake is drawn through and as this section reaches a point near the top of the path of revolution it encounters a spray of wash solution. At this point an ingenious automatic valve prevents the wash solution from mingling with the gold solution, each solution going to separate storage tanks. The section of the drum passes from the weak solution wash to another spray of wash water and then just before it is about to be again submerged in the tank full of pulp a scraper fixed against the drum cleans off the now practically barren cake of tailing which drops into a hopper and thence goes to the tailing stacker. With the Oliver filter, the solutions are kept as separate as in any of the leaf filters, the tailing is as clean and carries as little moisture as in the other types, and the tailing is removed from the filter by a mechanical scraper.

The precipitation of the gold from the solution by means of zinc is the *sine qua non* of the process. Where formerly zinc shavings were used, now zinc dust is employed with very different equipment and results. At one time the precipitating room contained a series of boxes filled with zinc shavings through which the solution flowed. Now the zinc is fed in the form of dust to the gold cyanide solution before it is pumped to the precipitating plant. During its course through the pump it becomes intimately mixed and the pipe line is made sufficiently long to furnish time for precipitation to become complete while the emulsion is travelling through. The emulsion now consists of a barren cyanide solution in which the gold precipitate is in suspension, it is therefore delivered to filter presses from which the barren solution flows continuously while the precipitate is collected in the press in the form of a cake. At regular intervals the filter presses are opened and the cakes removed, melted down, and refined. By this method the gold precipitate is always locked up in the presses where it is perfectly safe. This filter press consists of a series of triangular cast iron frames about 3 ft. on the edge by 1 in. thick, supported on horizontal guides and clamped together by means of long longitudinal rods. Between these cast-iron frames are sheets of filtering canvas through which the solutions pass and between which the cake of gold precipitate accumulates.—Percy E. Barbour.—*Mines and Minerals*, May, 1911, p. 597.

A Laboratory for Employes.—"In the cyanide plant at Goldfield, Nev., the Florence Goldfield Mining Company has equipped an experimental laboratory for the use of employes. The primary object is to afford proper facilities for men who wish to make experiments so that they will not make such investigations in the large mill. The laboratory stimulates interest in the work, attracts the best millmen and raises the standard of the employes. Many experienced men go to work in the Florence mill so as to be able to do experimental work at small expense.

Experiments may be made at the expense of the company at any time that the employe is not on duty. Any determination of value is at the disposal of the company, under the terms of using the laboratory.

The laboratory is provided with vacuum and compressed-air pipes, glass jars for use as leaching vats, a fine-grinding machine, glass Pachuca vats, small leaf-filters, facilities for zinc shaving, zinc dust and electrolytic precipitation of gold and silver as well as apparatus for making amalgamation tests."—*Engineering and Mining Journal*, April 1st, 1911, p. 653. (M. T. M.)

Milling and Treatment at the Wahii Mine.—"The ore is weighed before passing the rock crushers, and samples taken daily for moisture estimation as the broken ore passes to the ore bins. The moisture varies from 4 % to 6 %.

Sampling.—The mill head sampler is taken after stamping and tube milling. Plate tail and vanner tail samples are taken, also sand and slime head and tail. Daily assays are made on all these samples. All sampling is done by hand, and check samplers are put on from time to time.

Rock-Breaking.—Gates and Heclon breakers are in use. The ore first passes a grizzly, spaced 1½ in. coarse to a No. 5 Gates or "S" Heclon, delivering to a second grizzly, the oversize falling into a No. 3 Gates. The ore is reduced to a 2 in. ring size. A continuous oil feed is supplied to the inner and outer sleeves of the breaker, and the oil is pumped through a small filter-press before returning to the breakers. By means of an exhaust fan, all dust is drawn from the breakers, and passes to a settling chamber.

The total cost for breaking and elevating is 4·5d. per ton.

From the breakers the ore is elevated by either a belt or a chain bucket elevator, and distributed to the bins by belt conveyors or by trucking.

Stamping.—The running weight of the stampers varies from 900 to 1,250 lb.; blows per minute, 102 to 105; drop 7 to 8 in. The screen mostly used is 10-mesh (aperature, 1·78 mm.), resulting in a daily stamp duty of about 5·6 tons.

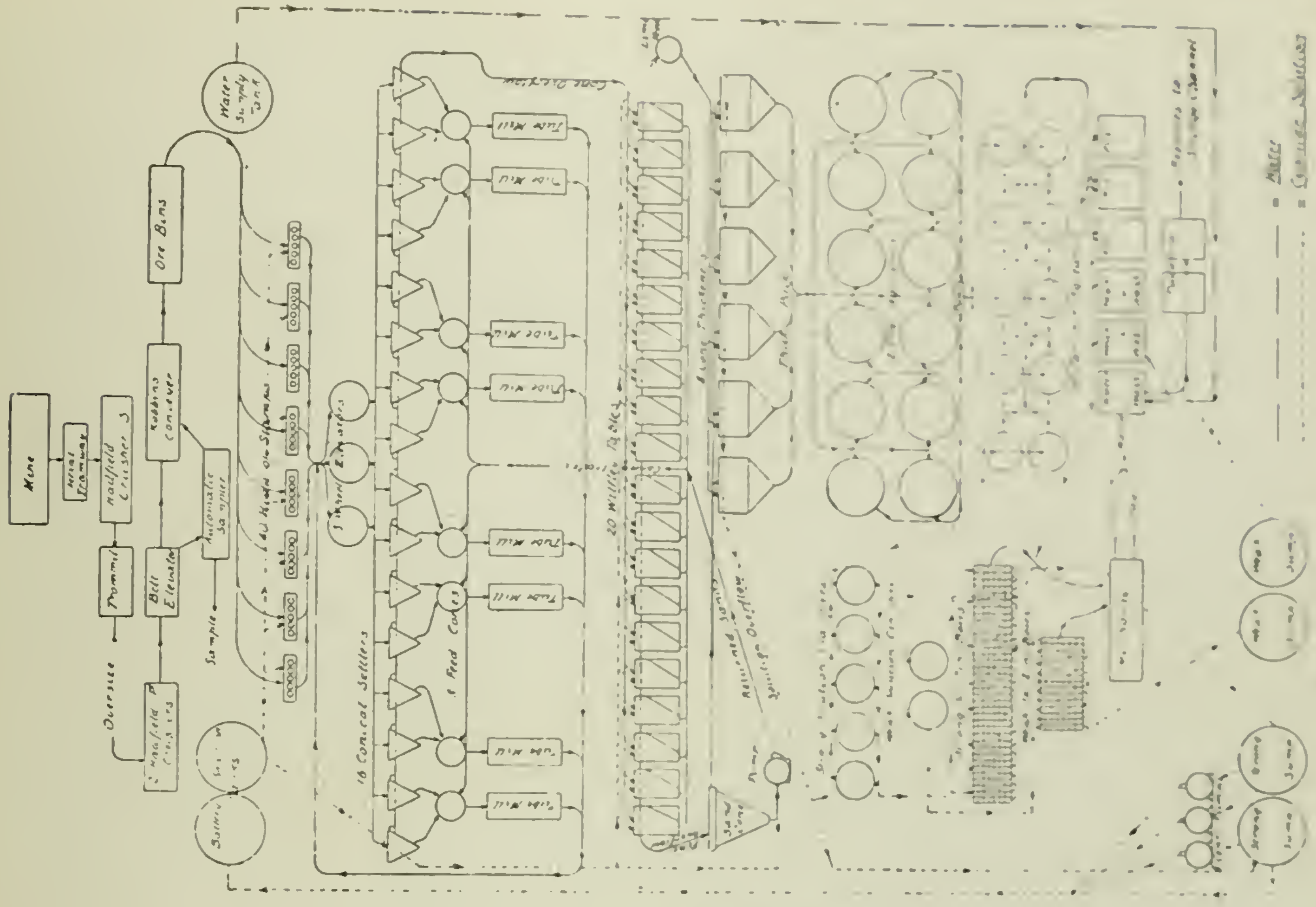
From 7 to 9 tons of water are used per ton of ore, and this is returned to the stamps after being cleared of sand and slime. The water is kept alkaline by the lime which is added to accelerate slime settlement.

Tube Milling.—Tube mills have proved by far the most efficient and economical grinding machines for Waihi ore. Those in use at the Waihi mills are 18 ft. long, 4 ft. 9 in. diameter (inside shell measurement). Various methods of lining the mills have been tried, and the most satisfactory is a cast-iron ribbed liner. The ribs are 4 in. wide, spaced 15 in. centres, and when new standing 3¾ in. above the body of the liner. When the lifting side of the rib becomes much rounded the liner is reversed. The body of the liner is 1½ in. thick. These liners promise a life of at least 18 months, and represent a cost of ·52d. per ton of stamp output. They are bolted to the mill, and can be taken out and a new set put in by two men in 16 hours. The pulp discharges through the trunnion. This is covered inside the mill by a cast-iron grating. The mills run at 25 to 26 revolutions, and require about 55 h.p. when loaded with flints to about 2 in. above centre of trunnion. This height of flints give the best grinding, and represents a flint load of about 5·5 tons. Flints are put into each mill daily in order to keep the load (and, therefore, the grinding) as even as possible. The consumption of flints is 2·6 lb. per ton of stamp output, representing a cost of ·86d. per ton. The pulp from the distributing pan is thickened in a V box at the head of each mill. This box is 3 ft. by 3 ft. by 3 ft. 4 in. deep, and discharges through a 1⅜ in. pipe into a 4 in. cast-iron bend, the outlet end of which passes into the trunnion. Flints are fed through this bend while the mill is running, thus avoiding any loss of running time.

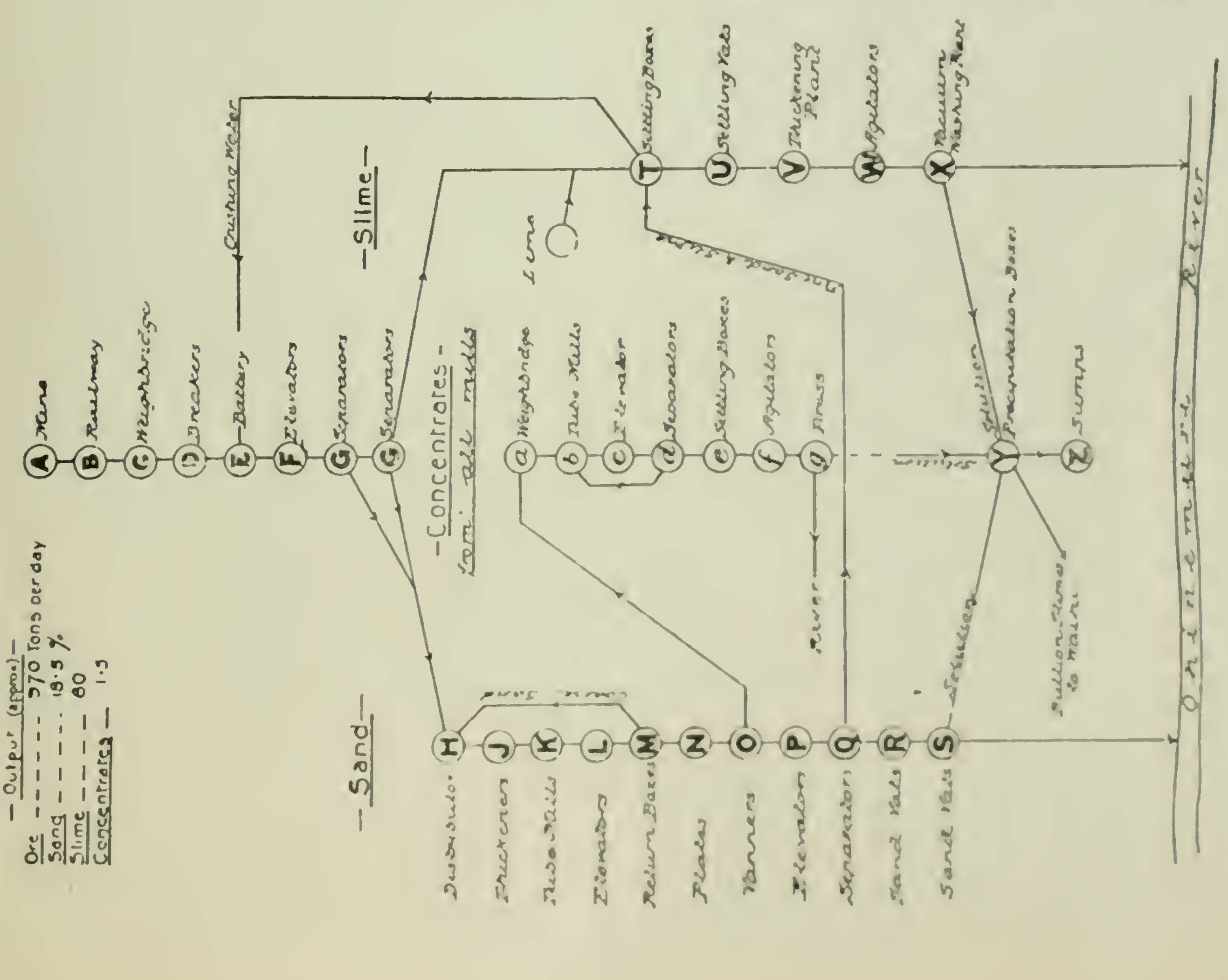
The proportion of water to ore in the mill is kept about 1 to 1. About 110 to 120 tons of stamp output (10 mesh) per day is at present the maximum work of a mill. The total amount of sand passing through a mill in 24 hours is 280 to 340 tons, owing to the large amount returned from the sizing boxes. The overflow from the sizing boxes passes to a V-shaped distributing tank and thence to the plates.

It is sometimes said that it would be more profitable to cut out amalgamation and rely on the cyanide plant to save the gold. No doubt the cyanide plant would do so; but, with the necessary amalgamation plant in existence, and operating at such a low cost (slightly over 1d. per ton of ore), it is considered better to continue amalgamation and avoid the accumulation of very rich concentrate in all wheel pits and launders.

Concentration and Cyaniding of Concentrate.—Union vanners and Wilfley tables are both used,



Waihi Grand Junction Co.—Flow Sheet.



Waihi G. M. Co.—Flow Sheet.

The former give a lower tail value (especially on the silver), but the latter have a much greater capacity, and, as close concentration is not required, they are preferable, being cheaper in first cost and more economical to run.

Concentration after tube mill grinding is not usually considered good practice; but, with the Waihi ore, it is found that, provided the coarser portion of the sulphides is separated by the vanners, it does not matter if the very fine sulphides escape to the cyanide plant.

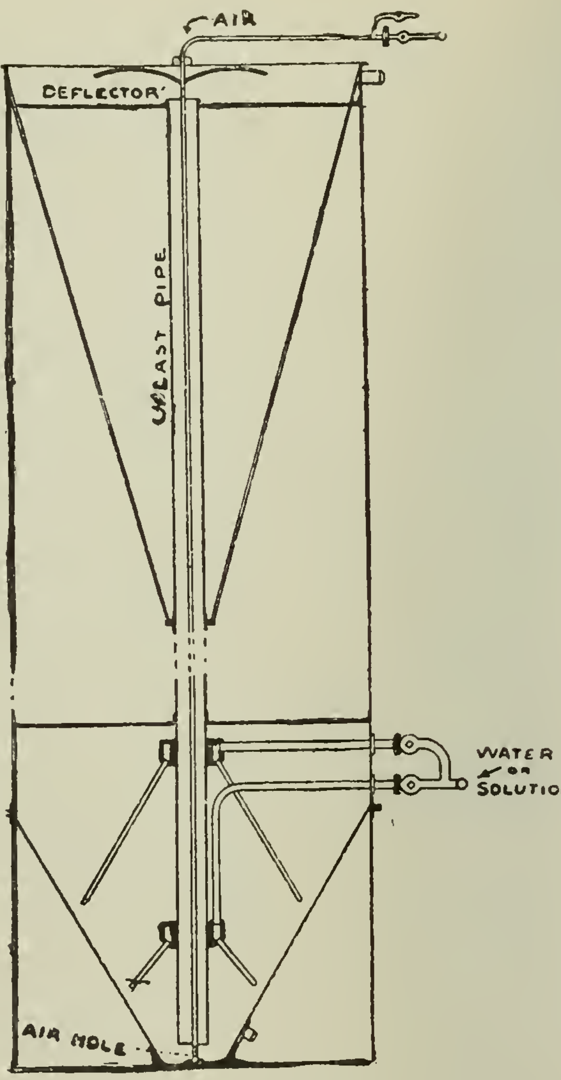
The average percentage of concentrates produced is 1·5, and an analysis shows as follows:—

	Per cent.
Sulphur	35·00
Iron	32·43
Copper	·05
Arsenic	·10
Lead	·32
Zinc	2·05
Manganese	1·00
Lime	·85
Magnesia	trace
Silica	27·65
Loss, etc.	·55
	100·00

The concentrate from each mill is sent to the concentrate treatment plant daily. It is shovelled from the collecting boxes into tubs holding about 600 lb. weight (wet), and kept under water until delivered at the treatment plant. The average recovery by concentration is gold, 22%; silver, 28%. The concentrate from the three mills is treated together, and a rough valuation of the output from each mill is obtained by bore sampling each tub. This valuation is required in order to allocate to each mill its proportion of the bullion recovered. The concentrate treatment plant is situated at Victoria mill. On arrival at the plant the output of each mill is weighed. The monthly total is about 500 tons. The grinding is accomplished in two tube mills—one 18 ft. × 4 ft. 9 in., and one 19 ft. 6 in. × 4 ft.—and the concentrate is sluiced from tubs into the head boxes of the mills, slaked slime being added in the proportion to 10 lb. per ton. From the mills the pulp is elevated to a V sizing box, 3 ft. 2 in. × 3 ft. 2 in. × 6 ft. deep, the sides being vertical for the first 2 ft. 6 in. The coarse delivers through a ⅞ in. nozzle and returns to the mills, and the fine overflow passes to the thickening boxes, by which moisture is reduced to ·7 water to 1 of dry concentrate. The concentrate is giving from 8 to 10 days' air agitation in 23 conical-bottom tanks, 16 ft × 6 ft., which are worked in series. The solution is kept at an average strength of ·45 KCy. The sodium salt is used, the average consumption being 16 lb. per ton, which represents ·25 lb. per ton of original ore The solution is separated by filter pressing. The bullion is precipitated by the zinc filament method.

Cyaniding of Slime.—The overflow from the separating cone runs to a nest of V thickening boxes, being mixed on the way with milk of lime in the proportion of 2 lb. to 3 lb. lime per ton of ore, representing a cost of ·5d. per ton. The lime is freshly slaked and fed into a grinding pan, the overflow passing to the slime launder. The thickening boxes are 5 ft. × 5 ft. × 5 ft. deep, and each box can deal with about 10 tons (dry weight) of slime per day.

The overflow from the last row of boxes is clear, and returns to the stamps or battery water reservoir. The thickened slime flows from the apex of the box to collecting or settling tanks. These are built of steel, 32 ft. diameter × 14 ft. deep, one being required for each 100 tons of slime per day. By means of overhead gear a stirring arm is rotated slowly for a few minutes every hour; the clear water overflowing passes to the reservoir. The thickened slime is drawn from the settlers at intervals and sent to the vacuum thickening plant, where the moisture is reduced to about 30%. Two methods of working the vacuum thickening process are in use. In one case the frames are lifted from the loading tank and discharged over a disintegrator, and in the other the frames are stationary, the slime cake being dropped off into the pulp in the loading tank, which has a V bottom, terminating in a goose-neck discharge pipe, 24 in. diameter. When sufficient thick slime has accumulated in the tank a valve is opened, and the slime, under a head of about 14 ft., is forced into a disintegrator The latter method is somewhat the cheaper to operate. In either case one man per shift can attend to an output of at least 300 tons per day. The filter frames are constructed of light, closely corrugated sheet iron, attached to a wooden head-piece, and are 16 ft. long × 4 ft. 6 in. deep. A ¾ in. pipe—slotted to receive the lower edge of the corrugated sheet—is fitted round the ends and bottom. One end is closed, and the other passes through the woo len head-piece, and is coupled to a flexible hose attached to a 3 in. pipe, common to all the frames. The frames are covered with canvas, which is caulked to the head-piece. Any frame can be cut out of action by closing a cock on the suction pipe. A sight-glass is fitted to the suction pipe on each frame, by



Section of B and M Tank (Waihi Co.)

means of which the workman can see if the solution is coming through clear. From 10 to 12 frames are coupled to two 'H' iron girders to form a 'basket.'

The loading or pulp tanks are designed to give easy clearance to the basket, and to prevent fine sand accumulating on the bottom the slime is kept agitated by compressed air during the time the basket is out of the tank. With a vacuum of 20 in. to 24 in., a slime cake 1½ in. to 2 in. thick is formed in about 90 minutes. The basket is lifted from the loading tank and carried to the disintegrator by either hydraulic or electric power, and 80% of the slime cake falls off the filters as soon as the vacuum is broken, the remainder being easily detached by hand.

Cyanide solution is run into the disintegrator in such proportion that the mixture contains 1 of dry slime to 1·3 or 1·5 of solution showing 1·2% KCy. The pulp is agitated for 5 days, and the most economical agitators in use are tall conical-bottom tanks with air agitation. The most recent are constructed of ferro-concrete, 55 ft. high and 13 ft. inside diameter. The cost of these is about £240, and is less than for similar-sized steel tanks. No centre pipe is used, slightly better results having been obtained without it. The air is introduced through a small pipe at the bottom of the cone. About 7 h.p. per tank is absorbed in agitating and circulating the pulp from one tank to another. The extraction of the gold is practically complete in 24 to 30 hours, but a profitable amount of silver is extracted up to 5 days. The consumption of sodium cyanide is 1·66 lb. per ton of slime.

From the agitators the slime flows to the vacuum washing plant, in which the filters are similar to those previously described. The loading time is 45 to 60 minutes, and with a vacuum of 20 to 24 in. a 1½ in. cake is formed. The basket is then transferred to the wash tank, and displacement of dissolved values is usually complete in the same time as was occupied in forming the cake. The residue assays—gold, 13 to 14 gr.; silver, 12 to 14 dwt.; representing an extraction of—gold, 89%; silver, 71%. In the washing plant each filter frame can deal with 6 tons (dry weight), in 24 hours: this represents about 83 dry lb. per square foot of filtering area. The total time occupied in forming a cake, transferring to wash tank, washing, transferring to discharge hopper, discharging and returning to pulp tank, is about 2½ hours. When the vacuum is broken the slime falls from the cloths quite readily, unless the cloths are very old. An ordinary life for a cloth is 6 to 9 months on the washing plant; they then go to the thickening plant for a further period of 3 to 6 months. Every 3 or 4 months it is necessary to wash the cloths in a weak solution of hydrochloric acid to remove the lime deposit, which gradually chokes the canvas. A double-ended vacuum pump is used, one end operating each basket of frames. The solution of washes pass through the pump to a clarifying and then to a storage tank, for delivery to the precipitators.

The power required for a plant to handle 800 tons a day is as follows:

	Horse-power
5 vacuum pumps at 11 h.p. ...	55
1 air compressor ..	40
3 cranes (electric) at 5 h.p.	15
2 solution pumps at 3 h.p.	6
	116

The total cost of clean up of precipitators, melting, and plant is—

	Per ton
Labour	2·43d.
Power	1·16d.
Stores (including lighting, filter cloth, oil, waste, packing, hose, caulking rope, and sundry) ...	·67d.
	4·26d.

Labour is paid at from 8s. to 10s. per shift of 8 hours. Power (gas) is calculated at £10 per b.h.p. per annum. About 80% of the ore is treated as slime.

Precipitation and Melting.—The zinc filament method is used, and the precipitation boxes have 8 compartments, each 17 in. × 17 in. × 26 in. The precipitation room is floored with concrete and surrounded with a concrete wall 18 in. high, so that, in the event of a fire, the gold slime would not escape beyond the room. A clean up is made each week at each mill, and all new zinc is dipped into lead acetate solution. No difficulty is ever experienced in obtaining good precipitation, provided the solution entering the boxes is kept free from ore slime.

The bullion slime from each mill is brought into one melt-house. This is situated at the Waihi mill, and the melt-house staff carry out the clean up at each mill. The coarsest of the zinc slime is treated with sulphuric acid.

Surplus moisture is taken out on a vacuum vat, and the wet slime is mixed with 20% to 25% of borax glass and 10% soda ash. A small set of rolls is used to mix the flux and slime, which is then transferred to light iron trays holding about 100 lb., and these are put into a drying oven. The dried slime cakes firmly and is roughly broken to a convenient size for melting, which is done either in No. 200 plumbago crucibles (gas fired) or on a cupellation test. For the last 11 months all melting has been done on the cupellation tests with coal fuel, but at present experiments are being made with pot melting by gas fuel, which promises considerable economy. Great difficulty was found in making tests that would stand, first, the corrosive action of the slag during melting, and, next, the action of the litharge during cupellation. A concrete shell with mabor filling gave the best results. The gas for pot melting is conveyed from the mine producer gas plant in a 3 in. pipe, and the cost at the melt house is about 2d. per burner hour (1,000 cub. ft.). Two burners are necessary for each No. 200 crucible, and the slime can be melted at the rate of 40 lb. per hour. The slag is ladled from time to time, and the bullion is finally poured into 1,000 oz. bars, which run about 940 fine in bullion. These bars are cupelled down to about 1% base—mostly copper—and cast into slabs 8 in. × 10 in. × ¾ in., which are weighed, sampled, and sent to the refinery department. About 13,000 lb. of bullion slime is handled each month, producing about 125,000 oz., the average value being about 12s. per oz. The bars from the amalgam are refined with the cyanide bullion. The output of each mill and the concentrate plant is kept separate until sent to the refinery. All slags, litharge, and dross are run down in a 20 in. round water-jacketed blast furnace, the lead bars being used for cupelling the next month's bullion, and the slag is crushed and passed over a Union vanner for recovery of metallics. The final slag tail usually assays from 15s. to 30s. per ton.

The gases from the cupels and blast furnace pass through long settling flues, and finally to condensing chambers, before escaping

Dore Refining.—For several years the sulphuric acid process was in use, but was discarded two years ago in favour of the present electrolytic method. The slabs as received from the melting department weigh about 200 oz., and run about—gold, 12%; silver, 87%. The stationary cathode type of cell is used, and the output of the 30 cells in use is 5,000 oz. to 6,000 oz. in two shifts (16 hours). The current is supplied by a 17·5 k.w. generator. Each cell requires 90 amp. at 3 to 4 volts. The gold sludge from the cells is boiled for from 2 to 3 hours in a cast-iron kettle with strong sulphuric acid, and, when washed and melted, runs 996 to 999 fine. Both the gold and silver are melted in oil-fired furnaces. Waihi bullion contains up to ·5% selenium, and special precautions have to be taken to free the gold from selenium, also to remove every trace of lead from the refined gold. Less than 1 part of lead in 10,000 of gold will render the gold too brittle for coinage. The brittleness only becomes noticeable after alloying with copper. The cost of refining is about ·36d. per oz. of dore.

Summary.—At present the ore averages—gold 9 dwt., silver 3½ oz. per ton; and the average residue assay is—gold 20 gr., silver 17·5 dwt. per ton, representing a recovery of—gold 91·2%, silver 75%.

This extraction can be improved slightly, chiefly in the direction of finer grinding, but it is necessary to proceed very carefully, as it is an easy matter to overstep the economic limit of grinding, and so obtain extraction at a sacrifice of cost. The main desideratum in the successful treatment of Waihi ore is fine grinding.

The total milling and treatment cost is now 5s. 4d. per ton, made up as follows:—

	s.	d.
Breaking..	0	5·34
Stamping...	0	10·66
Tube milling	0	4·62
Extraction	2	8·00
Power	0	11·38
Total	5	4·00

The average consumption of sodium cyanide is 1·913 lb. per ton, and represents a cost of 1s. 5·87d. per ton.

The average consumption of zinc is ·513 lb. per ton, and represents a cost of 1·632d. per ton.

Power to operate the mills is supplied by water, steam, and producer gas. Steam power costs about £18 to £20, and gas power about £10 per b.h.p. per annum. A grand total of about 2,800 b.h.p. is absorbed by the three mills and treatment plants, not including the large amount used at the mine for pumping, air compressing, and winding, the demands for which are increasing very rapidly with depth.

A hydro-electric power scheme to deliver 6,000 b.h.p. is now being installed. The generating station is situated 51 miles from Waihi, and, when completed the mills will be operated entirely by water and electric power. All repairs and renewals and foundry work required for the company's operations are turned out in the Waihi and Victoria mill workshops."—E. G. BANKS.—*Mining and Engineering Review*, March 6, 1911, p. 240. (A. R.)

MINING.

SIGNALLING FROM CAGES IN MOTION.—"A new apparatus, simple of construction and easily workable, has been thoroughly tested—from May, 1909, to April, 1910—at the testing shaft of the Prussian Mining Department at Louisenthal, and gives complete satisfaction. The shaft is 420 metres deep, the usual ascending or descending speed of the cages is 6 m. per second, and for the testing purposes this speed was increased to 7 m. per second; but the apparatus, a detailed description of which is given in *Glückauf* (January 28, 1911), has proved absolutely reliable, and signals can be given at any time from the moving cages. The bridging over of plats and intermediate levels by the contact lines does not present any difficulty. The apparatus is also provided with a telphone, which, by the application of a very simple device, can, in cases of emergency, be used for telephoning from the moving cages."—*London Mining Journal*, March 4, 1911, p. 230. (A. R.)

COST OF WINNING COAL.—"The *Echo des Mines* in a recent issue contains an article on the cost of winning coal in the chief European producing centres, from which the following data are taken: France is one of the countries where the expense of raising coal is highest. The following figures show the average cost per ton of coal raised in the collieries of the various countries:—

	Francs.
Belgium	16·14
France	15·95
Russia	12·90
Germany	12·68
Spain	12·66
England	11·08
Austria-Hungary ...	10·95

During the past two decades the price of mining coal has much increased, as a glance at the schedule below exhibits:—

	1886.		1907.	
	s.	d.	s.	d.
England	4	11	8	5
Wales	5	2	11	8
Westphalia ...	4	7	9	6
France	8	11	12	0
Belgium	6	7	13	7

One of the reasons for this augmentation is due to the diminishing daily output of the coal-cutter, as the subjoined figures will show:—

	1899 Tons	1908 Tons
England	311	279
Germany	264	246
France	211	189
Belgium	173	160

'Salaries have been also continuously increasing, and the many social changes for the benefit of the employees together with some legislative burdens, have tended to swell the total. Owing, chiefly, to this expension of the cost of the personnel, the difficulties of exploiting coal mines have been so enhanced as to diminish profits to a more or less serious extent.'"—*South African Mining Review*, March, 1911, p. 56. (A. R.)

REFRIGERATION IN MINES.—"In order to obviate the many economic and sanitary disadvantages connected with the increasing height of the temperature in depth below 1,000 m. or 1,200 m., Messrs. Rosenbeck and Rath, mining engineers, advocate, in a recent issue of *Glückauf*, the installation of freezing apparatus similar to those which have been so successfully introduced, especially in tropic climes, or hospitals, slaughter-houses, theatres, market halls, etc. As specially suitable for erection underground, they recommend evaporation plants with compression pumps, the most suitable freezing agent being sulphurous acid. The plant would have to be erected in a specially constructed cooling chamber on the level which it is to serve. From this chamber the cold air would be conducted to the several working faces by a network of pipes, which would, by passing through drives and crossents, cool the air in them as well. Although the cost of an installation sufficiently powerful to cool a mine section worked by shifts of sixty men each would amount to about M. 15,500. and the working costs, inclusive of interest and sinking fund, to M.23,080 per annum. The authors arrive at the conclusion—which they substantiate by a series of careful calculations—that, quite apart from its sanitary advantages, the economic results achieved by such an installation in increasing the working capacity of the men, etc., would not only fully cover all expenses but leave a very handsome profit."—*London Mining Journal*, March 18, 1911, p. 280. (A. R.)

A STANDARD OF ADEQUATE VENTILATION.—"The proper ventilation of mines has an important economic aspect, since it affects to a large extent the efficiency of the workmen, and, consequently, the cost of mineral production. The author dealt with (1) the quantity standard—that is, the anemometer measurement of the air; and (2) the quality standard, which is determined by analysis. The law in New Zealand and Queensland lays down a minimum of 100 cub. ft. per man per minute, whilst in Transvaal and Victoria the amount is 70 cub. ft. A quality test has been fixed in Victoria, Western Australia and the Transvaal, based on the recommendations of the British Factory Ventilation Committee, which recommends that the quantity of carbon dioxide shall not exceed 20 parts by volume in 10,000 parts of air. A less strict regulation, however, is in vogue for British metalliferous mines. Mr. Reed summed up his remarks as follows:—

1. That the volume constituting an adequate amount of air, within the meaning of the law, is not ascertainable by the rule of thumb formula of a quantity standard, but may only be found from the analysis and temperature of the mine air.
2. That when mine air is found by analysis to contain less than 19% by volume of oxygen, or more than 1¼% of carbon dioxide, or when the lights burn dimly, men should be withdrawn from such vitiated air; but it is not suggested that these proportions constitute a demarcation between adequate and inadequate ventilation.
3. That sufficient air should be circulated in a well-ventilated mine to keep the temperature below 85° F., and, if saturated, below 80° F.; but it is not advisable to prescribe a wet bulb temperature.
4. That in deep mines an adequate amount of ventilation may only be maintained by the use of fans or blowers, in conjunction with downcast and upcast air shafts, or compartments in shafts of sufficient sectional area.
5. That in mines with such air shafts and airways of sufficient sectional area, exhaust centrifugal fans are preferable to blowers; but when the conditions are otherwise, and mines are worked from single outlet (shaft or drive), and when air-boxes or pipes are used in lieu of a second air-way, a blower is more efficient, as it may be worked against a greater water gauge, and it delivers fresh air at the face of operations."—FRANK REED.—*Mining and Engineering Review*, March 6, 1911, p. 252. (A. R.)

FAULTS IN MINES. "Mr. W. Lindgren, one of the leading American mining engineers, makes the following statement as to faults in veins: 'In considering the probable permanency of a given vein, its general character must be taken into consideration. Continuous, well-defined outcrops and wide bodies of quartz are in general, good indications of maintenance in depth, as is also any evidence of strong faulting and movement. A fissure which can be definitely proved to extend only a short distance will, in all probability, be found to be correspondingly limited in depth.' As volcanic agency has so much to do with lode formations it would be surprising to find them undisturbed, and in all the great mines of the world, the lodes, particularly quartz lodes, are both uneven in width, in value and in continuity. The banket beds of the Rand are more uniform than ordinary quartz veins, but even these are broken up by earth movements, by intrusive dykes and faults. It is much more so in an ordinary quartz vein, as, in addition to earth movements, they have been subjected to violent volcanic action—the action that makes mountains, in which most of the fissure lodes occur.

The higher values occur in chutes as in the Mysore mines, or in lenses as in the Kalgoorlie mines. The Kalgurli, Talisman and Camp Bird Mines may be quoted as instances of these massive lenses of rich ore, as was also the Lake View in its earlier days. Even in a great mother lode extending for many miles one section will contain a run of good ore, whilst another will be barren, then another run of good ore may be found, and so on. Nature does not act on uniform lines, but where the lateral extent of the lode outcrop is extensive and well defined within good walls the general rule is for the lode to go down strong, but subject to poor and rich zones."—*South African Mining Review*, March, 1911, p. 45. (A. R.)

COST OF HAND AND MACHINE STOPING AT THE WIT DEEP. "Of the rock broken in stopes during the year 74% was from hammer stopes, the cost per ton being within one penny of that in the machine stopes. On the fathom basis, the cost of hand stoping was only 53s. 7d., compared with 302s. 7d by machines."—*The South African Mining Review*, March, 1911, p. 59. (A. R.)

TIN MINING IN THE TRANSVAAL. "The output of tin has increased enormously since last year, some £306,100 worth of the metal being produced since June of last year. The tin bearing pipes on Zaaiplaats and the adjoining mine are mainly responsible for the large output, but the Rooiberg quartzite workings contributed nearly £60,000 worth of metal to the output.

Potgietersrust Tin Fields.—(*a*) *Zaaiplaats Tin Mine.*—The developments on this mine during the year have been so satisfactory as to necessitate an entire recasting of the popular idea that the peculiar pipes or lenses in which the tin occurs are of a local and temporary nature and can only be trusted so far as they are opened. Since the present management took over this property two years ago, not one pipe has been lost. No. 6 pipe, which was the second largest ore bearer on the property, after continuing to 500 ft. in a rich and comparatively regular manner, turned horizontally, and for a distance of 50 ft. the tin and pipe minerals were diffused in the country in a manner that undoubtedly would suggest to everyone the termination of the pipe. At the end of 50 ft. the tin entirely ceased, but certain alteration minerals and "gas stains" were followed for another 100 ft., when a definite jointing or fault plane in the country was encountered. Coming up along this plain more "gas stains" were found, and these were followed down for 100 ft., when the pipe suddenly reformed, and the end of the work, which is altogether 746 ft. from the surface, is now in a pipe of cassiterite 13 ft. × 25 ft. in extent, after having been for nearly 200 ft. absolutely devoid of tin and the pipe practically lost in the country rock.

Whatever theories of lateral secretion from the surrounding granite may be evolved in laboratories by consulting geologists, there is no doubt but that the mining evidence all points to the conclusion that the tin in these pipes comes up from below, following lines of weakness, and where none existed making vents for itself through what now is massive granite.

The workings in No. 5 pipe, into which Nos. 2, 3, and 4 have consolidated, are now 750 ft. in incline depth, and the face is 18 ft × 20 ft., and as rich and satisfactory as ever. Measured along its sinuosities this pipe has been worked for over 1,000 ft.

In mining out the impregnated zone on the surface at No. 13, it was found to consolidate into a pipe. This pipe, 40 ft. from the surface, measured in cross section 95 × 25 ft. It has now been followed by a shaft to 130 ft., and is there richer than above and of the same section. Roughly, this chute appears to average in richness between a minimum of 7% and a maximum of 25% tin. It has been supposed in the past that the working of one of these lenses was a completely exhaustive process, developing nothing and leaving only an empty and worthless hole behind. The developments in all the lenses now prove this idea to be false. Following down a lense is similar to following down a river from one of its sources, or following down a tree from one of its extreme twigs. Fresh tributaries are constantly joining or being joined, and it is impossible to tell whether the twig or tributary down which one has come is the main branch or only one of the smallest. It is, however, fair to presume that in these branches, from when they join the worked pipe to the surface, there will be found on an average approximately as much tin as has been found in the worked pipe. In nearly all the worked pipes large branches have come in, and these have been worked for a few feet and then left in reserve. What the extent of these reserves is, it is of course impossible to state; but, on the above presumption, this amount will certainly not be less and may be several times greater than the quantity already extracted, in which case the life of the mine is assured for several years, even were a fault to entirely cut off the further extension of the pipes in depth, of which, of course, there is not the least suspicion.

There is distinct evidence that not only do the pipes in one group come together in depth, but that also the groups of pipes are converging on each other, and it is probable that at no great depth the point to which they all converge will be reached. This point appears to be in the neighbourhood of No. 6, on the Government leased block, and about 1,000 ft. below the surface; but what happens at that point it is impossible at present to form even the vaguest theory.

(*b*) *Groenfontein.*—On Groenfontein and on the Government lease, the output of tin has gone on throughout the year in the most satisfactory manner, though the developments have not been so interesting or enlightening as on the Zaaiplaats Tin Mines. It seems that Zaaiplaats pipes are at the lowest geological horizon of any pipes struck so far—*i.e.*, they outcrop nearer the base of the granite than do the others, and that the nearer the base of the granite the more defined are the pipes, which appear to split up and to become more and more diffused as they get further from their point of origin, much as the main stream of a great river loses itself in an intricacy of small channels in its delta.

If the sources of the tin on the Government lease and on the pegmatite area on Groenfontein are ever to be traced, much more enterprise and courage will have to be shown in following down indications other than actual tin. Except at the Zaaiplaats Tin Mines, very little enterprise has yet been shown in anything else but extracting obvious cassiterite. Where this has stopped, the pipe has been abandoned as played out, even where a little observation and courage would have found obvious indications and traces showing the direction along which the mineralisation that produced the tin had come.

(*c*) *Waterval.*—The occurrence of tin in the felsites on this farm has now been followed to a depth of some 140 ft. on an incline of 45 degrees. The tin seems to occur as an irregular impregnation with no definition, breaking irregularly through the felsites, but not having established a definite enough course to be styled a pipe. The impregnated section is about 40 ft. in diameter, and though not usually carrying more than 1 to 2% of tin, occasionally becomes very rich in patches. The granites on this farm are not deep, and it is possible that when this tin course reaches the granite, it may be found emanating from the regular pipe, but at present we have no experience to guide us in forming any opinion as to the prospects of this sort of deposit.

(*d*) *General.*—Speaking generally of the whole Potgietersrust tinfields, when these were first discovered we were faced with an entirely new set of conditions, of which we had no knowledge or experience, and it was wise to treat the deposits as of very doubtful extent and great uncertainty. Every years' work has, however, gone to prove that these pipes are more and more trustworthy and permanent, and far greater courage is justified in their treatment to-day than would have been the case three years ago, and it is very much to be hoped that the larger companies interested in the tinfield will see this and will spend money boldly in attempting to follow down the diffused deposits of tin in the felsites and upper granites to their sources in the lower horizon.

Rooiberg Mineral Development Company, Limited.—There are prospects that this property will be a large producer in the near future, the ore reserves having been largely increased by constant discoveries of extra fillings or zones of payable ore running off the main lodes or fissures, and only visible when stoping operations are in progress. The tin-bearing fissures appear strong and permanent, carrying payable values over large areas. The lack of development at depth is being remedied by a new permanent main shaft already down some considerable distance, and prospecting in the vicinity of the main workings shows that other payable deposits of tin exist on the property.

Weynek.—The original Weynek company has practically ceased operations, a little prospecting being carried on below the first level. There is no probability of the mine re-entering the producing stage at present. Both ancients and moderns have picked out all the payable ore near the surface, and the lowest levels are in well-defined but poor lode matter carrying a little tin and copper. On the adjoining area, the owners of the farm have done a large amount of prospecting work on a low grade ore-body, showing payable values near the surface, but poor and broken up at the lower level."—U. P. SWINBURNE. Mines Department Report. *London Mining Journal*, April 15, 1911, p. 400. (A. R.)

MISCELLANEOUS.

THE ORIGIN AND ACCUMULATION OF OIL.—"*General Considerations.*"—A problem of special importance is the circulation of oil from its source and its mode of accumulation in porous rocks. The matter is being investigated by laboratory method by various scientists. The circulation is accomplished by agents such as capillarity, gravity, and gas or rock pressure. The accumulation of oil requires a porous reservoir with an impervious cover or roof. Certain features of geological structure and conditions of water saturation are important factors. The circulation, though at present largely a matter of supposition, must be affected by the physical properties and relations of oil, salt water, and the rocks in which they occur. One of the potent forces is doubtless capillarity, since both the shales and the sands are porous formations.

Shales and sandstones are porous formations containing infinite numbers of minute spaces capable of holding liquid. The spaces or pores may be likened to capillary tubes and may be assumed under proper conditions to promote capillary action. Capillarity was perhaps effective upon the included water of shales long before the distillation of oil began in them. The action extended to the oil which began to originate and find its way into the pores of the shale. Its production was exceedingly minute, yet it was acted upon by capillarity. The relation of the specific gravity of oil and water caused the oil to rise to the top of the water and thus into the sandstones. It is assumed that this action continued as long as distillation took place, until eventually the oil had left the shales to a large degree and had accumulated in the sandstones. The action may have been further aided by various compressions of the formations and other unknown physical phenomena until the shales had given up most of their oil to adjoining porous sandstones.

It is apparent that petroleum is greatly influenced by the presence of water and it is a fact that there is abundant water in the Illinois oil-sands. The relation of the specific gravities of oil and water undoubtedly played an important part in the circulation of oil. The specific gravity of a substance may be defined as the relation between its weight and the weight of a like volume of pure water. The specific gravity of petroleum is less than one, or in other words, oil is lighter than water. If both are present the oil rests upon the surface of the water and is to that extent controlled by the latter. If oil and water are not associated, the petroleum moves downward along bedding planes and through coarse, porous strata under force of gravity. In such a case it may occur in pores at the bottom of a syncline.

A third theoretical agent of the circulation of oil from its source of distillation to its present position is perhaps that of gas pressure or "rock pressure." This pressure is always noticeable when a new oil or gas area is opened up. The oil generally rises far up into the casing of the new well and often above its mouth. If gas is present and the casing is closed so that the product cannot escape into the air a pressure is developed inside the pipe. This may accumulate instantly and thus indicate a very porous reservoir beneath, or it may take considerable time to gather and thus show a less porous one. The two conditions have often occured in the same locality and yet the same pressures were eventually secured. It is believed by the writer that rock pressure is promoted by expansion of highly compressed liquids and gases in passing from porous beds into a cavity. It is thought that gas pressure may help to promote movement of oil through the containing rocks.

The Porous Stratum.—Petroleum was valueless as a commercial product when it was originally formed, because its diffusion was so complete that a bore into the containing rock could scarcely have obtained a showing of oil. Its accumulation in pools of commercial value first demands more porous beds than the shales in which it is supposed to have originated. The strata of sand interlain with the shales are suitable reservoirs because in most cases they are much more porous than the compact shales. Exceptionally, the sands themselves contain portions which are extremely compact and impervious. These non-porous areas may act as retaining covers and effect the concentration of underlying oil where structure is favourable. They may be extensive enough to separate adjoining pools, or they may be small enough in extent to cause mere local "dry spots" in the midst of very productive territory, in which the sands are otherwise highly porous. The presence of small streaks of shale within the sandstones is frequent in Illinois formations. Often two or three averaging 5 ft. to 10 ft. in thickness may occur in a thickness of 50 ft. to 80 ft. of sand. The driller terms these, "breaks." The sand and the "break" merge into one another in most cases and oil is not often found where sand and shale are thus mixed.

Impervious Cover.—An important requirement or the accumulation of oil and gas is an impervious cover, or retaining roof, which will hold the oil and gas captive in the porous stratum. In Illinois there is almost invariably a cover of hard, compact, shale over the sand. This is particularly true of the sands in the Pennsylvania formations. The producing sands in the Mississippian formation are overlain in some instances by limestone. The impervious covers have doubtless caused the retention of the oil in the sand during the periods of earth movements which caused structural folds in the rocks. If an oil pool did not have an impervious cover between it and the surface, the lighter portions of the oil would long since have volatilised and passed off as natural gas, while only the heavy oil or asphalt-like residue would remain. Where a thin cover lies over a productive oil sand some of the lighter portions of the petroleum have escaped and heavy, lubricating oil is generally found. This is of low gravity and consequently of low grade, and generally serves as fuel oil. The abundance of shales within the 'coal measures' and the upper Mississippian rocks of Illinois have prevented an extensive volatilisation and consequently the oils are of good grade, averaging about 30 in gravity

Geological Structure.—Another very important necessity for the accumulation of oil and gas in pools, is the presence of structural features in the rocks. The sedimentary strata were deposited under water horizontally, or practically so, and the natural distillation of oil probably took place primarily while the beds were in that position. Subsequent disturbances took place causing the strata to be folded, forming as it were, an arch or dome in one case and a corresponding trough or basin in the other. The arches are known as anticlines while the depressions are called synclines. When these undulations took place, the water, petroleum, and gas within the sand formations were forced to move and distribute themselves according to the laws of gravitation and hence according to their specific gravities. The water was the heaviest of the three fluids, and, therefore, sought the synclines as far as possible, depending, of course, upon the porosity of the sands. Its tendency was to displace the oil and gas, forcing the oil to float on the water and the gas to rise still higher. The oil was enabled to rise as far as the water extended up the slopes of the syncline, while the gas was able to free itself from the fluids and rise to the highest place in the porous bed, usually the crests of the anticlines.

The earth disturbances effecting the changes in the position of the strata may be responsible also for minor irregularities which occur on the anticlines and synclines themselves. The surface of an oil sand on the anticline may be pitted or undulating. This condition may affect an extensive area or only a few acres of ground. The general accumulation of oil and gas is governed by the anticline proper, covering many miles, and the segregation of pools may possibly be caused by smaller folds on the large one. Coupled with this intricate system of synclines and arches on the parent fold, there is variation in the porosity of the sands; the two conditions greatly affect the distribution of oil and gas. It is readily recognised that either factor may, locally, explain the presence of dry holes within productive territory. Some question has arisen as to whether these minor arches are true anticlinals of deformational character or whether they represent merely original thickening and thinning of particular beds or, again, whether they result from unequal settling during the consolidation of the sediments. Locally, any or all of these factors may account for the conditions.

Another important type of geologic structure, in which an accumulation often occurs, is the "terrace" or flattened area upon the flanks of a syncline or anticline. The terrace, strictly speaking, is an interruption in the uniform dip of the sides of a basin, giving rise to an approximately horizontal plane. Such terraces are to be found upon the sides of the structural basin in southern and central Illinois. A segregation of oil takes place upon a favourable terrace much in the same manner as in the anticlines and the synclines. The water of the basin enables the oil to rise to the terrace, where it may be trapped by friction. But the oil, originally in the sloping sand above the terrace, may migrate further up the general incline so as to float on the water surface. The gas follows its usual course in freeing itself from the oil and accumulates in the terrace head or continues up the general dip to the adjacent anticline or to some impervious barrier.

Water Saturation.—One of the most important factors, if not the greatest, in the concentration of oil in raised structure, is the presence or absence of water in the oil-bearing stratum. W. T. Griswold offers some very interesting observations upon this subject with reference to the Appalachian region. The theories are more or less applicable to the Illinois rocks, inasmuch as they are of similar age and character. His conclusions are as follows :

'In *dry* rocks the principal points of accumulation of oil will be at or near the bottom of the syncline or at the lowest point of the porous medium, or at any point where the slope of the rock is not sufficient to overcome the friction, such as structural terraces or benches. In porous rocks, *completely saturated*, the accumulation of both oil and gas will be in the anticlines or along level portions of the structure. Where the area of porous rocks is limited, the accumulation will occur at the highest point of the porous stratum ; and where areas of impervious rocks exist in a generally porous stratum the accumulation will take place below such impervious stop, which is really the top limit of the porous rock. In porous rocks that are only *partly filled* with water the oil accumulates at the upper limit of the saturated area. This limit of saturation traces a level line around the sides of each structural basin, but the height of this line may vary greatly in adjacent basins and in different sands of the same basin.

'Partial saturation is the condition most generally found, in which case accumulations of oil may occur anywhere with reference to the geologic structure. It is most likely, however, to occur upon terraces or levels, as these places are favourable to accumulation in both dry and saturated rocks.

'Under all conditions the most probable locations for the accumulation of gas are on the crests of anticlines. Small folds along the side of a syncline may hold a supply of gas, or the rocks may be so dense that gas may not travel to the anticline, but will remain in volume close to the oil.'

The application of the above observations to the Illinois wild-cat areas can only be made in a limited manner at the present time, owing to the lack of data on the water saturation in such wells as have been drilled to date. It is hoped that in the future the operators in Illinois will note with as much exactness as possible the wet condition of the sands they encounter. It will then be possible for the geologist or engineer to offer better suggestions as to the probable conditions in prospective oil areas."—RAYMOND S. BLATCHLEY.—*Illinois Geological Survey's Year Book.*—*Mining World*, March 4, 1911, p. 499. (A. R.)

THE GREATEST GOLD MINE OF FRANCE.—That France possesses any gold mine, great or otherwise, is a source of surprise to many of us, more accustomed to think of France as the producer of a golden vintage rather than of golden ore. Yet there are several gold mines in France, and one of them is not only great but also of historic antiquity. During the period of the Roman occupation of Gaul, some veins of gold-bearing quartz in what is now La Vendée were worked on rather an extensive scale to a depth of nearly a 100 ft.

The La Bellière mine is situated in La Vendée and is equally distant from the three towns of Nantes, Cholet, and Angers. It is reached by taking the express train to Angers, a local train to La Possoniere and the Anjou department railway to Montrevault, which is about a mile from the mine at St. Pierre-Montlimart. The ore deposit consists of quartz veins in schist and may be considered in two parts. At the western end of the property there are three veins, the first and third being about three metres thick and the second about two. The second vein joins with the first towards the east, and about the middle of the property all three pinch out. Further to the east are two great lenses of quartz, 15 to 18 m. wide at the widest place, with lengths of 100 and 150 m. respectively. The ore has been exposed by four sets of levels at a depth of 30, 50, 80, and 110 metres. The engineer in charge told me that the lenses were as thick on the 110 m. level as at 50 m. This is satisfactory, as indicating the size of the ore bodies. Approximately three-fourths of the present daily production is obtained from the lenses and one-fourth from the workings on the vein proper.

The ore consists of quartz, containing 15·16 gm. gold per ton (one gm. is equal to 0·643 dwt.), a little silver, partly alloyed with the gold and partly in some unidentified silver mineral, enough arsenopyrite to make the ore average ½% arsenic, and a little chalcopyrite, galena, and pyrite. The general process is to stamp the ore to pass through 30-mesh screen and then over amalgamating plates, 43½% of the gold content being arrested at this stage. The pulp is then classified into sand and slime, the sand being concentrated on Wilfley tables, yielding 4½% of concentrate. All three products are then cyanided, and a further extraction of 47% obtained, making a total of 90½%. The concentrate is roasted in Maletra furnaces for the production of white arsenic. During 1909 68,880 metric tons were milled at a cost, for mining and milling, of 25·10 f. per ton; 953,701 gm. of gold were recovered, or 46·15 f. per ton.

The mine is opened up by three two-compartment shafts arranged in a straight line 250 m. apart. The relation of the shafts to the ore body is shown. The mill is near the westernmost shaft. Besides these three shafts there are some minor adits and shafts for admitting materials for filling the stopes. Hoisting is done in cages by geared electric hoists.

The method of stoping the lenses is as follows: A drift is driven along each wall and a third through the centre of the lens. Rises are then made at intervals of 15 m. and extended to within a safe distance of the level above; the rises along the walls are carried through and serve to bring down the filling for the stopes. Beginning at a safe distance above the bottom level, slices two metres wide and two metres high are taken out and the broken ore moved down the chutes. The roof is supported by a little light timbering, as required, and as fast as the slices are made the space is filled. Part of the material used for this purpose is the tailing from the cyanide plant, but as the men object to it, claiming that the cyanide solution contained in the tailing makes the air bad, other surface waste is used as far as possible.

The filling is done on day-wages, but the mining is done by contract at the rate 5·75 f. per cubic metre, out of which the miners have to pay for the powder used, but have everything else furnished them. They have also to set any timber required for their own safety. Good men make about 5·50 f. per shift. Development work is also done by contract. Most of the underground work is done in two shifts of eight hours, but the tramming and hoisting is done in three 8-hour shifts, as the reserve capacity of the bins at the mill is rather small. In shaft-sinking 6-hour shifts are employed.

The drilling is all done by hand. An attempt was made to introduce stoping-drills, but the men objected to them and used to stuff bread-crusts into the valve-chambers in the hope of breaking them. I did not see the drill used here, but if they were like those employed at another near-by prospect, the men were not without reasonable excuse. It may be remarked also that the air was used at a low pressure, too low for any drill to do good work.

The ore from all three shafts goes to scales to be weighed and then to a large jaw-crusher. The product, about 3½ in. ring, goes to a short belt-conveyer, where two men remove any particles of wood or iron. The belt delivers it to two smaller crushers and these feed two other belts, which convey it to the top of the stamp bins. There are 70 stamps, 40 of 1000 lb. and 30 of 1500 lb. each. The 70 stamps crush 280 tons per day, indicating an average duty of 4 tons. The pulp passes over amalgamating plates 8 ft. long, and then over a couple of shaking amalgamating tables. In this way 43½% of the gold is caught. The pulp then passes to Dorr classifiers and the resulting slime goes to large settling-vats where it is thickened and then cyanided by agitation. The sand goes to 8 Wilfley tables. The resulting concentrate (amounting to about 4½ to 5% of the ore and containing 10% arsenic) is reduced in fine grinding pans and then agitated with strong cyanide solution for 15 hours. This gives a fairly complete extraction of the gold with a cyanide consumption of 2 lb. per ton, which is properly regarded as a notable improvement over the former consumption of 8 lb. per ton.

The concentrate after cyanidation is roasted in two Maletra furnaces, the crude arsenic being caught in dust chambers and then leached with hot water, to remove the soluble portion and re-distilled. The re-distilled fume is caught in a series of three condensers. The product from the first of these is ready for sale; that from the other two is leached again, and distilled a third time. The residue contains 8 to 10% copper, 5 to 6% lead, and about 10 gm. gold per ton. Formerly, when the gold content was higher, this residue was sold in Wales, but now it is discarded. It would certainly seem as though it would pay to install a small reverberatory furnace to treat this material by smelting, after a sufficient quantity had been allowed to accumulate. About 14 tons of concentrate is treated daily, and 1 ton of arsenic is extracted. There can be but little profit in the arsenic, as it sells for 300 f. per ton, and less than two-thirds of the output found a market last year.

The sand from both sets of Wilfley tables and the middling from the second set are cyanided by percolation in vats. It should be remarked that the mill possesses three tube-mills, which are to be employed to grind the sand from the Dorr classifiers, but these are not now in use, as the present slime-plant is taxed to its full capacity. The sand now goes first to de-watering tanks, where it is drained to make it more porous, and then conducted to a screened set of vats where it is leached by percolation. The solution contains 0·16 lb. KCy per ton, and is passed at the rate of 30 tons per day for 15 days. The percolation-vats are of modern type and are provided with Butters distributers and Blaisdell excavators. The tailing is handled in two ways. That from the slime-plant is taken by a skip to the top of a dump to the west of the mill. The sand is handled by a Blaisdell tailing-stacker on the eastern side of the mill. The waste cyanide solution cannot be allowed to escape into the adjacent stream on account of the fish, so it is allowed to stand in a large vat, permitting the air to destroy the cyanide before it is finally pumped to the top of the tailing pile and so allowed to seep gradually into the stream.

The precipitation of the gold is done in the usual way in zinc boxes, the only unusual feature being the ceremonious manner in which the work is done. There is only one entrance to the room, which is protected by double locks, the floor inside is carpeted, and the workmen wear slippers, while the boxes are neatly covered like gigantic coffins.

About 250 men are employed on the surface, only a few of them being engaged in the mill. These men work for 12 hours and their wages average 4 f. per day. They are local men who have been trained so as to make satisfactory workmen.

A commendable feature of the entire equipment is that the company has refrained from mechanical experiments, perhaps regarding it as enough of an experiment to begin work in a country where gold mines were (with the exception of La Lucette) previously unknown, without adding anything in the way of untried machinery and processes. Hence the machinery is of standard design, most of it made by Fraser & Chalmers. The electrical power plant and the system for supplying water are especially worthy of note.

The treatment of the ore does not offer any special problems, as the amount of arsenical concentrate is not small, nor is its gold content especially high, being only about three times that of the average ore. Naturally, there is a good deal of cyanide consumed in treating this material, but recently the manager has reduced the consumption; and now that it is one-fourth what it was in the preceding year he has reason to congratulate himself. The question of disposing of the tailing presents a minor problem, as the space now available for this purpose is not sufficient. This difficulty has been solved, in part, by the increased use of tailing for filling.

At the end of the last financial year the company signalized its development as a successful enterprise by the payment of interest on its shares at the rate of 6% for the five years during which the mine has been in operation. This was in addition to setting aside the legal reserve and providing for amortization at the rate of 25% on the shafts and underground working, and 15% on the surface installation. The amount of ore available in the underground workings is 256,490 tons, a gain of about 25,000 tons over the preceding year. With quartz veins in schist it is usually desirable to keep development work well ahead of stoping, on account of the variable nature of such deposit.—Thomas T. Read.—*The Mining Magazine*, Vol ix. March, 1911, p. 209. (J. E. T.)

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Abstracts of Patent Applications.

(C.) 267/10. David Gilmour. Improvements relating to the linings of tube mills. 8.6.10.

This application relates to tube mill liners and consists of providing means for securing the renewable wearing end plates of the mill. These means consist of flat bearing surfaces or brackets attached to or forming part of the end cover, to which the wearing plates can be bolted, thereby obviating leakage through the bolt holes that occurs in ordinary practice.

(C.) 318/10. Goven Miller Meikle (1), David Hamilton Evans (2). Improvements in slime separators. 5.7.10.

This application relates to slime separators, and describes a device consisting of a cylindrical vessel with cone bottom, within which is arranged a vertical pipe, which is either tapered or flared at the lower end. Immediately below and extending into this pipe is placed a cone shaped member so as to allow of an annular space between the two. The upper end of this cone is connected to a pipe leading through the wall of the outer cylinder and forms the exit for the slime. This discharge pipe is provided with float valve regulating the discharge. A water inlet pipe is also provided to supply water at the cone bottom in a tangential direction to assist the separation.

(C.) 615/10. Peter Nerman Nissen. Improvements in stamp mills. 13.12.10.

This application is for a stamp head having a lesser diameter at its lower end and a restricted neck in the mortar box so as to prevent the splash of pulp over the top of the mortar. There is also a claim for forming the neck of a circular mortar in two parts, one of which is detachable in order to facilitate the removal and replacement of the crushing members. A further claim is for the means of securing a flexible screen frame to a mortar by means of a key at each end which stretches and clamps the screen frame fast.

(C.) 151/11. Emanuel Rosenberg (1), John Sedgwick Peck (2). Improvements in braking induction motors. 10.3.11.

This invention relates to that method of braking induction motors in which a continuous current, in place of the normal alternating current, is supplied to the stator or primary circuit, whilst at the same time resistances in the secondary rotor circuit are automatically controlled so as to give a satisfactory braking torque.

The means of controlling the amount of resistance described in the specification are:—

(1) By a switch actuated by a centrifugal governor driven by the rotor.

(2) By the variation in voltage and frequency of transformers in the secondary circuit. These variations control the speed of a small motor actuating the governor or otherwise controlling the resistance.

(3) By an electrical control instead of the centrifugal governor, such electrical control again depending upon the voltage and frequency of the transformers in the secondary circuit and being effected through an instrument resembling a frequency indicator.

(4) By the centrifugal action of a liquid in which electrodes are immersed to a greater or lesser extent according to the speed.

(5) By a sluice gate fixing the level of the liquid, such sluice gate being operated by a solenoid actuated by the secondary transformers above referred to.

(6) By a sluice gate as above, but operated by a centrifugal governor driven by the motor shaft.

(C.) 172/11. Edmund Seal Donisthorpe. Improvements in or relating to apparatus for synchronising the working of machinery. 24.3.11.

The broad claim No. 1 of the invention is an apparatus for synchronising the working of two machines (or motors) consisting of a jaw clutch where the one-half is connected with the slower but stronger machine, and the other half with the quicker but weaker machine. The latter half of the clutch will therefore always be in proper contact with the slower running half of the clutch. Claims 2 to 7 deal more particularly with the synchronising of kinematograph and grammophonic apparatus.

(C.) 185/11. Charles Francois (1), Alfred Chapitel (2). Improvements in breaking machines. 31.3.11.

This application refers to a crushing machine of hammers hinged on a rapidly revolving shaft, which breaks the ore by impact therewith. The applicant claims novelty in the construction of such a machine by making the hammers strike the falling material, which impinges again on anvils placed above them. The oversize is returned to the hammers by means of a slowly revolving screen having vanes to elevate the coarse particles which do not pass the screen slots.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 244/11. Hugh McLean. Improvements applicable to the cam shafts of stamp mills. 29.4.11

(P.) 246/11. Robert Burns (1), Norman Burns (2). An improved rotary engine or internal combustion turbine. 2.5.11.

(P.) 247/11. Francis Ernest Dunnett. Improvements in composition or paint. 2.5.11.

(P.) 248/11. Morris Frank Sherwell. (1), Cyril Francis Tonkin (2). Improvements in gaseous fluid actuated rock drills. 2.5.11.

(P.) 249/11. Alexander Purser. A new and improved automatic adjusting cock for pipes and the like. 3.5.11.

(C.) 250/11. Wilhelm Gunther. Processes and apparatus for extracting and lixiviating ores metallic residues and the like. 3.5.11.

(P.) 251/11. Johannes Davidtz. Improvements in metallic packing for pistons and the like. 4.5.11.

(P.) 252/11. William Ross. Improvements in launders, troughs and the like for conveying liquids and mixtures to solids and liquids. 6.5.11.

(P.) 253/11. Elisha Tippett. Improvements relating to rock drilling machines for removing the cuttings from the holes formed therewith. 4.5.11.

(C.) 255/11. Cecil Watson. Improvements in buffings and draft gear for railway and other rolling stock. 5.5.11.

(C.) 256/11. Richard Bernard Rinne. Improvements in apparatus for automatically signalling to the driver of a hoisting or hauling engine the fact that a skip, cage or like vehicle has left the track or guiding means. 6.5.11.

(C.) 257/11. John Hedley Clarke. Improved means for attaching elevator buckets to belts. 6.5.11.

(P.) 259/11. Percy Alexander Robertson (1), Noel Yeo Robertson (2). Improvements in tube mill feeding means. 8.5.11.

(P.) 260/11. Ernest Vivian Pearce. An improved process for extracting the precious and/or other valuable metals from the products and/or bye products of milling and smelting operations. 8.5.11.

(P.) 261/11. William Ross. Improved system of overflow launders for tanks used in the process of gold recovery or extraction from ores. 8.5.11.

(P.) 262/11. Samuel Raybould. Improvements in means for diverting the pulp issuing from the mortar box. 9.5.11.

(C.) 263/11. Hans Thorvald Hansen. An improved method for fixing springs to vehicles and disselbooms and attachments for same. 9.5.11.

(P.) 265/11. James Hamilton. Apparatus for elevating or conveying liquids and semi liquids. 9.5.11.

(C.) 266/11. Michael Ferenczy. The nitre-oxidation process for extraction of gold from refractory ores or sands. 10.5.11.

(P.) 268/11. Thomas Henry Bradbury. Improvements in electric resistance furnaces. 11.5.11.

(P.) 269/11. James Lester Clark. Improvements in pipes or launders particularly suitable for conveying abrasive or corrosive material. 11.5.11.

(C.) 270/11. Wilbur Alson Hendryx. Improvements in and relating to apparatus for separating solids and liquids. 12.5.11.

(C.) 271/11. Edward May Munro (1), The Railless Electric Traction Co. Ltd., (2). Improvements in or relating to trolley poles for electric traction systems. 12.5.11.

(P.) 272/11. Steven William Smith (1), William Joss (2), John Duff Innes (3), Charles Innes (4). Improved cock for the regulation of air or other elastic fluids or gases. 12.5.11.

(P.) 273/11. James Snodgrass. Electric furnace for heating drills or blanks. 12.5.11.

(P.) 274/11. Gabriel Christan Erasmus Keet. Improved safety device for motor cars, locomotives, or similar vehicles. 12.5.11.

(C.) 275/11. Thomas Cooper. Improvements in roller or ball bearings for shafts, axles and the like. 12.5.11.

(P.) 276/11. John Murdoch Skeely (1), Rollo Bowman Ballantyne (2). An improved process for the purification of boiler feed waters. 13.5.11.

(C.) 277/11. Charles Alfred Jeffreys. Improvements in attachments for rock drills for removing the cuttings from the boreholes. 13.5.11.

(C.) 278/11. Alfred George Bettington. An improved fence. 13.5.11.

(C.) 279/11. Charles Hansen. An improved arrangement or means for damming or stopping the flow of underground streams and rivers in mine shafts and workings. 15.5.11

(P). 281/11. Peter Norman Nissen. Improvements in driving means for the ore feeders of stamp mills. 18.5.11.

(P.) 282/11. Henry Jeremiah Smith. A new and improved device or tool for straining wires and the like. 18.5.11.

(C.) 283/11. William James Barnett (1) and John Mitchell Lafayette Henry (2). Improvements in bolts and nuts. 18.5.11.

(C.) 285/11. Edward May Munro (1) and The Railless Electric Traction Co., Ltd. (2). Improvements in or relating to current collectors for electrically propelled vehicles. 19.5.11.

(C.) 286/11. Lewis Condict Bayles. Improvements in valve motions for rock drills. 19.5.11.

(C.) 287/11. Lewis Condict Bayles. Improvements in valve motions for rock drills. 19.5.11.

(C.) 289/11. Henry Rowe Rowe. Improvements in the construction of sewage purifying tanks. 20.5.11.

(C.) 290/11. John Cunningham Cunningham. Improvements in apparatus for cooling water by evaporation. 23.5.11.

(C.) 291/11. John Sorley. An improved composition for the destruction of insects. 23.5.11.

(P.) 295/11. William Thomas Hulley (1) and Frederick William Lippiatt (2). Automatic motor machine. 23.5.11.

(P.) George Frederick Ferns (1) and William George Jackson (2). Roof guttering support. 23.5.11.

(P.) 297/11. Carl Rugen Fredericks. Improvements in spark arrestors applicable for locomotives, traction engines, stationary engines and the like. 26.5.11.

(C.) 299/11. Alexander Hendry (1) and Malcolm Allan Hendry (2). Improvements in and relating to machinery driving belts. 26.5.11.

(C.) 300/11. Heinrich Ostwald. Improvements in ball mills. 26.5.11.

(C.) 301/11. Charles Eber Baker. Improvements in the treatment of ores. 26.5.11.

(C.) 303/11. Edward Baynes Badcock. Improvements in or connected with internal combustion liquid pumps. 26.5.11.

(P.) 304/11. Douglas William Stacey. A new process for the extraction of gold and other metals from slimes, by cyanide or other chemicals. 29.5.11.

(P.) 305/11. William George. Improved rolling digger. 29.5.11.

(P.) 306/11. Frederick Walter Cindel. Tube Mill Feeder. 29.5.11.

(P.) 307/11. Jacob Casper Smit. Syphon water elevator. 30.5.11.

(C.) 309/11. Georg Lene (1), the Firm of Feodor Siegel (2). Improved transporter. 2.6.11.

(C.) 310/11. Frank Franz (1), Eugene R. Day (2). Improvements in ore concentrators. 2.6.11.

(C.) 311/11. Frank Franz (1), Eugene R. Day (2). Decks for concentrating tables. 2.6.11.

(P.) 312/11. John George Volk. Improvements in means for feeding grinding material into tube and like mills. 3.6.11.

(P.) 313/11. John Davidson. Improvements in means for use in boring and re-boring the cylinders and valve chests of rock drills and the like. 3.6.11.

(P.) 314/11. Edmund James Sherwood. An invention for Transvaal artificial stone. 3.6.11.

(P.) 317/11. Edward Santon. Improvements in assay furnaces. 6.6.11.

(P.) 319/11. Brison Duncan (1), Francis Lockhart Duncan (2). Improvements in apparatus for depositing residues and the like on mine dumps or other depositing sites. 7.6.11.

(P.) 320/11. Ernest Alfred Wilderspin. Improvents in and relating to brazing cast iron and other metals. 8.6.11.

(P.) 324/11. William Joseph Holmes (1), William Albert Norman (2). Improvements in self locking nuts. 9.6.11.

(C.) 325/11. Francis Temple Page. Improvements in straining appliances. 9.6.11.

(C.) 326/11. Alexander Trifonoff (1), Daniel Gardner (2). Method of treating antimony and arsenic ores. 9.6.11.

(C.) 327/11. Nathaniel Greening. Improvements in wire fabrics. 9.6.11.

(P.) 328/11. Charles Coventry Indge. An automatic tally and office register. 9.6.11.

(C.) 329/11. Valentin Jancec. Improvements in apparatus for transporting, manipulating, and fixing props, timbers, or the like in stopes or other workings of mines and the like. 10.6.11.

(C.) 330/11. William Berthelson. Improvements applicable to acetylene head lamps for the use of miners and others. 13.6.11.

Changes of Address.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be notified of non-receipt of Journals and Notices at once.

Adams, A. R., *l/o* Langlaagte; Lancaster West G. M. Co., Ltd., P. O. Box 360, Krugersdorp.

Atkinson, C. E., *l/o* Shamva Mines; Bushtick Mines Ltd., Bushtick, Bulawayo.

Beaver, W., *l/o* Cleveland; Warren Hill Syndicate, Mine Office, Ariston P. O., via Klerksdorp.

Betty, W. K., *l/o* Johannesburg; 130 Fellows Road, Hampstead, London, N. W.

Cotter, A. J., *l/o* Boksburg; P. O. Box 3692, Johannesburg.

Dodds, J. B. K., *l/o* Cleveland; Nourse Mines, P. O. Box 32, Denver.

Fleming, J., *l/o* Florida; Kerry Road, Parkview.

Halford, J. E., *l/o* Schweizer Reneke; P. O. Box 2286, Johannesburg.

Hawley, T., *l/o* Salisbury; Eldorado Mine, Rhodesia.

Henderson, J. M. C., *l/o* Johannesburg; c/o J. A. L. Henderson Esq., 4, Bishopsgate St., Within, London E. C.

Holden, W. C., *l/o* Johannesburg; 14, Radnor Park Road, Folkestone, England.

Jack, W. D., *l/o* Cleveland; Warren Hill Syndicate, Ariston P. O. *via* Klerksdorp.

Jenkins, C. W. B., *l/o* Yambulla: c/o Miss Murray, Hillview, Turranurra, N. S. Wales, Australia.

Jennings, H., *l/o* Washington; c/o Smith & Perkins, Mills Buildings, 15, Broad Street, New York, U. S. A.

Kriekhans, Karl., *to* General Manager, Salida Mining Co., Ltd., Padang, Sumatra Dutch East Indies.

Leyson, L. T., *l/o* Benoni; c/o Standard Bank of S. A. Ltd., Commissioner St., Johannesburg.

Lloyd, J. J., *to* P. O. Box 80, Diego-Suarez.

Lucke, P. K., *l/o* U. S. A., c/o T. Yates Smith Esq., 28 Huntly Gardens, Glasgow, Scotland.

Matthews, E. L., *l/o* Johannesburg; New Modderfontein G. M. Co., P. O. Box 25, Benoni.

Minaards, W. I. K., *l/o* S. Rhodesia; Midas Deep, P. O. Box 8, Randfontein

Parsea, A. L., *l/o* Inyati; Etna Mine, Gatooma District, Rhodesia.

Rascher, L. H., *l/o* O.F.S.; Eldorado Mine, S. Rhodesia.

Reynolds, J. P., *l/o* Gatooma; Shamva Ilex G. M. Co., Abercorn, *via* Salisbury, Rhodesia.

Ross, H., *l/o* Krugersdorp; The Vogelstruis Estates and Gold Mines Ltd., P. O. Box 116, Roodepoort.

Wilmoth, L. J., *to* 21, George Street, East Melbourne, Victoria, Australia.

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